TRANSACTIONS

OF THE

AMERICAN INSTITUTE OF MINING AND METALLURGICAL ENGINEERS

(INCORPORATED)

VOL. LXIX

CONTAINING PAPERS AND DISCUSSIONS PRESENTED
AT THE MEETINGS HELD IN NEW YORK AND CANADA IN 1923.

NEW YORK, N. Y.
PUBLISHED BY THE INSTITUTE
AT THE OFFICE OF THE SECRETARY
29 WEST 39TH STREET
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THE MAPLE PRESS - YORK PA

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PREFACE

In this volume are the papers and discussions presented at the New York Meeting, February, 1923, and at the Canadian Meeting, August, 1923; also the Proceedings of these Meetings.

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Robinson, C. S., D, '13.
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Smyth, Henry L., D, '14–16.
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*Spilsbury, E. G., M, '85–7; V, '93–4: P, '96.
*Stanton, F. McM., M, '97–9.
*Stanton, John, V, '92–3.
*Stearns, I. A., V, '05–6.
Stearns, T. B., D, '16–18, '22–23.
*Stetefeldt, C. A., V, '85–6, '95–6.
Stone, G. C., SD, '12; D, '13–19; T, '13–18.
Stoughton, Bradley, S, E., '13–21.
Struthers, J., e, '03–5; s, '06–10; t, E, '06–12; D, '11; S, '11–12.
*Swoyer, J. H., V, '71.
*Symons, W. R., V, '71–2; M, '73–4.
Taylor, Samuel A., D, '15–20.
*Taylor, W. J., M, '90–2, '99–01.
Thacher, Arthur, D, '18–20.
Thayer, Benjamin B., V, '12–13; P, '14; D, '17–18.
*Thomas, David, P, '71. .
*Thomas, Samuel, V, '79–80.
*Thompson, Charles O., V, '81–2.
*Thurston, Robert H., V, 78–9.
*Valentine, M. D., M, '02–4.
Walcott, C. D., V, '04–5.
Walker, Arthur L., D, '11.
*Walker, W. R., D, '19.
Webster, William R., M, '95–7.
*Weeks, J. D., V, '86–7; M, '90–2: P, '95.
*Wellman, S. T., V, '83–4: M, '90–2.
*Williams, Gardner F., V, '11–12.
Williams, T. M., M, '71.
*Williamson, J. Pryor, T, '71–2.
*Winchell, H. V., M, '01–3; V, '09–10; P, '19.
Winslow, A., M, '99–01.
*Witherbee, Thos. F., M, '76–8.
Wood, Walter, C, '06–8.
Wrather, W. E., D, '21–23.
*Young, Edward L., C, '11–12; D, '13–14.

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*Deceased.

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1905.	Henri LeChatelier	Paris, France.
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1902.	Manuel Maria Contreras	1902	1884.	Theodor Richter	1898
1888.	A. Daubrée	1896	1899.	W. C. Roberts-Austen	1902
1906.	James Douglas	1918	1890.	Albert Serlo	1898
1884.	Thomas M. Drown	1904	1880.	C. William Siemens	1883
1890.	Moritz Gaetzschmann	1895	1909.	James M. Swank	1914
1873.	L. Gruner	1883	1872.	David Thomas	1882
1919.	Robert W. Hunt	1923	1902.	Dimitry Constantin Tschernoff	1921
1891.	Bruno Kerl	1905	1873.	Peter R. von Tunner	1897
1895.	Joseph Le Conte	1901	1885.	Hermann Wedding	1908
1891.	J. P. Lesley	1903	1910.	Tsunashiro Wada	1920
1899.	Floris Osmond	1912			

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[a] The purpose of this division is to obtain papers of high merit for publication by the Institute on the subjects of alloys, metallography, and founding.

[b] The purpose of this division is to obtain papers of high merit for publication by the Institute on production, transportation, refining and utilization of petroleum and its products.

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PROCEEDINGS

New York Meeting, 1923

The 127th meeting of the Institute was held in New York, Feb. 19 to 22, 1923. In addition to the usual large volume of technical matters under consideration, the meeting was noteworthy for the attention given to its business session; the convention of Local Section representatives also was full of interest. The total registration was 1201.

The annual business meeting on Feb. 20, President Dwight presiding, was attended by 125 members. Addresses supplementing their formal reports were made by the Secretary, who spoke enthusiastically of the work conducted by the technical committees; by John V. Davies, President of the United Engineering Society, who explained the functions of that organization and the relationship between it, the four Founder Societies, and the Engineering Foundation; by J. Parke Channing, a vice-president of the Federated American Engineering Societies, who recorded the more important accomplishments of that organization during the the last year; by the Treasurer, Charles F. Rand, who explained how the Institute's discouraging financial state of a few years ago had been converted into a very hopeful condition, with a sufficient surplus in sight to warrant the inauguration of a reserve fund for initiation fees, and to make possible the refunding of traveling expenses incurred by Local Section delegates to the Annual Meeting; and by the President, who stated his opinion as to the necessity and the desirability of publishing the Institute's monthly magazine.

On the report of the tellers, the following officers were declared elected:

E. P. Mathewson, *Director and President*—District 0
T. T. Brewster, *Director and Vice-president*—District 6
Frank M. Smith, *Director and Vice-president*—District 11
W. H. Aldridge, *Director*—District 0
G. H. Clevenger, *Director*—District 1
A. N. Diehl, *Director*—District 3
Reno H. Sales, *Director*—District 7
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At the meeting of Directors, Tuesday evening, J. V. W. Reynders was re-elected First Vice-president, Charles F. Rand, Treasurer, F. F. Sharpless, Secretary, and Percy E. Barbour, Assistant Secretary.

Technical Sessions

The technical sessions related to so large a variety of subjects that it was necessary to arrange three and often four simultaneous meetings.

The largest program was conducted by the Division of Petroleum Technology, Ralph Arnold, chairman. The papers there presented, only a few of which had previously been published, brought up to date our information on the progress of oil and gas developments in all parts of the world.

Joint sessions were held with the Mining Section of the National Safety Council, and with the Mining and Metallurgical Society of America; also, two correlated committees or Divisions of the Institute held joint sessions.

Symposium on Oil and Gas Developments During 1922

Ralph Arnold, Chairman

MONDAY MORNING, FEB. 19

Meeting for organization of the Division.

MONDAY AFTERNOON

Introductory Remarks. Ralph Arnold.
Oil Development, during 1922, in Mexico. By E. DeGolyer.
California. By Wayne Loel and Van Court Warren.
Oklahoma. By Sidney Powers.
Illinois. By D. M. Collingwood.
North Texas. By W. E. Wrather.
Gulf Coast Fields. By Alexander Deussen.
Louisiana and Arkansas. By Clyde M. Bennett.
Rocky Mountains. By Max Ball.
Lima, Indiana. By J. A. Bownocker.
Southwest Indiana. By John R. Reeves.
Kentucky and Tennessee. By L. C. Glenn.
West Virginia. By David B. Reger.
Pennsylvania, North and East Ohio. By Roswell H. Johnson.

TUESDAY MORNING, FEB. 20

Oil Development, during 1922, in Peru. By V. F. Marsters.
Argentina. By Stanley Herold.
Venezuela. By J. W. Lewis.
Colombia. By Joseph H. Sinclair and Edwin B. Hopkins.
Central America. By A. H. Redfield.
Trinidad. By F. Arthur Johnson.
West Indies (except Trinidad). By A. H. Redfield.
Canada. By Ralph Arnold.
Europe, Asia, and Africa. By H. C. Morris and David White.
East Indies. By A. H. Redfield.

TUESDAY AFTERNOON

Action of Mud-laden Fluids in Wells. By Arthur Knapp. Illustrated.
Relation of Bonuses and Costs to Present-day Prices of Crude and Its Products. By Thomas Cox.
Sunburst Oil and Gas Field, Montana. By Dorsey Hager. Illustrated.
Cannel Coal and Carbonaceous Shale Deposits of Pennsylvania. By C. R. Fettke.
Modern Tank Steamships. By J. G. Pew.
Oil Development on the Isthmus of Tehuantepec. By Stirling Huntley. Illustrated.

Mining Sessions

Monday Morning—H. G. Moulton, Chairman

Subsidence at Miami, Ariz. By J. Parke Channing. Illustrated.
Mining an Upper Bituminous Seam after a Lower Seam has been Extracted. By Howard N. Eavenson.
Examples of Subsidence in Two Oklahoma Coal Mines. By J. J. Rutledge. Illustrated.
Some Problems in Ground Movement and Subsidence. By Geo. S. Rice. Illustrated.

Monday Afternoon—B. F. Tillson, Chairman

Review of Present Status of Drill Steel Breakage and Heat Treatment. By F. B. Foley, C. Y. Clayton, H. S. Burnholz.
Hardness and Heat Treatment of Mining Drill Steel Shanks. By C. Y. Clayton.
An Electrical Method of Heat Treating Bars of Drill Steel. By C. C. Waite.

Tuesday Morning—R. M. Raymond, Chairman

Mining Methods at Marquette Mine. By S. R. Elliott, J. E. Jopling, R. J. Chenneour, E. L. Derby.
Mining Methods at the Bunker Hill and Sullivan Mines. By H. M. Childs and Stanly A. Easton.
Economic Application of Zonal Theory of Primary Deposition of Ores. By John Carter Anderson.
Magnetic Methods for Exploration and Geologic Work. By W. O. Hotchkiss.

Tuesday Morning—Howard N. Eavenson, Chairman

Interpretation of Results of Coal-washing Tests. By Thomas Fraser and H. F. Yancey.
Mine Drainage Stream Pollution. By A. B. Crichton.
Pocahontas Coal Field and Operating Methods of the United States Coal and Coke Co. By Edward O'Toole.
Outline for Papers on Recovering Coal. By R. V. Norris.

Tuesday Afternoon—Arthur Thacher, Chairman

Hecla Mining Co. By James F. McCarthy and C. H. Foreman.
Mining Methods at Butte, Mont. By Wm. B. Daly and Associates.
Morning Mine. By Frederick Burbige.
United Verde District. By Arthur H. De Witt Smith.
The Silver King Coalition Mine. By Robert S. Lewis.
The Rove Tunnel. By M. Mathieu. Illustrated. Presented by John R. Freeman.
Mechanical Loaders at Underground Mines. By Charles E. Van Barneveld. Illustrated.

Wednesday Morning—George A. Packard, Chairman

Liquid-oxygen Explosives at Pachuca. By Michael H. Kuryla and Galen H. Clevenger. Illustrated.
Copper Range Co. By W. H. Schacht. Illustrated.
Lake Superior Copper. By F. W. Denton.
Mining Methods of the Miami Copper Co. By J. H. Hensley, Jr.
Mining Methods of the Inspiration Consolidated Copper Co. By A. C. Stoddard.
Cripple Creek District. By Fred Jones.

Metallurgical Sessions

MONDAY AFTERNOON—Arthur L. Walker, Chairman

Heap Leaching at Bisbee, Ariz. By A. W. Hudson and G. D. Van Arsdale.

Soluble Silica in Preparation of Zinc-sulfate Solution for Electrolysis. By J. O. Betterton.

Pyritic Smelting and Basic Converting at the Kosaka Copper Smelter Japan. By Kenzo Ikeda.

Relative Efficiency of Amalgamation and Cyaniding. By Allan J. Clark and W. J. Sharwood.

TUESDAY AFTERNOON—C. A. Meissner, Chairman

Design and Operation of Roberts Coke Oven. By M. W. Ditto.

Heat Distribution in the New Type Koppers Coke Oven. By Jos. Van Ackeren. Illustrated.

Forms of Sulfur in Coke and Their Relation to Blast-furnace Reactions. By A. R. Powell.

Combustion of Blast-furnace Cokes in Fuel Beds. By R. A. Sherman and John Blizard. Illustrated.

Combustion of Coke in the Blast-furnace Hearth. By G. St. J. Perrott and S. P. Kinney. Illustrated.

Use of Magnetic Ore in the Blast Furnace. By G. P. Pilling.

WEDNESDAY MORNING—George K. Burgess, Chairman

Continued Discussion on the Physics of Steel. By W. R. Webster.

A. S. T. M. Tentative Specifications for Foundry Pig Iron. By Richard Moldenke, Chairman Committee A3, American Society for Testing Materials.

Low-temperature Brittleness in Silicon Steels. By Norman B. Pilling. Illustrated.

WEDNESDAY AFTERNOON—Albert Sauveur, Chairman

Deterioration of Malleable in the Hot-dip Galvanizing Process. By W. R. Bean.

Heating and Cooling Curves of Large Ingots. By F. E. Bash.

Application of Colloid Chemistry to Production of Clean Steel. By H. W. Gillett.

Influence of Temperature, Time and Rate of Cooling on Physical Properties of Carbon Steels. By H. M. Howe, F. B. Foley and Joseph Winlock.

Institute of Metals Division

MONDAY AFTERNOON—Arthur S. Dwight, Chairman

ANNUAL LECTURE

Solid Solutions. By Walter Rosenhain, F. R. S., D. Sc.

TUESDAY AFTERNOON—William B. Price, Chairman

Nature of Solid Solutions. By Edgar C. Bain.

Cored Crystals and Metallic Compounds. By Edgar C. Bain.

Observations on Occurrence of Iron and Silicon in Aluminum. By E. H. Dix, Jr. Illustrated.

Polishing Aluminum and its Alloys for Metallographic Study. By E. H. Dix, Jr.

Thermal Properties of Aluminum-silicon Alloys. By Junius D. Edwards.

Practical Spectrographic Analysis. By W. F. Meggers, C. C. Kiess, F. J. Stimson. Illustrated.

WEDNESDAY AFTERNOON—George K. Elliott, Chairman

Occurrence, Chemistry and Uses of Selenium and Tellurium. By Victor Lenher.
Determination of Gases in Metals. By Harold L. Simons.
Study of Bearing Metals. By C. H. Bierbaum. Illustrated.
Tests on High-tin Bearing Metals. By P. E. Priestley.
Bright Annealing of Copper Wire in an Atmosphere of Natural Gas. By P. E. Demmler.
Thermal Conductivity of Some Industrial Alloys. By H. M. Williams, and V. W. Bihlman.

Sessions on Industrial Relations and Safety

MONDAY AFTERNOON—Robert Linton, Chairman

Reports of Sub-Committees on:

- Americanization.
- Cripples in Industry.
- Education.
- Employment.
- Housing and Recreation.
- Mental Factors in Industry.
- Prevention of Illness.
- Safety.

WEDNESDAY MORNING*—B. F. Tillson, Chairman

Safe Practices in Mine-hoist Equipment. By Rudolph Kudlich. Discussed by C. C. Sunderland and J. B. Warriner.
Non-destructive Testing of Steel Hoisting Rope. By R. L. Sandford. Discussed by David Lindquist.
Mine-fire Prevention. By Orr Woodburn.
Installation of Fire-fighting Equipment in Mines. By B. F. Tillson. Illustrated.
Mine-fire Control Safety Orders. By Geo. S. Rice.

WEDNESDAY AFTERNOON*—B. F. Tillson, Chairman

Industrial Psychiatry. By Jau Don Ball.
Industrial Relations, Tennessee Copper Co. By J. N. Houser.

Joint Meeting with Mining and Metallurgical Society of America

WEDNESDAY AFTERNOON—Allen H. Rogers, President, M. & M. S. A., Chairman.
Practical Results Obtained from Psychological Tests at Columbia. By Adam Leroy Jones.
Relations between the Mining Industry and the Technical College. By F. W. McNair.
Training of Engineering Students. By E. P. Mathewson.

Local Section Convention

WEDNESDAY MORNING—E. P. Mathewson, Chairman

Informal discussion of matters of interest to the Local Sections; especial attention was given the question of licensing engineers.

THE SMOKER

The annual smoker was held on Monday evening and was attended by a large and enthusiastic audience. Professor Kemp introduced the famous explorer, Capt. Walter E. Traprock, with quotations from the scientific (?) portions of his memoirs "The Cruise of the Kawa" to the South Pacific. Captain Traprock, after modestly comparing his exploits with those of Columbus, DeSoto, and others, related his latest adventures under title "My Northern Exposure" in taking a ship to the North Pole. As if corroborative evidence were necessary, he illustrated his lecture by views of ships, ice flows, and seals, and an easily identifiable photograph of himself holding fast to the Pole. All doubts were dispelled on beholding this display of winter scenery. The meeting then adjourned to light refreshments and singing.

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* Joint sessions with the Mining Section of the National Safety Council.

THE BANQUET AND OTHER SOCIAL EVENTS

On Tuesday evening, H. von Eckermann entertained the Institute with eleven reels of most interesting moving pictures illustrating the charcoal iron industry, as conducted at Ljusne, Denmark. The pictures covered every step in the operations, from the making of the charcoal to the production of the finished iron. These pictures were followed by others, from the Ditmars collection, illustrating the teeming life of the ocean waters. After the pictures an informal dance was begun, lasting until midnight.

The presence of Prince Gelasio Caetani, Royal Italian Ambassador to the United States, together with the presentation of the James Douglas gold medal to Frederick Laist, of Anaconda, Mont., made the banquet the most notable in the history of the Institute.

The banquet was held in the gold ballroom of the Waldorf-Astoria. The only decorations were two huge flags, of Italy and the United States, draped high above the speakers' table and joined by the illuminated seal of the A. I. M. E. There were 688 members and guests at dinner, which was enlivened by the excellent rendering of favorite Italian operatic airs.

After speeches by the retiring and by the incoming president, Dr. A. R. Ledoux nominated Frederick Laist for the James Douglas Gold Medal for "Research and Achievement in the Metallurgy of Copper and the Electrolytic Production of Zinc." The medal was presented by Col. Arthur S. Dwight, together with an engrossed diploma and a beautifully bound volume containing a collection of the papers Mr. Laist has presented to the Institute during the last twelve years.

The Woman's Auxiliary conducted a long and varied program under the guidance of a large committee. Most important was the business session on Tuesday, at which reports of the growing activities of the parent organization and of its numerous local sections were read. Visits were made to theaters and art galleries, including Senator Clark's.

A reception was given by Mr. and Mrs. R. L. Lloyd, at the Waldorf, in honor of the incoming and the retiring presidents.

On Thursday, Feb. 22, a special train carried 320 members and guests to Ansonia, Conn., to visit the plant of the American Brass Co. Between the forenoon and afternoon trips luncheon was served in the Armory of the 169th Infantry. J. A. Coe, President of the American Brass Co., made a brief speech of welcome which was acknowledged by President E. P. Mathewson.

Canadian Meeting, 1923

The 128th meeting of the American Institute of Mining and Metallurgical Engineers constituted a two weeks trips through Ontario and Quebec. The last visit of the Institute to Canada was made in 1907, when the mines, which now have brought world-wide fame to some of the camps, were not discovered. The party reached its greatest numbers at Porcupine, where 167 visitors and nearly one hundred local members participated.

On Monday, Aug. 20, was spent in Toronto. Registration office was opened in the Parliament Buildings and maintained until late at night, when the party left on its special train for the great northland.

At noon special launches conveyed the members and many Toronto guests across the harbor to the Royal Canadian Yacht Club where the party was entertained at luncheon as guests of the city of Toronto. Mayor Maguire of Toronto made a speech of welcome and gave the keys of the city to the A. I. M. E. After luncheon the party became guests of the Toronto Harbor Commission and enjoyed a trip around the harbor.

In the evening, a reception was held in the Legislative Chamber, Parliament Buildings, by Hon. Charles McCrea, Minister of Mines, and the Hon. W. F. Nickle, the Attorney General. Following this reception motion pictures illustrating nickel, gold and silver mining in northern Ontario were shown in the West Hall, Parliament Buildings; afterwards light refreshments were served in Private Bills Committee Room. The guests were conveyed in automobiles to their special train, to which was attached the private car of the Minister of Mines.

On Tuesday morning, at Coniston, two cars were detached and run into the Mond Smelter. These members next visited the smelter of the British America Nickel Corpn., at Nickelton, which was designed and built by President Mathewson; and afterwards the mine of this Corporation, one mile distant; but there was not time to go underground. The train then made the Creighton mine of the International Nickel Co., where a large party, including many ladies, was taken underground through various workings and stopes.

The train then proceeded to Copper Cliff where the smelter was visited by those metallurgically inclined. After dinner, in the dining cars, the ladies were entertained at the home of John L. Agnew, General Superintendent of the Canadian Copper Co.; the men were entertained at a smoker at the Copper Cliff Club. The train moved during the night and the party woke Wednesday morning in Cobalt.

Two Days in Cobalt District

In the forenoon the party was taken by automobiles to the Nipissing mill and refinery. At the latter had been collected the richest and rarest specimens from all the mines and engineers of the district, together with typical rocks, polished sections, various mineral products, and a relief map of the entire district showing the areal geology in colors. As the party departed, colbalt-blue cupels were given as souvenirs.

In the afternoon, various geological and mining parties went down the Myers shaft of the Nipissing mine and were shown the stopes that made Cobalt famous. One large party visited Ragged Chutes to see the fourth installation in America of the Taylor pneumatic air compressor. Three automobile loads of golfers were taken to the Country Club, overlooking beautiful Lake Temiskaming. The ladies were entertained at tea by Mrs. J. G. Dickenson at the O'Brien mine.

In the evening, the Town Hall was filled with the visitors and local hosts who listened to a number of unusual addresses. A. M. Bateman presided. Dr. J. M. Bell gave an interesting talk on the geology to be observed at South Lorrain, when visited on Thursday. Dr. Willet G. Miller, the Provincial Geologist, told of the discovery and early development of Cobalt. Dr. C. H. Taylor, the inventor of the Taylor hydraulic air compressor told of the conception of his idea when he put in a dam which had a 4-ft. overflow in the winter; this cut a hole through the ice at the base and compressed the air, which accumulated a short distance down stream and made a geyser through the ice.

President Mathewson gave a short address and was followed by William Kelly, of the Institute Board of Directors, who expressed appreciation of the welcome and entertainment.

Thursday, a party went by motor to South Lorrain where the Keeley and Frontier mines were visited. Luncheon was served at the Keeley where Doctor Bell was host. The party then went to Silver Centre on the Lake to join the main party.

The special train proceeded to Haileybury, where the main party went on board the steamer Meteor for a trip down Lake Temiskaming and a picnic at the Notch.

A brief visit was made to the town of Ville Marie on the Quebec side, near which is the Wright silver mine, formerly the Ance a la Mine, the oldest mine of record in Canada.

Thursday evening was spent in Cobalt. A dance was given in the Town Hall by the Mine Operators Association for the visiting ladies.

Friday, the party arrived in Swastiska and motored six miles to to Kirkland Lake. The movie theatre was filled to overflowing when Percy E. Hopkins described the geology of the district, illustrating by colored slides. The party was then conducted through the mills and mines of the district and then back to Swastika to its train.

Saturday morning the party awoke in Iroquois Falls and later visited the model plant of the Abitibi Power and Paper Co., Ltd. This plant has a capacity of 500 tons of newsprint daily, which requires 725 cords of wood.

PORCUPINE GOLD AREA

The Institute Special arrived at Timmins soon after luncheon and was welcomed by a large party of men and women. The station was decorated with English and American flags and bunting and a large tent had been erected alongside the platform, which was information headquarters of the local entertainment committee. There was telephone connection and writing facilities, mail was taken care of, and a fleet of automobiles constantly in attendance to carry anybody anywhere.

In the afternoon Mrs. R. J. Ennis entertained at tea and both Saturday and Sunday there were many private functions between visitors and old friends.

Saturday night a banquet was given in the huge skating rink at Timmins followed by dancing.

Sunday, the visting of friends, mines and mills was continued until 5.30 p. m., when the special train left for Quebec. A large number of the party, including the party of the Minister of Mines in his private car, returned to Toronto.

QUEBEC

Monday was taken up by the ride to Quebec through a most interesting and picturesque country.

Tuesday, an automobile tour of the quaint and interesting old city of Quebec was made, visiting the Citadel, Battlefields Park, Sillery, Quebec Bridge and other points of interest. In the afternoon, sightseeing trips were made to St. Anne de Beaupré where the cyclorama of Jerusalem and the Crucifixion, one of the three famous cycloramas of the world was exhibited. Thousands of people make the pilgrimage to St. Anne de Beaupré annually to visit the holy shrine, where is kept the miracle working relic of St. Anne, the mother of the Virgin Mary.

Wednesday forenoon was spent among the asbestos mines and milling plants at Thetford, which produce from 75 per cent. to 90 per cent. of the world's supply of this mineral. Wednesday evening, the party reached Montreal and the special train was left for the last time. The Mount Royal Hotel was Institute Headquarters.

Thursday, Technical Sessions were held in the hotel according to the following program:

Mining and Milling Session

John A. Dresser,* Chairman

Contribution to the Kick *vs.* Rittinger Dispute. By H. E. T. Haultain.
New Slow Motion Pictures of Ball Paths in Tube-mills, and Rock Crushing in Rolls. By H. E. T. Haultain and F. C. Dyer.
Proposed Plan for Crushing, Grinding, and Concentrating Low-grade Sulfide Copper Ore. By Arthur Crowfoot.
Mining Methods at the Bawdin Mine, Burma. By A. B. Calhoun.
Emergency Power for Mines. By Graham Bright.
Loading Ore Underground with Scrapers at the Utah-Apex Mine. By S. P. Holt.

Geology Session

Frank L. Hess, Chairman

Helium, a National Asset. By Richard B. Moore.
Cherts and Igneous Rocks of the Santa Elena Oil Field, Ecuador. By Joseph H. Sinclair and Charles P. Berkey.
Practical Results Obtained by Schlumberger Method of Electrical Prospecting. By Sherwin F. Kelly.
Magnesite Deposits of Grenville, Quebec. By G. W. Bain.
Secondary Enrichment at Eagle Mine, Bonanza, Colo. By C. Erb Wuensch.
Porcupine Ore Deposits. By Louis D. Huntoon.

Iron and Steel

William Kelly, Chairman

X-Ray Examination of Irregular Metal Objects. By Ancel St. John.
X-Ray Examination of Steel Castings. By H. H. Lester.
Nitrogen in Steel. By C. Baldwin Sawyer.
Desulfurizing Power of Iron Blast-furnace Slags. By Richard S. McCaffery and Joseph F. Oesterle.
Effect of Silicon on Equilibrium Diagram of System Carbon-iron, near the Eutectoid Points. By H. A. Schwartz, H. R. Payne and A. F. Gorton.
Some Commercial Alloys of Iron, Chromium, and Carbon in the Higher Chromium Ranges, by C. E. MacQuigg.
Some Effects of Zirconium in Steel. By Alexander L. Feild.
Selecting Material for Formed and Drawn Parts. By L. N. Brown.

Thursday evening there was a reception and formal banquet followed by dancing.

Friday was spent in automobile sightseeing trips to the points of interest in Montreal, one party went to Cornwall by train and made the return trip by boat, returning to Montreal late in the afternoon. This ended the trip and evening trains south and west carried the various members toward their respective homes.

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* President of Canadian Institute of Mining and Metallurgy.

PAPERS

Present Tendencies in Exploration for New Mines*

By Augustus Locke, San Francisco, Calif.

(San Francisco Meeting, September, 1922)

This paper has two parts—one for broad discussion, the other for technical illustration. For technical illustration, two experiments are described in the judgment of leached copper cropping. These are part of a general leached cropping study prosecuted during the past four years by the author and his associates—a study in which Charles H. White has been, in an important way, involved, and has accomplished similar experiments with definite results. The experiments described were by H. W. Morse, P. F. Boswell and E. E. Erich. An outstanding peculiarity is the use of numerous cropping specimens taken on formal patterns.

Experiment No. 1

The first experiment concerns 524 specimens of cropping and float from two 8-acre mineralized areas, over ore and over waste. Earlier results from these specimens were described in a previous paper.[1] The present results decisively select the good area and indicate an improvement in technique during the interval. They are shown in Figs. 1 to 8.

In their first trial, the men were fresh to the problem, for, from this district, they had seen only a suite of sulfides and that only cursorily. They appreciated the danger of prejudice. None of them knew the derivation of particular specimens. After the first trial, two of them knew the result in relation to the specimen key. After his third trial, Doctor Morse wrote: "From the glance I had at the key the other day, I learned that most of the good specimens were among the low numbers. So, whenever I saw the number, I threw the specimen back in the heap for another judgment. In spite of this, I feel that my judgment is no longer pristine and should regard the first judgment as the safe one to expect if an unknown capping were being examined. I know some of the specimens by their features and cannot judge them impartially."

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* Presented at a joint meeting of the American Institute of Mining and Metallurgical Engineers and the Society of Economic Geologists.

[1] *Mining and Metallurgy* (1922) **184**, 27.

The third man finished his three trials with no information regarding success and no knowledge of the key. His respective judgments were liable to prejudice only by his preceding judgments and this prejudice he tried to avoid by making, during a trial, no comparisons with other trials. He aimed to prevent the later trial from merely repeating earlier opinion. We may conclude that his judgments were all uninfluenced

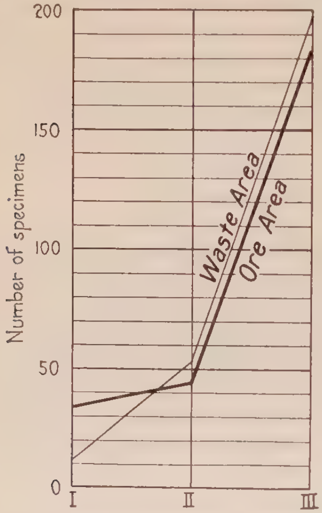

Fig. 1.—First trial by A; time 2 hr. I, Almost certainly ore; II, probably ore; III, hopeless.

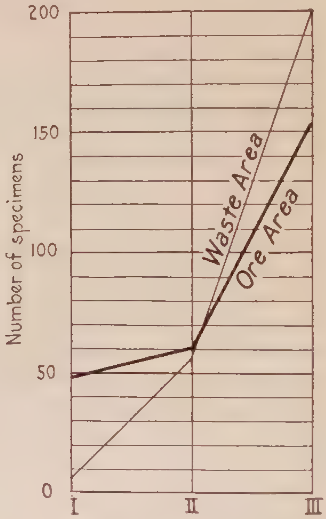

Fig. 2.—Second trial by A; time 4 hr. I, Almost certainly ore; II, probably ore; III, hopeless.

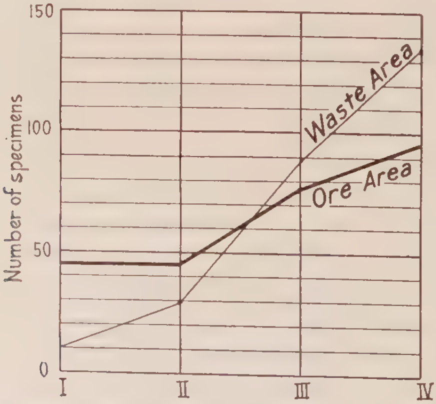

Fig. 3.—Third trial by A; time 4 hr. I, Almost certainly ore; II, possibly ore; III, some signs of ore; IV, hopeless.

by knowledge of the particular specimen's position, and that the first judgments of the other two men were similarly uninfluenced. Furthermore, because the later judgments of the other two men checked their first ones reasonably well, they too seem little influenced by this sort of partiality.

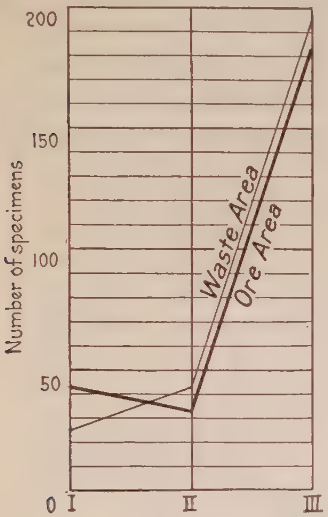

FIG. 4.—FIRST TRIAL BY B; TIME 2 HR. SYMBOLS SAME AS FIG. 1.

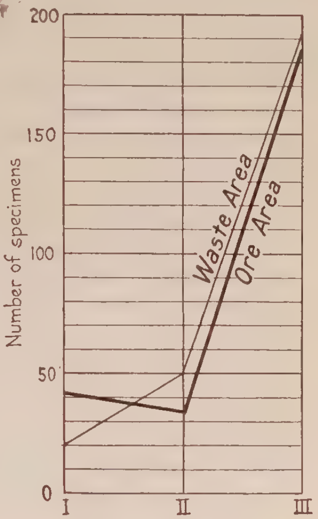

FIG. 5.—SECOND TRIAL BY B; TIME 4 HR.

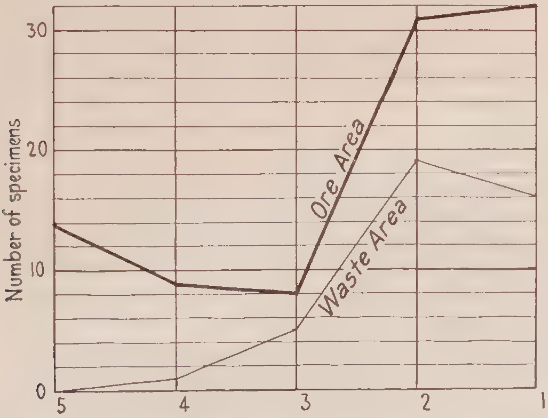

FIG. 6.—SUMMARY OF RESULTS OF A AND B, INDICATING DISTRIBUTION OF SPECIMENS JUDGED GOOD AT LEAST ONCE OUT OF FIVE JUDGMENTS; 3, JUDGED GOOD THREE TIMES OUT OF FIVE TRIALS.

This leaves the principal chance for partiality to the original selection of the specimens. Roland Blanchard, who has done much croppings work, maintains that ideas on favorability will promote selection in a good area from good looking stuff, and in a poor area from poor looking stuff, so that approval or condemnation will gain undue emphasis.

The samplers knew which area was good. But they took the samples as grabs where the point of the thrown pick happened to fall. Moreover, they had not "found their eyes" with this material; they said it all looked alike and, two weeks later, in a careful judgment of the specimens, one man mildly favored the good area, one the poor area, and the third favored neither. Partiality from prejudice in the samplers, then, seems unlikely.

The criteria of judgment used in Experiment No. 1 depended, at the outset, on information from the examined sulfides, that the pyrite-chalcocite ratio was high. But it soon developed that the capping itself independently conveyed this information; for this showed the limonite subordinately indigenous and chiefly cast out from the voids into the kaolin surrounding them. This permitted the conclusion by chemical reasoning that the ratio of pyrite to chalcocite was at least about two molecules to one.

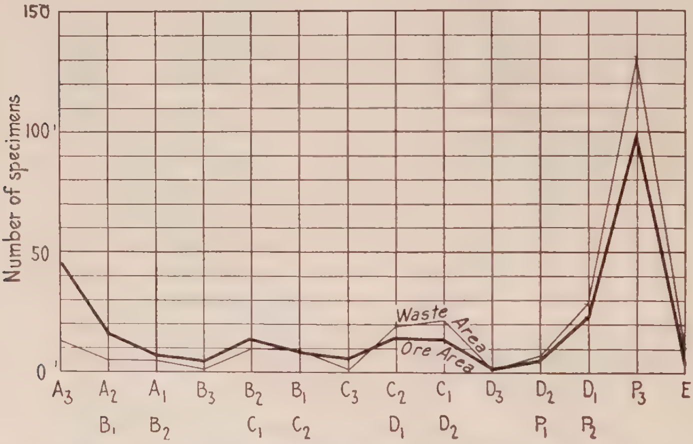

Fig. 7.—Results of three judgments by C; time 2½ hr. for each trial. *A*, good; *B*, fair; *C*, poor; *D*, doubtful; *P*, hopeless; A_2B_1, judged *A* twice *B* once; *E*, erratics, as B_1P_2.

To make ore, the total volume of sufides would have to exceed the volume needed, say, in Miami, where the pyrite-chalcocite ratio is only ½ to 1. Any specimen, then, having meager voids, was discarded as hopeless. And of those specimens that were left, the ones having the larger volumes of voids, or, in the rare case, the larger quantities of indigenous limonite, were counted good.

A distinction was made between pre-oxidation and oxidation voids, and the probability was determined that most of the voids were derived from sulfides.

A slightly darker shade of brown was found to go with the other qualities regarded as favorable. But this, being less emphatic than the other qualities, was disregarded.

The trials came down chiefly to a judgment of voids. This basis was simpler and more specific than that used previously by the writer on the same specimens; instead of involving several variables of doubtful relative importance, it made the whole decision rest on one. The more complicated basis had served to prefer the good area, but less emphatically than this and more laboriously.

The criteria here used are, of course, especially fitted to these specimens; they have no general application. Other conditions might require the use of indigenous limonite, or of a combination of indigenous limonite and voids, or of the degree of particular kinds of rock alteration. Nor should it be concluded that, because the basis is simple, the whole matter of judgment was easy. The principal task, and the one needing discrimination, was the selection of a basis that would serve for sane conclusions. This, in fact, was not an experiment strictly in the judgment of leached capping. It was an experiment in the separation of specimens from two areas, one of which was known to overlie ore. The conclusions were calibrated by that fact. In this regard, Doctor Morse says: "We should be given an unknown set of specimens from over waste, and should be told to separate them into ore and waste. That would be a more serious test of skill."

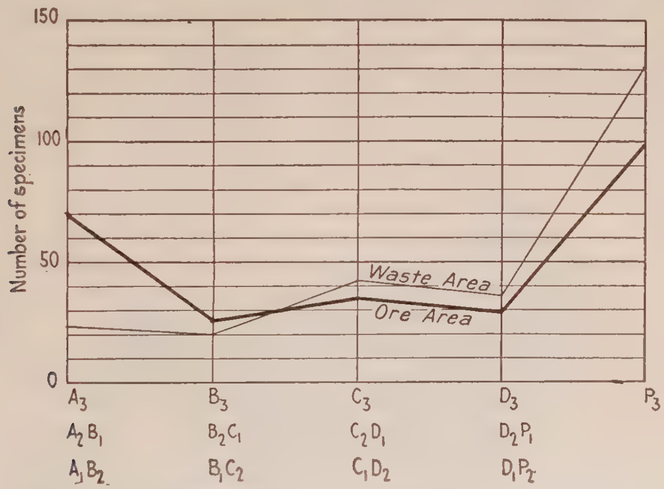

FIG. 8.—CONDENSATION OF RESULTS SHOWN IN FIG. 7.

EXPERIMENT No. 2

The second experiment concerns 6760 specimens of cropping and float, collected by G. G. Tunell, Jr., on a formal pattern, from a mineralized area ½ mile square, including the good ground of the first experiment. This area had been partly drilled, and was known to include several ore-bodies. About two-thirds of the ground was unexplored. From each 100-ft. square, there were taken ten specimens of cropping or float. The results are expressed in Figs. 9 and 10.

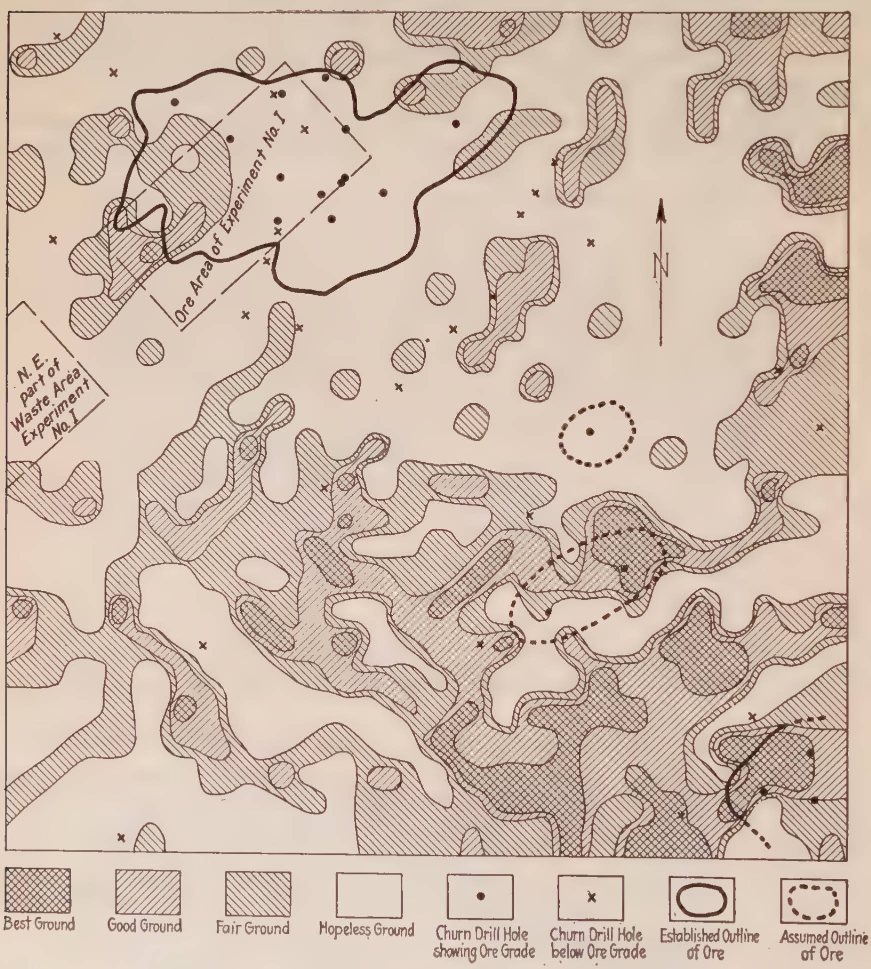
Ore Area of Experiment No. I
N. E. part of Waste Area Experiment No. I
N
Best Ground
Good Ground
Fair Ground
Hopeless Ground
Churn Drill Hole showing Ore Grade
Churn Drill Hole below Ore Grade
Established Outline of Ore
Assumed Outline of Ore

Fig. 9.—Classification of 160 acres by B by means of 6760 specimens, time 72 hr. Each specimen was rated on a scale of 1:10. Scale 1 in. = 600 ft. Depth of soil about 2 ft. Known depth of leached capping increases from 50 ft., in the southeast corner, to 550 ft., in the northwest corner. Specimens were about one-fifth from cropping and four-fifths from float. For Experiment No. 1, they were taken one from each 40-ft. square; for Experiment No. 2, they were taken ten from each 100-ft. square.

The judging of the specimens was accomplished, in the laboratory, by two men who had no knowledge of their positions. Unless, then, the selection of the specimens from the ground was prejudiced, the judgment was uninfluenced by a knowledge of position.

The selection of the specimens from the ground had little chance of prejudice, because, as in Experiment No. 1, it was accomplished by grab methods, with no standard of choice in the mind of the collector, and with no detailed knowledge of the position of the discovered ore. More drilling had been done in ore than elsewhere, and so, in a rough way, the position of the largest orebody was evident on the ground. But, with no standard of judgment, the collector could not have been forced into false sampling. The basis of judgment used by the men was similar to that used in Experiment No. 1.

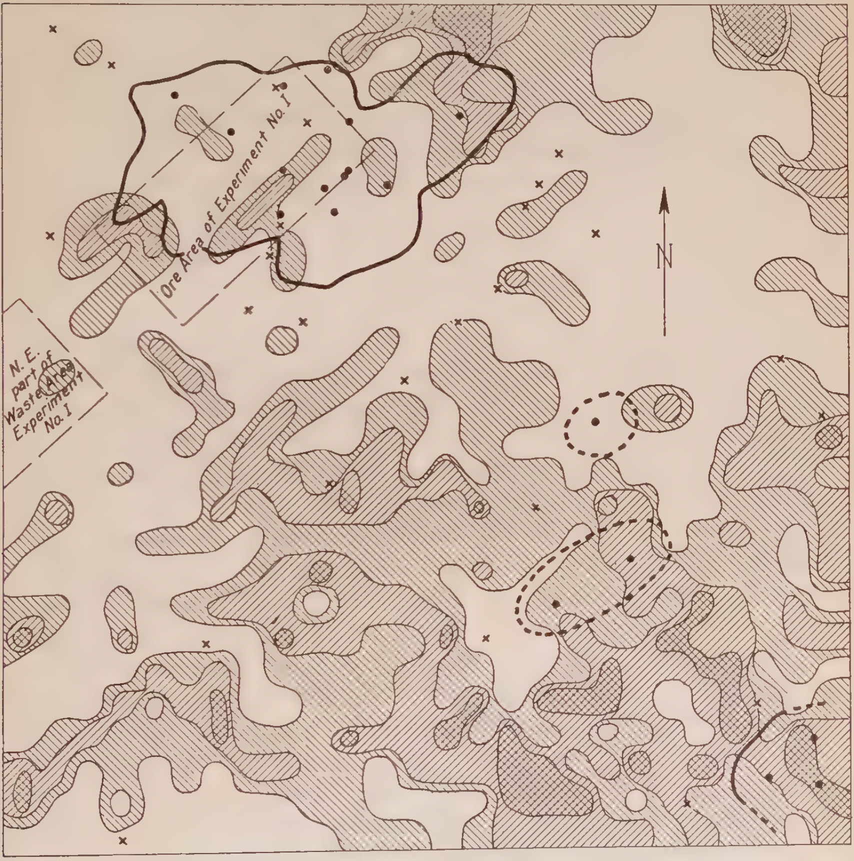
Ore Area of Experiment No. 1
N.E. part of Waste Area Experiment No. 1
N

Fig. 10.—Classification of 160 acres by *A* by means of 6760 specimens; time 18 hr. Each set of 10 specimens was rated on a scale of 1:8.

The results show: (1) marked preference for about one-third of the area; (2) no special choice of the ground over the largest known ore, that in the northwest corner; (3) suggestions of an arrangement of better ground in lines of northeast strike; (4) an increase of favorable ground, where in the southeast part the capping is thin, in comparison with the northwest part.

It will be noted that the good area of Experiment No. 1 lies above the largest known orebody of the ½ mile square area of Experiment No. 2; that the specimens of that good area were given a favorable rating; and that the same ground judged in Experiment No. 2 was given partly a moderately favorable, and partly an unfavorable rating. Is the discrepancy apparent or is it real? What is its cause?

The good area of Experiment No. 1 was judged good in relation to the poor area. It proved much better than the other area. Had that other area been especially good, instead of especially poor, the good area would have been judged, relatively, poor.

But this is not the explanation. The sample from the good area of Experiment No. 1 shows a strikingly better proportion of favorable specimens than the sample of the whole orebody area of Experiment No. 2. But that good area is only one-half that of the orebody. With specimens from the rest of the orebody excluded, it shows up about as well in Experiment No. 2 as in Experiment No. 1. The inference is that the good area of Experiment No. 1 had surface expression more favorable than the average of the orebody; the good area happened to fall on an especially good part of the cropping.

That this part of the cropping should be especially good is, in fact, accordant with the structure known from mine workings. Mineralization, hypogene and supergene, is guided by fractures of northeasterly trend and southeasterly dip. With the dip of 45°, which is usual, the easterly edge of the ore country would project up to the medial part of the good area of Experiment No. 1; the surface expression of the ore country would be slid northwesterly a distance of 500 ft., for the ore lies 500 ft. deep. More than one-half the country of Experiment No. 2 overlying the orebody would be the upward projection of barren ground.

But what is the assurance that the specimens are abundant enough for a fair sample? The specimens taken in the two experiments numbered, respectively, only about 25 and 40 per acre of ground; this compares with about 500 samples, each weighing 20 times as much, which would be taken, according to good practice, from this area of ore. That so many samples of croppings are needed is unlikely; but that the number used is enough, is by no means certain. Parallel sampling, with the coördinates moved over a little, is needed to answer the question.

These experiments show that, on this ground, laboratory study can be used for a separation of the specimens and that it affords a basis that is probably, though not certainly, logical. They emphasize the need of proving that the sample used is fair; and the need of field study with laboratory study, each to guide and control the other. One indicated procedure would be the tracing of all favorable spots, to their limits, by trenches through soil. Others would be the mapping of mineralized fractures; post-hole sampling, on a formal pattern, from capping in place; and a study, on the surface, of the vagaries of primary sulfide content for, with capping 500 ft. deep, only very large patterns of variation will make the surface indicative of vertically underlying sulfide. Using these devices, the geologist must always remind himself that deep capping may mean cumulative enrichment from surface downward and may therefore mean that the cropping jumped directly from protore, with no enrichment interval; that shallow capping, then, merely by being closer to sulfide, may sometimes contain more of the indigenous limonite that results from sulfide mixtures rich in chalcocite and less of the acid effect which results from sulfide mixtures rich in pyrite and may, therefore, be more favorable without proving better sulfide. A lively sense is needed, then, of differences due to thickness of capping.

Discussion of Present Tendencies

Ore discovery in western North America falls in three periods: the first, that of great, new districts—events of the advancing frontier; the second, that of new mines in old districts; the third, that of new ore in old mines, near known old ore.

We are now in the third period. Are the new mines then, like the new lands, all taken? This is the naive explanation. But it merits no off-hand acceptance, for it is balanced by an alternative quite as reasonable, namely that, though the easily found mines have been taken, many difficult to find, beyond the power of the prospector, await a finer ore-hunting art. We may not conclude that another generation will not show new mines again commonly coming out. Before we believe that ore discovery is over, we must think of the kinds of ore that may still lie hidden, and we must consider the present methods of search and their fitness for this hidden ore.

First, then, as to the kinds of hidden ore. Much ore lies covered by wash and, in Canada, by moss and water. In arid country, such ore is sometimes prolonged to the edge of the wash; but in forested country, the cover is more continuous and more baffling. Geology, aided by small drilling outfits, will better present methods; physiography, too, will serve, and we cannot deny the chance for aid through electrical devices.

The conspicuous bulky croppings, with iron and copper stain, are the copper hunter's favorites; they lie broadside to the sky; in Arizona, Utah, and Nevada, because of the success of low-grade mines, such croppings are grooved by prospectors' picks; in Colorado, more may lie untested. But, in general, ore with the familiar Utah Copper or Shasta County croppings promises but occasional new mines. If the ore, however, though disseminated and bulky, is especially high in pyrite or low in neutralizer (such as calcite or feldspar), the cropping evidences of copper may be obscure. If the ore is low in iron or lies in a forested country, where organic acids may dissolve the limonite from the outer skin, the cropping may be even more obscure. Imposing deposits of both these types may await discovery. The Utah Copper type itself, mixed with harder low-grade material, might carry this low-grade material projecting as knobs through the wash and have the better stuff buried. Under imaginable conditions, such things could have evaded every ore hunter that has so far seen them. Were the ore spotty, it would be especially hard to find.

The spotty garnet-limestone deposits, bearing croppings that are tough and easy to find, attracting the prospector because they yield little surface bodies of shipping ore but repelling the engineer because they make few mines, have put much ground in bad repute. Yet, alongside these may lie rich limestone ore that weathers soft and is covered. As possible indicators of something hidden, garnet croppings will require thoughtful study.

Bulky, massive, sulfide deposits, especially pyritic and especially low in rock relics, may weather to a sand, like that of Kyshtim, Russia, pinched down to a third the width of the sulfide itself. Unless gold emerges as a lure, such sand can lie neglected, though crossed by a well traveled road.

In general, smaller, richer orebodies seem more likely of future discovery than larger, poorer ones; steep bodies like the United Verde than flat deposits like the Miami; and those with deep primary than those with shallow secondary ore. Veins are more likely than bulky deposits, for veins may come out as thin discontinuous lines, and may lie with their big end down. Butte is an illustration; the veins there have been discovered less on the surface than underground and the surface is in sharp contrast to Miami, where the red hills show miles away.

Deposits whose locus fails to reach the surface, such as those at a flat bed and a steep fracture; veins cropping in the bad part of a rock series with better parts deeper; ore terminated by faults: these in some measure are saved for the future. Certain deposits may be saved because they are paired with others of more spectacular croppings. Thus, a copper body deficient in neutralizer will bear a cropping low in copper; a body excessive in neutralizer may carry as much copper at the surface as it does below and will be less liable to sulfide enrichment; if the two lie close together, the latter claims first attention, and, failing, kills the former.

Strange types of deposit, of which no example is known in North America, may go long unheeded. Thus a bulky copper body may have pyrite sufficient to expel the copper during oxidation and may have cavities from the hypogene corrosion of feldspar. This, to the American copper hunter, would look outlandish and, on present knowledge, would be rejected. There may be some valuable deposits of this sort.

The obvious gold deposits have small chance to hide in a settled country, for their placers signal their presence. But deposits devoid of placers through glaciation, fineness of the gold, or the solution of the gold, may have escaped detection even within hearing of the city.

Silver deposits, like Tonopah and Pachuca, hid by a cap of later barren rock, may be waiting by the dozen; where the cap is thick, they may wait a long time for some finer devices furnished, perhaps, not by miners, but by physicists.

Fortuitous orebodies, those to which no specific guides are known or likely, make a large class. Their past search, especially in new districts, has been scatter-brained; by technicians, they have been scorned; by others, they have been eagerly sought, but mainly with mere physical energy. In the future, with keener attention, their search seems certain to improve. In addition, there are prospects condemned by a little underground work. A few drill holes, yielding unenriched pyrite, may prejudice opinion against a hundred acres of mineralized ground; and yet those holes may yield as false a sample as a few grooves in a gold vein. The very sight of the pyrite gives a comforting assurance in condemnation. Unless the holes are so placed that their results are proved critical of the whole, the problem lies wide open. The adverse argument, of money used without resulting ore, is too obvious and convincing. We may hope that certain prospects, killed by this argument, may yet be revived into mines.

These examples suggest kinds of ore deposits likely of future discovery. Other kinds could be added, and still others, now unimagined, may exist. All have the common quality of being obscure to the multitude and of being the sort that would lie undiscovered, except by accident, under the feet of pioneers. The discovery of most of these is, then, a matter of fine and careful search.

What is the present practice? It is not a fine and careful search. The prospector works much as he always did, though with less spirit. We who act as examining engineers are only refined and less patient prospectors—better business men and better explainers of opinion, but hoping, by mere activity, to come at last to the happy valley where obvious signs of ore will crop before our eyes. This is a false hope. Future efforts must include more than energy and financial competence. They must provide convincing judgments on questions now avoided. They must provide information, cheaply, in advance of exploration, and dispel unreasoned prejudice. They must be so thorough that no chance for ore which, in the mining sense, is a good speculation, will be cast aside through apathy, or hurry, or shallow thinking.

This means a broad scheme of change and the question arises: how rapidly may this change be made and by what means? The only known means is the scientist. He is as necessary here as he is in chemical industries. To some small extent, he is already in the business, but chiefly by proxy, through his students, or, incidentally, because he is interested in geology for itself and ore is an item of geology. The trend toward the scientist in ore hunting does, in fact, show signs of becoming active. Men who control exploration are realizing the need of broad experimental studies by the best technicians they can find. They are realizing that a given prospect is not an isolated problem to be finished in a day, but part of larger problems. They are likely, in time, to support formal researches on these larger problems.

The search for new mines will become a more studious, deliberative affair. Not much longer will good practice allow the scout to see prospects, so many in a month, in separated districts. Rather, it will involve examinations, not so much prospect by prospect, as district by district. Under certain conditions, it will subject the particular district to long study.

Future practice will depend more on statistical methods than on rules-of-thumb. Other conditions being similar, it will, for example, regard favorability as increasing with proximity to known important ore. The expenditure of round sums, such as $200,000, for bold sinking in unproductive districts or in unfruitful parts of productive districts, will be undertaken less cheerfully; it will be undertaken only after convincing reasons, chiefly technical, have been found to support it.

There is even danger that in the reaction against older methods, the control of exploration may be overbalanced on the technical side, and the financial and purely management aspects subordinated. But this is an abuse that attends all progress.

The technical control of exploration seems on the way. Its usefulness is questioned by the man in the street, because ore discovery is, at best, a speculative business. But there is a difference between one chance in fifty and one in a hundred, though both are speculative. Before the lapse of twenty years, technical prospecting promises a reduction of the speculative quality in something like the proportions suggested.

DISCUSSION

G. N. Bjorge, San Francisco, Calif.—Some cases have come to my mind where this method would not work. In one case, a mine was developed on a well-defined vein through a depth of 1200 ft. with no ore. On the 1300-ft. level, it opened into exceptionally good ore and has continued well below that. In another case, a vein was developed that had a little ore at the surface, and at the 200- and 300-ft. levels; but below the 300-ft. level a considerable length of the vein was fully developed down to the 1400-ft. level without opening any more ore. At the 1600-ft. level and below, the ore was continuous for about 100 ft. On the 1900-ft. level, work was discontinued because of the change in wall rock, diabase above and limestone below. In a case like this, the surface might be examined at some point in that barren horizon so that a study of the croppings along that vein would not give any indication of what might be expected below. The possible change in wall rock would have to be taken into account before any opinion could be formed and then possibly little more than a speculative idea would be obtained unless a great amount of expensive work was done.

A. Werner Lawson, San Francisco, Calif.—Is it going to be possible with this method, to work out some sort of a general rule that will cover all these deposits, or must each case be a separate problem? If the latter, can a basis for determining the value of this scheme in each particular case be worked out?

Augustus Locke.—The problem is different in different places. I doubt if any particular rule can be worked out that will apply to all, or any great number of copper or other deposits. The only thing likely to apply is the method; that is, the investigator, instead of walking over a leached cropping and studying it as carefully as he can in a few hours, will take a fair sample of the cropping to the laboratory and attack it in all the known ways of the technique—attack it, in fact, with greater care than we heretofore have used.

A. Werner Lawson.—My thought was that possibly, by a study of twenty or thirty of these deposits, some general rule might be worked out by the application of which the value of all other deposits might be determined. If each deposit is to be treated as an entirely separate problem, this plan is simply a matter of working out the application of the surface data to anything that may be below.

Augustus Locke.—That is not what I meant. The schedule of logic that applies to one deposit cannot be expected to apply without adaptations to another. But it does, in some degree, apply. There are common characteristics in many deposits and experience with one aids materially in the judgment of another. Surely it will not be possible to draw up rules that will enable an amateur to go out and judge outcrops; he must be familiar with many before he can use the method; and, in addition, he must be something of a laboratory man.

Albert Burch, San Francisco, Calif.—In the application of this particular method was the rapidity of the erosion considered; that is, whether on a certain section of the croppings erosion had kept more nearly apace with the leaching than it had on other sections?

Augustus Locke.—That was one of the problems. The question arose as to whether copper stain was left merely because of some difference in erosion; and whether there was a systematic difference between outcrops in an old physiographic floor and those in gulches. Those in the physiographic floor had undergone especial residual concentration by the whipping away of the softer materials. Those specimens had their characteristics masked by exposure and the country over the big orebody had a lot of ground where there were outcrops of that sort.

J. C. Jones, Reno, Nevada.—In the examination of a deposit it is first of all necessary to familiarize ourselves with the croppings and every other bit of geological information obtainable on the development work done and then apply that information as best we can. This was well illustrated in the second experiment, where the author had found four orebodies developed, but the largest orebody did not show up very well; that is a common experience with us all. The great thing that the geologists are trying to do is to read the surface in such a way as to tell, if it is possible, what is underneath. Of course, they may spend a little money trying to find out whether the information obtained from the outcrop was really true. The work of the geologist, in the future, will be to try to see through the wash; to study his district, finding the mode of occurrence and trend of the orebodies; getting all the information possible from the surface and then be ready to modify his plan of development, according to the information he gets underground.

Sherwin F. Kelly, Lawrence, Kans.—A method of electrical prospecting has been developed by Prof. Conrad Schlumberger of the Paris "École des Mines." I am experimenting with it in this country, for the purpose of locating sulfide orebodies by utilizing the fact that the sulfides, in contact with groundwater, generate an electrical current that may be detected at the surface. This method would be valuable at places where the outcrops show practically no sign of ore by helping to determine whether or not such sulfide orebodies lie beneath.

I. B. Joralemon, San Francisco, Calif.—Most of us, when we recommend a property, have no real scientific reason for doing so. By applying the method just outlined, after it has been perfected, in many cases we may have an absolutely scientific reason for deciding that a prospect is worth developing. For many years this method will cover only certain particular cases. It is most important to follow these methods of careful study of outcrops, but to regard them as of positive and not of negative value. We can say that, based on this study, a prospect is worth developing, but we cannot say that it is not worth developing.

Augustus Locke.—We who are intimately connected with the development of this method are convinced that its application in the judgment of prospects should be made not by the technician alone but in conjunction with the mining engineer. There is always the danger that this kind of a new development will be overbalanced on the technical side, and any one who is familiar with the judgment of prospects will realize that danger at the outset and will see to it that the danger is not incurred.

In work on croppings, there is nothing to be gained by the over-emphasis of the information derivable from them. But, on the other hand, there is nothing to be gained by the over-emphasis of the information derivable by other methods. The important considerations are that the croppings, at the present time, are not half studied and that, when they are well studied, they yield important information at low cost. In a prospect with a bottom level 300 ft. deep, deeper exploration without full knowledge of that level would be bad practice. Most prospects have the surface as their bottom level. That level needs only cleaning up and study to furnish vital information. Certainly, that level will not tell everything, and mines would be lost if it were expected to tell everything. But to ignore it is as bad practice as to ignore any other level.

J. A. Carpenter, Johannesburg, Calif.—I have been much interested of late in the possible use of high-frequency currents in the detection of orebodies. Can any one tell us anything about this?

John Anderson, Tucson, Ariz.—I know a little of the device being developed at the University of Arizona, although I have no connection with the University. About two years ago, Mr. Chilson, the inventor, tried to interest me in the matter and explained his ideas quite fully. I suggested to him certain properties in that district where he might be able to test the device to good advantage, among others, the Mineral Hill Consolidated, about 20 miles south of Tucson. Then G. M. Butler, dean of the Arizona School of Mines and head of the State Mining Bureau, became interested and helped him in many ways.

The device consists of a high-frequency radio transmitter, which is grounded on a sulfide orebody or sulfide exposure somewhere in the mine, and a receiver that can detect the sound. While I have not seen it in operation, Mr. Chilson, last fall, worked over a property in which I am interested and indicated some large sulfide deposits of which there is no indication, and indicated a number that we know about. The device tells you nothing as to the type of the sulfide nor the grade of ore, but it has been successful in indicating sulfide deposits. While I was sceptical of the device at first, I am beginning to think that it may be developed into something that may be of real value in the determination of extensions and faulted sections of orebodies. Taking a given problem where we have learned the nature of our sulfide, something of its value, and of the structural relations under which it occurs, if it can then indicate a sulfide orebody under the structural conditions and formations that we have proved elsewhere in the mine to be profitable, we may be justified in doing other exploration work. I believe that there may be a decided future for the device in that way. The inventor has worked over several properties that I know, and on one in particular he indicated things that may be of decided value. When I have satisfied myself on that particular property, the development of which is under way, as to certain conditions under which the known ore lies, I think that I will try the device again. He spent five or six weeks in my absence working over the ground in the tests; if I satisfy myself as far as I can on other points, I will make a real attempt to demonstrate the machine. He did most of his experimental work on the Mineral Hill Consolidated property, which is a contact metamorphic deposit and, unfortunately, while the management gave him the entire freedom of the property, nobody seems to have paid any attention to his experiments, or to have followed them up to determine whether he actually got results or not.

I. B. JORALEMON.—Both Dean Butler and Mr. Chilson say that while, in some limited cases, their apparatus may be of great value, it is not of universal application. It should be applied with a great deal of common sense, like the dip needle and the magnetometer, with which a number of orebodies have been found. It is not a general formula for finding all the ore in the world and they do not regard it in that light.

F. M. ANDERSON, Berkeley, Calif.—One rather important phase of this subject has not been dealt with; namely, what we see within any given area of mountain surface is apt to be a reflex of things that have entirely disappeared rather than an indication of what is below. Denudation has been long and efficient and has reduced the surface sometimes for thousands of feet, and by leaching and solution the surface has gradually sunk. If you examine the rocks on that surface, you are studying ore deposits that may have existed above and have been carried away; all that you see are the residues of ore deposits that did exist, not any indication of what is below.

SEELEY W. MUDD, Los Angeles, Calif.—There are many things, besides those the author has suggested, that should be considered before determining whether or not a prospect should be developed. The financier will be the deciding factor and he will be thinking only of the balance sheet at the end of a few years. The prospector of the future should be a trained man, especially when he works in the older districts, where there have been mines and where there are possibilities of other orebodies. There is a good deal, of course, in the natural gift, or aptitude. One man will see a set of conditions and not be able to describe them, but the picture will be so vivid in his own mind that he can compare it with similar things and can reach a good conclusion. Another man, seeing the same set of conditions, can explain all the details, describe the picture, and write down the reason for everything and what he proposes to do. The first man may reach as accurate a conclusion as the other, but he cannot make a good report.

Mark L. Requa, San Francisco, Calif.—The Peirce mine, in Arizona, is an example of a mine being found in a place where least expected. On the far end of the vein, there was an outcrop of white quartz, practically barren, and no outcrop of the vein where the ore was. Other mines of that character doubtless will be found in the future, but it will be by the combination of scientific knowledge and experience. This fact has been realized in the petroleum industry. Only a comparatively few years ago the geologist in the petroleum industry was looked upon with a good deal of suspicion. As recently as 1909 or 1910, the petroleum geologist was ignored by most of the men engaged in the industry; today the greatest part of the prospecting and developing are done under scientific direction. As the years go by, the old type of prospector will disappear; the new type of prospector will be a graduate of some mining college, where he studied geology and engineering. But I have a great deal of admiration for those pioneers who prospected and developed the mines of the West fifty or sixty years ago. They performed a great work, and they were a great race of men. They were virile, rugged, strong Americans, afraid of nothing, ready to tackle almost anything. In the onward march of the industry let us not forget them or their accomplishments.

Andrew C. Lawson.—In recent years geology is becoming recognized as the means to the finding of ore. Only a limited number of people can be good geologists. We can train men, but unless they have an aptitude for that work, they do not make really good geologists. If we train a man in the technique of geology, he will be useful to the mining industry in the finding of orebodies, provided that the people who want orebodies are willing to pay for his services; that condition does not exist at the present time. People willing to employ a good geologist today not only must pay him an attractive salary, but they must not expect that every time he makes a trial he will be successful. There will be a large percentage of failures. We should do our best to make the prospector a good geologist, and make the work of looking for mines attractive to him. The oil men are employing geologists. It is hard to keep students at Berkeley because of the large salaries paid by the oil people; and because of this the country is flooded with boys who call themselves experts. More than three-quarters of the boys are incompetent to do their work, but they have the confidence of boards of directors. The country is full of such men who in time will not make a good livelihood out of their geological work; their training is too narrow. The only way to get a man to meet the various conditions that would be encountered in looking for a mine is to make of him a good geologist.

J. C. Jones, Reno, Nevada.—The old-time prospectors went out and searched for the mines. They had no particular school training, but they were among the keenest of observers and learned their geology by experience; that is, in the mines. When they were confronted by a set of conditions they looked into them. If anything went wrong in one place, they moved to another; and each time they made a mistake they learned something until they became quite fair geologists. They might not have called rocks by the names we do, but they had the knowledge necessary to find the mines. While a sound training in geology is of great value, there are limitations to the science. In civil engineering, for instance, the engineer is used to dealing with exact mathematical facts; the geologist is not. He is at times confronted with certain elements of uncertainty even when facts have been fairly well established; for instance, when it comes to foretelling the value of the ore that one is going to find, or deciding whether or not there is any ore.

Every January, at the Mackay School of Mines we have a four-weeks' course, known as the prospector's short course. In that course, the prospectors get a series of lectures in which sufficient detail work is given so that they have an inkling of our present ideas. They get a little geology and mineralogy, a little first aid and hygiene, a little mining law, surveying, gas-engine work, assaying and chemistry, and while at the end of that time they are not college graduates, they have learned something which, when applied to their own observations, proves to be of great help to them.

Frank L. Sizer, San Francisco, Calif.—Without a doubt prospectors, as a class, have wasted a great deal of effort in the pursuit of impossible propositions, and the knowledge given by the short course of the Mackay School of Mines is the best thing that has been done to provide something useful for these men. It is the duty of every engineer, whenever he has the opportunity, to set these men right and give them helpful advice.

P. G. Spilsbury, Phoenix, Ariz.—During the last four years, we have examined about 700 properties, which have been classified by districts and also by possibilities—favorable, unfavorable, or worthless. Our scout has spent about 40 years as a practical prospector all over the West; my assistant is the best geologist on copper work I can get; I act more or less as a business engineer to determine, from their evidence, if the property can be made profitable. In this work we have secured the confidence of the prospector. He can do work that no engineer can do and can do it cheaper. He can go into places where the average young engineer will not go, and he will do things and stand hardships for which we cannot pay.

We have been trying to encourage these prospectors by giving them facts. We will go out with any man who has a reasonable proposition, and if his property is worthless, we try to show him why. We spend much time trying to show prospectors, who are on worthless ground, why they cannot make a mine. We have, in this way, been able to do some good. Fourteen or fifteen of these men come into our office regularly for advice. They bring in their propositions and we talk them over, not with the idea that we will buy them, but that we are there to help them. We do not give any course in geology, but our geologist explains to them, if necessary, what the requirements are for success. We help these men a little financially. We get much information about districts that could not be secured in any other way, and I believe this is one of the methods by which we can work with the prospector. We cannot make him highly educated as he has not the basis; but if we can secure his confidence, give him a square deal, and try to show him where he can do his best, we can encourage his work and possibly get results from it.

In a general way, we are classifying prospects in the way the author has explained, only not quite so scientifically. We have our 700 properties so thoroughly classified that there is hardly one district in the Southwest on which we have not already made three or four examinations. In the way we have been handling it, this has not been as expensive as might seem.

When a prospector brings in information regarding his property, we first determine if there is a faint possibility of its being of interest. If we decide affirmatively, we send out our field man to make a preliminary examination. This man is a very good observer and brings his information to the office. This is considered by my geologist and myself. If worthless, we determine why and drop it. If there is a possibility of making a mine out of it, the geologist makes another examination. If his word is unfavorable, we drop the proposition giving, in our report, the full reasons for our decision. If his report is favorable, my field man, geologist, and I visit the property and go over it thoroughly. The final report therefore is the opinion of a prospector, a geologist, and a business man. If the property proves hopeless, we always tell the owner why we consider it so and give him the benefit of our information.

Only about 60 per cent. of the prospectors whose properties we decide are worthless take the explanation in the right spirit. The prospectors who are getting to know us are gradually realizing that it is our job to help them and that in this way we will gain results. We believe in the work of the prospector and we are giving him all the help and encouragement possible.

Economic Application of Zonal Theory of Primary Deposition of Ores

By John Carter Anderson, Tucson, Ariz.

(New York Meeting, February, 1923)

The theory of a zonal deposition of ores from ascending solutions which were differentiation products of a cooling magma, first propounded by Spurr,[1] in a measure explained by Lindgren,[2] and recently re-emphasized by Kemp,[3] while perhaps accepted in principle by many mining geologists, so far has not been accepted as capable of practical application in the study and development of ore deposits.

From his observations, the writer is convinced that there are laws governing the primary deposition of ores which, when properly worked out, will be of greatest economic value; and that the application of this knowledge to the development of new mines will mark an advance in the mining industry similar to that following the working out and application of the theory of secondary enrichment.

That there has been a tendency to a zonal distribution in the formation of ore deposits, by which the ores of one metal have been precipitated from the mineralizing solutions at a greater distance from the source than those of another, cannot be disputed; and the nature of this zoning in a given deposit or district has been frequently described.[4] But that a selective precipitation of the ores of two or more metals, other than lead and zinc, in successive zones may have taken place in a single deposit within limits economically possible of exploration is not generally recognized or admitted.

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[1] J. E. Spurr: A Theory of Ore Deposition. *Econ. Geol.* (1907) **2,** 790.
J. E. Spurr: Theory of Ore Deposition. *Econ. Geol.* (1912) **7,** 485–492.

[2] W. Lindgren: "Mineral Deposits," McGraw-Hill Book Co.

[3] James F. Kemp: Zonal Distribution of Ores around Igneous Centers. *Econ. Geol.* (1921) **16,** 474–79.

[4] Paul Billingsley and J. A. Grimes: Ore Deposits of the Boulder Batholith of Montana. *Trans.* (1918) **58,** 284–361; J. E. Spurr, G. H. Garrey and C. N. Fenner: A Contact Metamorphic Ore Deposit. *Econ. Geol.* (1912) **7,** 444–484.

The interest of the writer in this subject was aroused, some years ago, in the study of a claim in the Greaterville district, Pima County, Ariz., where a small high-grade gold-quartz vein, certainly the root of one of the gold veins that fed the Greaterville placers changed within 60 ft. of the surface into a larger vein of solid galena. This changed, in the next 50 ft., into a still larger vein carrying about 30 per cent. lead as galena, 15 per cent. zinc as blende, and up to 2 per cent. copper as chalcopyrite, with erratic, but, on the average, low values in gold and silver. Since that time every mine visited has added some evidence that there is a direct relationship between the distance from the source of the mineralizing solutions, that is from the magmatic reservoir from which they ascended, and the type of mineral deposited; and that a change from the ores of one metal to the ores of another belonging to a deeper zone may take place in any deposit.

This study has also shown that a change from one predominant metal to another can, and often does, take place within the ordinary limits of mine development; and that disregard of this principle may often render extensive and costly attempts at mine development futile.

We are familiar with the general tendency of silver-lead deposits to become zincky in depth; it is this change that has condemned many promising prospects and mines. At the San Francisco meeting, September, 1922, during a discussion of methods for finding mines, it was suggested that more attention be paid the complex mines that we now have. About two years ago, the writer[5] advanced the theory that, pending the successful working out of a process for beneficiating the complex lead-zinc-copper ores that could be economically applied in a plant of moderate size and cost, many mines might be developed by gaining greater depth on such of those deposits as were structurally strong and possible of considerable extension in depth, in the course of which the complex zone would give place to one of commercial copper ores. That such a change has taken place has been recognized in Butte and elsewhere but few have admitted, in their consideration of prospects and partly developed mines, that such a change was possible or had occurred as the consequence of a general law. That such a change is possible, and often probable, and that the deposits of copper ores found in depth may be of much greater size than the silver-lead and lead-zinc veins above is the belief of the writer.

Briefly stated, starting from the ultimate source of the mineralizing solutions in the molten magma of an intrusive batholith, from which they are essentially differentiation products, these solutions, which contain varying amounts of several metals, as they gain distance from the source tend to deposit their load in successive zones, in which the ores of one metal lie above or beyond those of another in more or less regular order. There may be an overlapping of these zones, producing the familiar deposits of complex ores lying below many of our silver-lead veins, but generally the bulk of the ores of one metal will lie above or below those of another. Probably, this zoning is more the result of the relief of temperature and pressure as the solutions travel farther from the source, successively cooling from the critical temperature of formation of one mineral to that of another, than anything else. Varying wall-rock conditions may aid or retard precipitation and the nature of the opening may hasten or hinder the circulation and thus increase or decrease the range of travel while the solution drops from the critical temperature of one metal to that of another; all of these factors may affect the vertical or lateral range of the several zones but almost everywhere some evidence of zoning can be found. No deposit can be said to have been bottomed so long as there is a channel through which circulation may have risen, and until the presence of high-temperature minerals characteristic of the deepest zone and the near approach to the main mass of the now cold magma proves that the deposit has been traced to its source. Structural conditions favoring a rapid travel of the solutions may have permitted the formation of a practically barren zone between zones of profitable deposition, which may have been explored unsuccessfully in many mines without proving the real bottom of commercial mineralization of some type.

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[5] The Future of Mining in Southwestern Arizona. *Mining and Oil Bulletin* (August, 1920).

Also, a quartz vein may, and at some depth almost certainly will, pinch out to nothing at or near the contact of an intrusive rock, below which contact ores of increasing size and belonging to a deeper zone may have been deposited. This connection has not been recognized because so many of the typical quartz veins, being located in districts that have not been deeply attacked by erosion since the period of mineralization, have been worked only down to the point where they narrow or diminish in grade and have not been followed far enough to establish this connection with contact deposits of the deeper types. Yet in some places erosion has exposed typical quartz veins to sufficient depth to permit this genetic relation to be established, and a correlation of the different types of deposits within one general region will often illustrate it.

The general succession observed shows that in dealing with deposits of the more common metals, the primary oxides of iron (chiefly magnetite, but sometimes hematite) were the first ores to form and hence are representative of the deeper zone. With and above them may be found garnet, epidote, pyroxene and similar high-temperature rock-forming minerals, associated with and followed in an ascending order by sulfides of copper and iron, principally bornite and chalcopyrite; pyrhotite.

arsenopyrite and pyrite, which may or may not carry gold and (or) silver; zinc blende; galena; silver sulfides and gold. Pyrite is apparently a stable mineral over a wide range of temperature and manganese belongs relatively high in the deposit, often occurring with the gold and silver. There is a strong presumption that the low-grade cupriferous pyrite, from which large bodies of commercial ore have been built by processes of secondary enrichment, does not represent the bottom of commercial mineralization but lies above the zone of maximum deposition of the primary sulfides of copper.

It is not the purpose here to go into details of the development of the theory, but to give an illustration of its practical application and demonstration, in the hope that this may stimulate a discussion which will lead to a more perfect knowledge of the laws that govern the primary deposition of ores. This illustration is drawn from the study of properties in the Pioneer district, Pinal County, Ariz., in which the Magma Copper Co. has the principal mine. The development of the Magma has always been of great interest to the writer because it has given increasing proof of the soundness of this theory and because from an inconspicuous outcrop of a fissure vein, showing some silver ores, it has been developed into a copper mine of considerable importance, which is increasing in size and value with depth.

To present as clear a picture of the deposit as possible the following outline of the geology is given. The vein outcrops on the steep west slope of the range, just east of the town of Superior, which shows a series of Paleozoic sedimentary rocks invaded at or near the base of the Troy quartzite by a great sill of diabase, and covered at an old erosional surface by patches of the Whitetail conglomerate, which is in turn covered by a thick flow of dacite. The sedimentary rocks of the region, which have been described by Ransome,[6] are represented by the Troy quartzite, about 400 ft. thick, overlain by the Devonian limestone, 350 ft. thick and about 1000 ft. of Carboniferous limestone, with minor interbedded shales. Fragments of the thin-bedded Mescal limestone, which normally underlies the Troy quartzite, are found floating in the diabase sill at different levels in parts of the district. This formation has not been reached in the Magma mine, but presumably will be found at some depth in or below the diabase sill. The strike of the formations is north-south with a dip of 30° or more to the east.

The veins of the district are fault fissures cutting the diabase, quartzite, and limestone with a general east-west strike and varying dip. At or near the top of the quartzite, the fissuring often splits, as many as five fingering veins spreading from one main fissure. Some of them extend to the present top of the limestones, where they were exposed by erosion and later covered by the Whitetail conglomerate and the dacite flow.

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[6] U. S. Geol. Surv. *Prof. Paper* 98 K.

The fault fissures showing mineralization generally have an intrusive porphyry dike, probably a diorite porphyry, filling the fissure up to or perhaps a little above the quartzite. In the best-developed mine of the district, the Magma, there is a more or less complete replacement of the dike by ore, the walls of the vein and of the dike being co-incident, and there is an evident genetic connection between the dike and the ore.

The earlier fissures have been cut by a series of post-mineral faults of great displacement, which strike a little west of north, one of which, at the Magma, has dropped the dacite flow to the west bringing it in contact with the quartzite; the throw, therefore, must have exceeded 1000 ft. These faults can be followed to the southeast beyond Ray and have been largely responsible for blocking out the range as it now appears. It is this series of faults that tilted the formations to their present dip.

When describing this deposit, the author has asked several engineers and geologists what difference it made if this tilting was premineral or post-mineral, but has not received an answer that showed an understanding of its economic importance. Yet when taken into consideration with the fact that there has been a distinct zonal distribution of the ores in the formation of the deposit, and the established fact that the size and grade of the deposit have increased remarkably in depth, so that the development on the lower levels has found more than 10 tons of ore per vertical foot to 1 ton of ore per vertical foot in the upper levels, and that this ratio is constantly increasing, it is of utmost economic importance.

That the formations of the region were practically level at the beginning of the period of mineralization and have since been tilted was recognized by Spurr in his report on properties in the Ray District in 1909, but to the author's knowledge this fact has not been recognized or taken into consideration elsewhere except perhaps by the Magma. It is best proved by the fact that bedded veins, which were formed at or near the contact between the quartzite and the overlying Devonian limestone, as the L. S. & A. contact vein near the Magma fissure, are completely oxidized for great depths below the present water level, while the ores in the fissure below that horizon are sulfides; and by a study of the relative displacements along parallel faults of the post-mineral series, and their resultant effect on the dip of the formations in the various fault blocks. A glance at a cross section of the Ray orebody shows how it has been blocked out since enrichment by faults of this series.

If, then, we are dealing with a deposit in which there has been a distinct vertical zoning in the primary mineralization and which was formed at a time when the overlying formations were about level, we can expect a general parallelism between the zones of mineralization and the overlying formations. To get depth in the deposit, then, we must measure not from the present surface but from a given horizon, say the top of the quartzite. For purposes of illustration, we will assume that manganese and lead-silver ores, with some gold and a little overlapping copper, were deposited in the limestones above; that the porphyry dike that intruded into the fissure reached about to the top of the quartzite, being of small width at the top and increasing in size in depth; that below the quartzite there was secondary enrichment and zinc ores carrying silver; that at 500 ft. below the quartzite we have bottomed all of the zinc and silver minerals except a little silver occurring with the copper ores, and have a vein that will average 8 ft. wide and about 5 per cent. in copper; and that below that point there is a steady increase in the size and value of the vein until at the 2000-ft. level it averages over 20 ft. in width and above 8 per cent. in copper.

If, after deposition, oxidation of the upper part of the vein and the formation of a zone of secondary enrichment, the whole formation is tilted in the direction of the strike of the vein, so that it stands with a dip of about 30° to the east, we find that a drift or tunnel started at a given level in the deposit and run to the east along the vein, will for each two feet of advance rise about one foot in the deposit. Thus in running 2000 ft. east from a station in the shaft at say 1500 ft. in depth, the drift actually rises in the deposit so that at its eastern end it will be at a horizon in the deposit corresponding to that cut by the shaft at 500 ft. in depth. Therefore, in an east drift in such a deposit it is to be expected that the vein will gradually decrease in size and value, and perhaps pass from commercial copper ore into lean copper ore and on into zinc ore, while a west drift should show an increasing width and value of copper ores.

Likewise, if we have developed a shoot of good copper ore on the 1500-ft. level which is 1000 ft. long, beyond which the vein decreases in size and value, and we then drive another level on the vein 500 ft. lower down, we should find that the vein will have a much greater length of good ore before it tapers down. A decreasing mineralization and size of vein on any level toward the east is not, then, an indication of the pinching out of the vein in that direction, but of the persistence of mineralization, which on a lower level should be of commercial size and value. So each deeper level, up to the full limit of the extreme length of the mineralization, should show an increasingly long oreshoot and develop an increasing tonnage of ore per vertical foot.

All of these conditions are fulfilled in the development of the Magma to the 2000-ft. level, and as diamond-drill holes have shown the persistence of ore below the 3000-ft. level, each succeeding level may be expected to demonstrate them more fully.

The primary zoning in the deposit is also illustrated in the development of a zinc-silver orebody on the 1200- and 1300-ft. levels. On the

1200-ft. level this orebody carried relatively high values in silver, a little lead and some zinc. Between the 1200-ft. and the 1300-ft. levels, the silver values constantly dropped and a trace of copper, as chalcopyrite, appeared. A raise connects the 1300-ft. and the 1400-ft. levels at this point and the assay map of this raise, which was sampled in 5-ft. sections, shows the changing character of an ore deposit in depth. The following figures are from memory, but are substantially correct. At the 1300-ft. level the vein assayed 15.1 per cent. zinc, 12.8 oz. silver, and 1.1 per cent. copper. Each lower 5-ft. sample ran higher in copper and lower in zinc, until at the 1400-ft. level the vein averaged 1 per cent. zinc, low silver, and 15 per cent. copper. Below that point there is no zinc of economic interest in the deposit, but the lower levels, as they advance to the east and so get higher in the formation and in the deposit, begin to show zinc; it is hoped by the management that as these levels advance still farther into the sedimentary rocks, they will discover some of the manganese and silver ores found in the upper limestones.

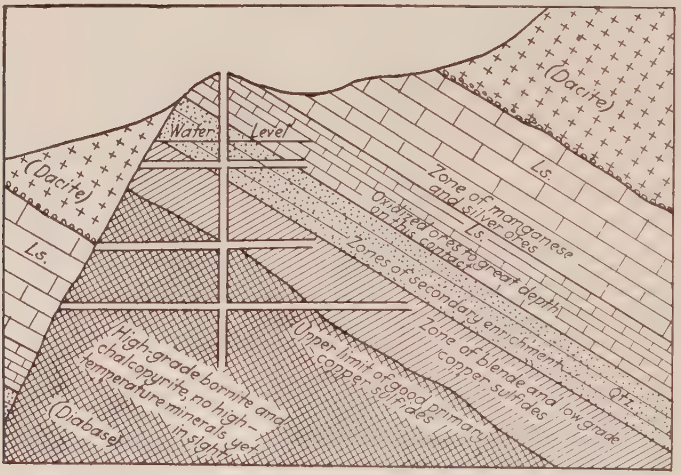
(Dacite)
Water Level
(Dacite)
Ls.
Ls.
Zone of manganese and silver ores
Ls.
Oxidized ores to great depth on this contact
Zones of secondary enrichment
Qtz.
Zone of blende and low grade copper sulfides
Upper limit of good primary copper sulfides
High-grade bornite and chalcopyrite, no high-temperature minerals in sight yet
(Diabase)

Fig. 1.—Zonal distribution of primary mineralization and effect on development work of post-mineral tilting of formations: each deeper level having a greater length in good primary sulfides. Drifts to east lose depth; to west gain depth in deposit.

We can accept it, therefore, as a proved fact that there was a primary zoning in the formation of this vein, wherein the maximum deposition of copper was deep in the deposit, and that as distance from the source increased by ascent of the solutions in the formation there was a decrease in copper content and a change from copper to other metals, with a constant decrease in the size of the deposit; and that manganese was one of the metals that traveled highest in the deposit. So the presence of manganese in the outcrop of another vein of the region should indicate a certain relative height above the point of maximum deposition of copper ores. Fig. 1 will illustrate in a general way these conditions as shown by the development of the Magma; this illustration is published with the permission of the Magma Copper Co.

Following the discovery of good ore by this company, development work in the district was greatly stimulated; on four properties a large amount of development work was done, at least three of which show superficial conditions similar to those at the Magma. That is, an east-west porphyry-filled fissure, showing some manganese, small amounts of precious metals, and some copper carbonates. The fourth covered a contact vein showing copper carbonates at the quartzite-limestone contact, which is faulted by one or more east-west fissures.

In Fig. 2 are shown the conditions at the first of these properties. A tunnel was driven on the vein to the east for 1490 ft., gaining 600 ft. or more of backs at the end. Various outcrops showed manganese and the porphyry dike that fills the fissure, where crosscut in the tunnel, showed low values in gold and silver. If it takes depth in the deposit to reach the point of commercial deposition of copper ores, and the manganese silver ores are at a comparatively high horizon above that, this tunnel failed as a test of the deposit, for it was relatively 700 ft. higher in the deposit at the end than at the portal. The diamond drilling from within this tunnel also did not reach a great enough depth to be conclusive, or indeed to be valuable.

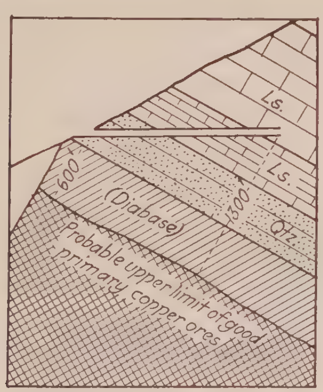

Fig. 2.—Tunnel lost depth in deposit with each foot of advance under mountain.

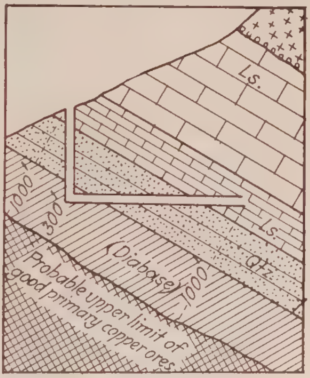

Fig. 3.—All depth gained in sinking was lost in drifting.

The conditions at the second of these properties are illustrated by Fig. 3. A vertical shaft, starting in the limestone 100 ft. or more above the quartzite contact, was sunk 750 ft., reaching into the diabase below the quartzite. This by no means reached the relative level of the zone of good copper ores in the Magma, and the 1500-ft. drift to the east from the shaft rose constantly higher in the formation until at the end it was at or above the level of the collar of the shaft. This tunnel, too, failed of its object of deep development in the vein.

At the third property, as illustrated by Fig. 4, a tunnel was driven over 1400 ft. along the strike of the vein to the east, finally reaching a horizon in the limestones about 700 ft. higher than the portal of the tunnel, and consequently that much higher above the chance for good ore in quantity, and so high that a 500-ft. winze sunk on the vein near the end of the tunnel was still above the portal in actual position in the vein. This work gained considerable depth under the present surface, but lost depth in the deposit and so failed to prove the possibilities of the deposit in depth.

At the fourth property, as illustrated by Fig. 5, an inclined shaft was sunk on the contact vein and a large amount of lateral work done at several levels, showing a fair amount of oxidized ores. Unlike the other properties, this work did not lose depth in the deposit neither did it gain any, being relatively no deeper at the 800-ft. level than it was at the surface.

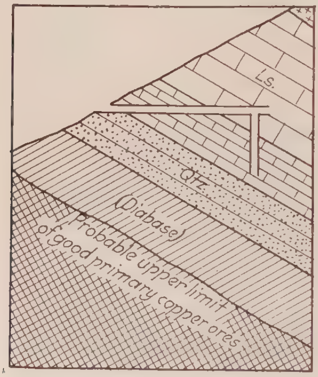

FIG. 4.—TUNNEL LOST MORE DEPTH IN DEPOSIT THAN WAS REGAINED BY A WINZE.

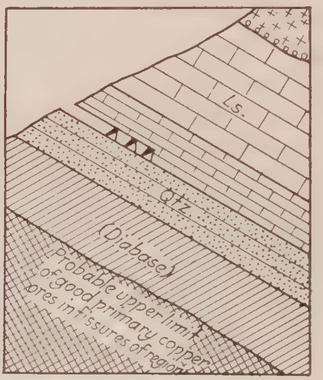
Ls.
Qtz.
(Diabase)
Probable upper limit of good primary copper ores in fissures of region

FIG. 5.—AN INCLINED SHAFT SUNK ON OXIDIZED ORES BEDDING OUT AT THE QUARTZITE-LIME CONTACT NEITHER GAINED NOR LOST DEPTH IN SINKING.

These illustrations are taken from a large number of cases that have been observed, where similar conditions if not fully proved are strongly indicated; because they show the economic loss resulting from a failure to understand and apply the theory of zonal deposition of ores in development work and the possibilities that its application opens for the discovery of new mines.

There are big mines yet to be found, and if we but work out a fuller knowledge of the laws that govern the formation of ore deposits and perfect this theory, as I believe we shall in the next few years, we shall have an invaluable aid in all exploration and development work.

DISCUSSION

Frank L. Nason, West Haven, Conn. (written discussion).—As I understand this paper, the author describes zonal distribution and shows just where to sink a shaft and to drive crosscuts in order to develop the ore which he believes to exist. Given an ore cropping, he deduces the zone in which the outcrop lies, then he aims for the inner zone where he believes the richest ore to be. Is not this exactly what the ordinary prospector would do without the zonal theory?

In order to demonstrate the economic application he should locate a series of sedimentaries and then seek a magma within range of its temperature and pressure, and then hunt for the zone, or zones. This is what has been attempted in a copper mine in Mexico. The mines were pockety and were giving out. About $30,000 was the cost of the reasoning and the blank holes were stopped in the "laccolith."

J. E. Spurr, New York, N. Y. (written discussion).—Undoubtedly, the zonal theory of primary ore deposition will lend itself to practical application, as it is studied in various regions and its local complexities and variations are discussed. Each ore deposit must be studied by itself. To conclude that in every mining district there is a column of different ores, as stated in the typical and theoretical column and the complete sequence, would result in pursuing in many—probably most—cases an economic and geologic will-o'-the-wisp. But the understanding of the nature of ore deposition contained in the zonal theory will certainly find here and there a practical application in prospecting and mining. In one important district, where barren quartz and barite veins occur at the surface from the knowledge of the local primary ore column that I have worked out, it would be well worth while to crosscut from certain deep mine workings near by, to get under these outcrops, with a depth of 4000 or 5000 ft. that could be thus gained, in the belief that at this depth the barren veins may have changed to lead-silver ores.

The author calls attention to a sound geological principle that should be kept in mind—the possibility of the deformation of ore zones after ore deposition. At Matehuala, San Luis Potosi, Mexico, I have called[7] attention to a case of deformation by faulting, whereby the lead-silver zone is brought to the same horizon as the copper zone, normally several thousands of feet lower. The author emphasizes the case, which theoretically must occur, of deformation by tilting of the whole zone whereby a vertical succession of deposition of different ores is brought, not perpendicular to the earth's surface, but diagonal with it; and in extreme but readily conceivable cases, horizontal, or parallel to it. The realization of the possible complications of original vertical ore zones by faulting, tilting and folding, will lead to a re-examination of certain districts as to the distribution of different metals, and its significance. The case which the author takes is that of tilting, resulting in a situation where the gaining of depth in the primary ore zones may be accomplished either by vertical sinking or by the certainly easier method of horizontal drifting, if done in the right direction—best, of course, by incline sinking, if done in the right direction. He shows that in such cases horizontal drifting, or incline sinking, in the wrong directions may be of little or no value as development work.

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[7] *Econ. Geol.* (1912) **7**, 444–484.

James F. Kemp, New York, N. Y.—This paper brings out two points, each of great importance: the first is the zonal distribution of metals in veins, a serial order which his observations corroborate; the second is the tilting of a vein containing sulfides, after it has been formed and has been long exposed to weathering and secondary enrichment; and had thus, by the tilting, had the relation of the secondary products to the ground-water level, entirely reorganized. Primary sulfides might thus be brought above the water level at one end; while products of weathering (the oxidized zone) might be brought below it at the other.

In originally bringing forward the zonal distribution of the metals, J. E. Spurr carried one step farther the general results of our studies of the influence of igneous rocks, a proper appreciation of which started about the beginning of this century or the end of the last, and reached its fuller development in recent years. It has led us to fix our attention on a localized source of heat in a mass of intrusive igneous rock, from which solutions have migrated outwards and upwards to produce the veins. The final result is not essentially different, whether we believe the solutions are derived from rainwater that has descended below the surface, and in its wanderings toward this heated mass has become charged with components of the ores and gangue; or whether we believe them to originate in gases and vapors dissolved in the magma itself and to have been pried when this mass solidified and crystallized into anhydrous minerals. In each case, possessing high temperature and pressures, they are driven outwards, dropping gradually the series of dissolved metals in zones.

Waldemar Lindgren early emphasized the physical conditions under which the minerals would be deposited. J. E. Spurr, in 1907, ranged them in the specific zones that are perhaps familiar to most of the audience. His observations were that contact zones and pegmatites would be the nearest to the intrusive mass; that next would be a coarsely crystalline gold quartz; then copper ores; then zinc ores, passing gradually into lead ores; then high-grade silver minerals, with tellurides of gold, and minerals that would be formed at relatively low temperatures and pressures; and finally a vein filling of barren gangue. He had seen a number of instances of this sort, and had realized the importance of distribution.

Since then, others have found the same general distribution. Many members of the Institute will recall the paper of Paul Billingsley and J. Alden Grimes[8] on the "Ore Deposits of the Boulder Batholith" in which, by many sections of mines, they showed a similar distribution of metals, but they differed in one important point from the succession given by Mr. Spurr. His deepest zone of ores, aside from the contact zones and pegmatites, was a coarse gold-bearing quartz. Billingsley and Grimes start with the copper zone, pass into the zinc and lead and high-grade silver ores, but end with a gold-bearing quartz at the top of the series. This relationship also seems to be the one which prevails in the Magma mines. I do not know how to reconcile these two views, except that perhaps quartz veins analogous to pegmatite veins sometimes carry some gold, and that Mr. Spurr may have had veins of that sort in mind.

In the application of the zonal idea we should restrict ourselves to those areas where a number of metals are involved in the vein filling. We have districts where one metal is predominant. Apparently the ore-bearing solutions coming up from the depths in the gold-quartz region, in the western slopes of the Sierras, have carried chiefly the yellow metal, together with the sulfide of iron, in the form usually of pyrites.

We would not venture to infer, from the Billingsley and Grimes arrangement of the zones, that these California quartz veins would pass in depth to lead, or first perhaps to high-grade silver, then to lead, then to zinc and to copper. The veins have been mined to such great depth in the Sierras already, and have shown only gold as the valuable metal in them, that we would not anticipate the regular zonal series there. One would infer that originally the solutions had not the different metals in them. Other instances might be cited, but time will not suffice. Therefore, the zonal arrangement is used mainly in regions of complex solutions which, as I think, come from magmatic sources, but which others attribute to meteoric.

The second point, relating to the subsequent history of the vein and its wall rocks, is equally important. I was greatly impressed with the way in which the author has developed this theory and the importance it would have for one prospecting such a vein as the Magma in forecasting what he might anticipate in his drifts. Evidently, in the illustration where the development failed to find useful ores, the lack of them could be well explained by restoring the tilted strata to their horizontal condition, and then letting them assume their present position. One could then foresee what one could anticipate ahead of exploration. One would sometimes drift in exactly the opposite direction to the one actually taken, if guided by a conception of the geological history of the district.

The finding of oxidized ores below the permanent water level today is always significant and makes one reason back to the past climatic conditions. A striking illustration of that is the soft ore in the Gogebic Range in northern Michigan, where we have these products of oxidation down to more than 2000 ft. in depth, and accurately exposed in the mines, yet they must at one time have stood above the permanent water level, or else, our well-established views of the changes from the original iron carbonates would show they could not have been formed. We hark back, therefore, to the old Keweenawan period that was arid, as shown by its red sediments. For a long time in that period, the ground water must have stood at great depth and the oxidation must have proceeded to more than 2000 ft. below the present surface. The Gogebic ores furnish a good illustration of the second principle which the author advocates.

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[8] *Trans.* (1918) **58,** 284.

George W. Roddewig, Butte, Mont.—In the upper levels of the Elm Orlu, in Butte, there is a manganese-silver ore, the manganese that carries the silver being mainly in the carbonate form. Whenever the siliceous form of manganese appears, the silver becomes low. As we proceed downward, the vein filling becomes manganese silver and zinc silver, zinc being in the form of sphalerite. Still farther down, the zinc is frequently associated with iron pyrites and the copper seams come in. Several exposures in some of the lower levels contain copper and zinc in the same vein, to such an extent that it is difficult to tell when the ore is mined whether it should be sent out for its zinc or its copper. There seems to be a difference in the value of the ore from one side to the other of the vein. The zinc predominates, for instance, on one wall and the copper on the other; they grade together to a dividing point somewhere in the center of the vein.

L. D. Frinck, Butte, Mont.—Is it usual to have a zinc ore on one wall and a copper ore on the other wall at the same level?

James F. Kemp.—It is most unusual. I think, however, as mining goes deeper probably the copper will more and more predominate and the zinc will gradually die out.

George W. Roddewig.—It happens that the copper predominates on the wall nearer the copper mines, but it is quite a distance from the copper mines at this particular point. One thing should be taken into consideration: there are not sufficient openings of that kind at the present time to base a theory on what we have found.

J. C. Jones, Reno, Nev.—In working out the conditions under which we may expect to find ore, the question arises as to whether this zone structure is entirely the result of primary mineralization of the vein, or whether it is partly the result of superimposing the zonal secondary enrichment. It seems to me that there is a certain amount of zoning from secondary enrichment in connection with the possible primary zoning.

John Carter Anderson.—In regard to the zone of secondary enrichment, a certain amount of low-grade copper ore higher in the deposit than the main body did build up an enriched zone below the level to which oxidation reached. But there is a distinct change below that, in which there is only low-grade copper or zinc before good primary ore is reached; and the deeper you go in the deposit the higher is the grade of the ore. Just before the fault was reached on the west end of the 2000-ft. level, the veins averaged 20 per cent. across its full width. This would be the deepest point in the deposit yet opened in the mine. So while there has been secondary enrichment, it was active only on that relatively limited part of the copper that was higher in the deposit.

J. C. Jones.—Geologists have not emphasized enough the fact that secondary enrichment does not necessarily depend on the water level. The secondary enrichment of copper may take place far above the water level; on the other hand, if there is no particular reducing element, it may take place far below the water level. That fact is particularly well brought out in a mine in Nevada, where all secondary enrichment is at a depth of about 150 ft. while the water level is around 350 ft. That does not necessarily mean that there is no secondary enrichment at those lower depths, because there is, but the secondary enrichment is closely connected with bodies of chlorite and pyrrhotite, ferrous-iron minerals, which reduce the solutions sufficiently for them to deposit their copper on the iron sulfides in the form of chalcocite. The water that percolates through these chlorite bodies does not carry a trace of copper and is alkaline; it is no longer acid as are the surface waters.

Arthur Thacher, St. Louis, Mo.—Of course, the water level is not constant, in some cases we go through large bodies of sulfide ore and then meet large bodies of carbonate ore below.

John Carter Anderson.—In regard to these primary ores the bornite and chalcopyrite ores, below the zone of secondary enrichment which was not great, were pronounced by Mr. Graton as absolutely primary. I do not think there is any question as to the primary nature of the richer ores in depth.

Emerson M. Butterworth, Los Angeles, Calif.—Has any ore been found to the left of the fault?

John Carter Anderson.—No, the drop west of the fault carried the oil still deeper. The drilling to find it is being done from the 1800-ft. level.

Emerson M. Butterworth.—The ore west of the fault does not come to the surface?

John Carter Anderson.—No; the throw of the fault is so great that the vein would not outcrop, but they are drilling for it. They will probably have to go 2500 ft. deep before they get a real show of ore west of the fault.

Magnetic Methods for Exploration and Geologic Work*

By W. O. Hotchkiss,† Madison, Wis.

(New York Meeting, February, 1923)

Rock exposures are usually a very small part of the surface area in any mining district and the prospector and geologist must base their deductions as to the area, extent, and structure of various formations upon exposures that seldom form 5 per cent. of the area studied. Most often, outcrops form less than 1 per cent. of an area, so that exploration for mineralized contacts or mineral-bearing formations necessitates drilling, test pitting or tunneling, shaft sinking, and other underground work. In these days of high costs any method that will give definite facts regarding rocks obscured by soil or other material should be used to the greatest possible extent.

In the early days of the Lake Superior iron districts, mines were located in the few places where ore was present in outcrops. Later, larger orebodies have been found, in depressions, obscured by soil and other foreign material. Mineralized rocks are oftentimes more weathered, softer, and more easily eroded than unmineralized rocks. It is quite natural, therefore, to expect that such orebodies (particularly where large enough to influence erosion in former physiographic conditions) should lie in depressions and be obscured by later debris. It is not difficult to believe that the history of the development of Lake Superior iron mining may be duplicated in other mining districts, that even larger orebodies than those first mined may be found in areas where the rock exposures, which led to original discoveries, are few or lacking.

The chief properties of rocks that can manifest themselves through an obscuring cover are magnetism, electrical effects, radio-activity, gravity, and sound transmission. All of these have been used, to a small extent, to tell us a few things about what lies below the surface that we could not otherwise determine, but none has been used so long and so widely as the magnetic properties. The magnetic properties of the earth and of many of its rocks have been the subject of extensive research. A great mass of definitely known facts is available as a basis for the use of magnetic methods for prospecting and geologic field work.

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† State Geologist.

MAGNETIC FIELD OF EARTH AS AFFECTED BY MAGNETITE IN ROCKS

That the earth is a big magnet has long been known. The direction of the lines of force and the strength of its field have been mapped, in a general way, over the entire earth's surface. The action of this magnetic field of the earth is exactly like that of a magnet in the laboratory. Most of us have laid a cardboard covered with iron filings over a magnet, and tapped it to have the filings show the directions of the lines of force. We then put a piece of iron in the field and again tapped the cardboard; the lines of force changed their directions and crowded into the small piece of iron. If we replaced the small piece of iron with a piece of bismuth and then tapped the cardboard, the lines of force tended to avoid the bismuth. In this way we found that some substances are more permeable to magnetic lines of force and some less permeable, than air. If we had tried all manner of substances we would have found that iron is the most permeable; that magnetite is next, with approximately one-half the permeability of iron; and that practically all other substances are relatively very slightly permeable—some a little more, and some a little less, than air.

Magnetite is the one common mineral that affects the magnetic field of the earth to any marked degree. Other iron minerals usually affect the earth's field approximately only $1/100$ as strongly as an equal amount of magnetite. Minerals like quartz and dolomite are slightly less permeable than air (which is taken as unity) a minute fraction of 1 per cent. but with exceedingly refined methods even such small differences can be detected in the field.

Far too little is known of the distribution of magnetite in rocks. By ordinary field methods we are able to detect small variations in magnetite content, consequently we are concerned with many relatively minor variations that only close detailed magnetic field work will determine. We do know, however, that all the ordinary sediments and igneous rocks vary widely in the amount of magnetite they contain. We know that in present beach sands some beds contain much magnetite and some none. We know that sandstone beds of all ages contain moderately magnetitic beds interspersed with those free from magnetite. We know that shales and limestones may contain magnetitic beds and non-magnetitic beds. We know also that non-magnetitic beds differ in magnetic permeability sufficiently to influence the earth's magnetic field enough for its detection by delicate measurements. In the igneous rocks, a segregation of magnetite at contacts is very common; in general, no two contiguous masses are of identical magnetic effect.

In practical exploration and geologic field work we are interested in determining the positions of tabular formations, such as sedimentary beds or igneous flows, also intrusive igneous contacts, which have bearing on the occurrence of orebodies and geologic structure. The question then arises: what quantity of magnetite must be present in a given situation to enable us to follow it and determine its location beneath the obscuring material? It is not easy to answer this question categorically, because we know too little about the effect of magnetite on the earth's field. Whether magnetite acts like so much soft iron and therefore its magnetism is wholly induced by the earth's field; whether it acts like steel and possesses permanent magnetism (which may be either opposed to or coincident with the earth's field); or whether the magnetite in a rock may in part behave in one way and in part the other way, are questions we cannot answer until much research work has been done.

The author has found but two determinations of the effect of magnetite on weak fields like the earth's.[1] Tests of two crystals of magnetite showed that the magnetism induced by a field of 0.6 gauss (about the strength of the earth's field) in one crystal was 85 times and in the other 285 times as strong as the inducing field. If a rock contained 1 per cent. of magnetite with the smaller of the above values, the field near its surface would be nearly twice as strong as the normal field of the earth.

The second factor that affects our ability to follow a magnetic bed or contact is the law of radiation of energy. It is commonly stated that energy varies with the inverse square of the distance; this applies only to energy radiating from a point. Energy radiating from a line (such as the upper edge of a thin, vertical, magnetic bed of rocks) varies inversely with the first power of the distance. If the magnet pole is an infinite plane the lines of force would be parallel and there would be no weakening of the force with distance.

Obviously the simplest and most common case that interests the geologist or prospector is the magnetic bed or contact. Such a situation is nearest to the second case cited. The actual magnetic pole would be intermediate between a line and a plane, so that we can say that the magnetic effect is inversely proportional to the distance. Thus, if a very thin magnetic bed or contact that at a distance of 1 ft. gave a magnetic field of 40 per cent. greater than the normal earth's field, at a distance of 40 ft. (through 40 ft. of cover) it would give an effect equal to 1 per cent. greater than the normal field. This difference of 1 per cent. in the earth's field is easily detected by the ordinary dip needle.

The third important factor influencing the magnetic effect of the magnetite content of rocks is the attitude of the beds. Horizontal beds are too nearly perpendicular to the direction of the earth's field to give strong effects. Vertical or steeply inclined beds have the greatest effect. Other factors will occur to any one who has worked on magnetic rocks, such as evenness or unevenness of distribution of magnetite in the rocks, temperature effects, magnetic storms and permanent magnetism as contrasted with that induced by the earth's field, but these are details that must be worked out in every situation where magnetic methods are applied.

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[1] Pierre Wess: *L'Eclairage Electrique* (1896) **7,** 487–508.

Amount of Magnetite in Rocks

On this subject we know far too little. Probably the best source of information concerning igneous rocks is Professional Paper 99 of the U. S. Geological Survey, in which the percentages of minerals in various rocks, computed from the chemical analyses, are given. The magnetite content thus computed may be different from that actually in the rock, but it is the best information we have and probably gives a fair approximation to the facts in most cases. From this source, Table 1 has been prepared.

A study of this table indicates that, in general, the basic rocks contain more magnetite than the acid rocks, but that any kind of rock, from acid to basic, may be entirely lacking in magnetite. Professional Paper 99, from which these values were taken rather at random, shows the truth of this generalization more strongly than Table 1. It must be remembered that these figures represent the sample analyzed, and not the average of the rock.

The calculated magnetite content of the rocks in various mining districts can be found in the paper referred to. For the rocks in the Butte district, the variation is from no magnetite (in the aplite from the Nettie mine, and the Red Mountain rhyolite) to 3.25 per cent. of magnetite in the quartz monzonite from the Gagnon mine. In the Keweenaw Point copper district of Michigan, the magnetite varies from nothing, in the Mt. Houghton felsite, to 8.12 per cent., in the diabase porphyrite of the Little Montreal River. In Cripple Creek rocks, the calculated magnetite content varies from nothing, in the biotite trachyte of the Portland mine, and the phonolite of Mitre Peak, to 7.42 per cent., in the trachyte dolerite of the Isabella dike. In the Globe district, it varies from 0.46 per cent. in the granitite at Schultze Ranch, to 6.03 per cent. in the quartz diorite near Pinal Peak.

A number of rocks in Great Britain, according to Rücker,[2] give changes in the magnetic field near the rock surface varying from nothing to over 9 per cent., this variation will show dip-needle readings of 9° or

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[2] On the Magnetic Permeability of Rocks. *Proc.* Roy. Soc. (1890) **58,** 521.

TABLE 1

Kind of Rock	Calculated Per Cent. Magnetite in Rock	Per Cent. Silica in Rock	Locality of Rock
Granite	0.00	96.16	Cumberland, England
Muscovite granite	1.39	75.51	Tonopah, Nev.
Porphyry	0.23	79.69	Tasmania
Porphyry	0.93	79.75	Blowing Rock, N. C.
Granite	0.23	74.40	Pikes Peak, Colo.
Granite	3.71	75.62	Sudbury, Ont.
Granite	1.39	76.41	Adirondacks, N. Y.
Granite	0.00	74.87	Brookville, Md.
Granite	5.57	74.62	Waushara, Wis.
Syenite	3.48	64.64	Little Belt Mt., Mont.
Quartz monzonite	3.25	65.91	Rimini, Mont.
Quartz monzonite porphyry	0.46	68.14	Swan City, Colo.
Trachyte	0.23	66.20	Eolian Islands
Trachyte	3.94	63.37	Carapook, Victoria
Andesite	0.93	64.82	San Francisco Mt., Ariz.
Andesite	3.94	63.03	Falls Creek, Colo.
Phonolite	7.42	65.52	Campbell Islands
Diorite	1.86	56.90	Skagit Range, B. C.
Diorite	3.94	55.18	Tuolomme Riv. Calif.
Diorite	6.03	53.60	Lost Trail Creek, Colo.
Augite diorite	17.86	44.57	Ural Mountains
Basalt	0.93	51.82	Scotch Plains, N. J.
Basalt	6.96	48.35	Colfax Co., N. Mex.
Basalt	3.71	50.66	Oroville, Calif.
Basalt	11.60	49.45	Hawaii
Gabbro	0.00	42.10	James River Gap, Va.
Gabbro	3.02	50.99	Nelson Co., Va.
Olivine gabbro	5.34	43.41	Tuolomme Co., Calif.
Orthoclase gabbro	9.74	45.65	Duluth, Minn.

more from the normal reading of the instrument if the depth of cover is not great.

These instances illustrate the variation in magnetic content of the igneous rocks; they tell nothing of how the magnetite in a single igneous mass may vary from point to point nor how these segregations, if present, may be arranged. However, we know sufficient to warrant the belief that there will be more or less segregation along the contacts of intrusive, and in particular flows or parts of flows. The actual facts in any particular case cannot be foretold but must be determined in the field. It is quite likely that variations much greater than here indicated could be found in any of the districts named if a magnetic survey were made

Furthermore we know nothing of the magnetite content of the sedimentary rocks present in any of these districts.

Sufficient facts are shown, however, to warrant the inference that a fairly large proportion of igneous contacts, flows, and sedimentary beds will show differences in magnetite content sufficiently large to exert easily detectable effects on the earth's magnetic field, even when obscured by a cover many feet in thickness. The inference is likewise warranted that many igneous contacts and many sedimentary beds do not possess sufficient differences in magnetite content to disclose their position when obscured by even the slightest cover. These facts for any given situation can only be determined by a test on the ground, but the test is so simple and so quickly and cheaply made that the engineer or geologist who neglects to make it when the facts are needed is guilty of as gross neglect as though he refused to look at an outcrop.

MAGNETIC INSTRUMENTS AND EFFECT ON THEM OF CHANGES IN THE EARTH'S FIELD

The two instruments used are the compass and the dip needle. The ordinary compass, or the transit with a compass needle, needs no discussion. In regions of strong local disturbance the dial compass is commonly used. The dial is used in reverse fashion from the ordinary sun dial. Time is taken from a reliable watch set on sun time, then the dial is turned so the shadow cast by the sun agrees with the time of the watch, so that the gnomon is parallel to the axis of the earth and in a true north-south line. The declination of the compass needle from true north can then be observed. In open country a small area can be studied with an ordinary compass by using some distant object as a constant sight. The size of the area and the distance of the object used will determine the accuracy of the observations. The dial compass properly adjusted can be reliod upon to show declinations of half a degree. Declinations to the east of the normal for the region indicate that the attracting object is east of a north-south line through the compass, and west declinations, that it is west.

The dip needle is by far the more valuable instrument; in fact, in regions of mild attractions the chief use of the dial compass is to ascertain the true direction of the traverse on which the dip-needle observations are being made.

The dip needle, as the term is used in this paper, is the ordinary counterbalanced dip needle. If a needle is carefully balanced and then magnetized, it will take a position parallel to the lines of force of the earth's field. The inclination of these lines, measured from the horizontal, varies from 90° at the magnetic poles to 0° at the magnetic equator. In the northen states, it is about 70° to 78° where no local attraction exists.

The south end is upward in the northern hemisphere and downward in the southern hemisphere. In the northern hemisphere we therefore apply a counterweight to the upper, or south-seeking, pole of the magnetic needle so as to bring the needle nearly to a horizontal position.

The action of the dip needle is easily understood if one keeps in mind that there are just two forces acting on it: the force of gravity G on the counterweight, which tends to pull the south end of the needle downward, and the magnetic force M, which tends to pull the north end of the needle downward. Fig. 1 illustrates these forces and the lever arms a, b through which they act.

If we imagine M to decrease gradually in magnitude, its turning moment becomes less and the force of gravity will pull down the south end of the needle. If M decreased to zero, the pull of gravity on the counterweight would turn the needle to a vertical position. On the other hand, if we imagine M to increase gradually in magnitude, its turning moment will become greater and consequently it will pull down the north end of the needle.

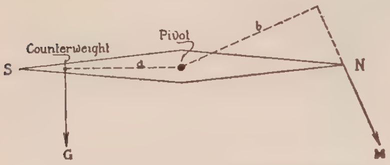


FIG. 1.

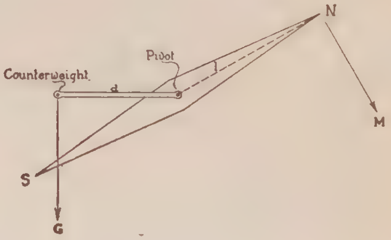


FIG. 2.

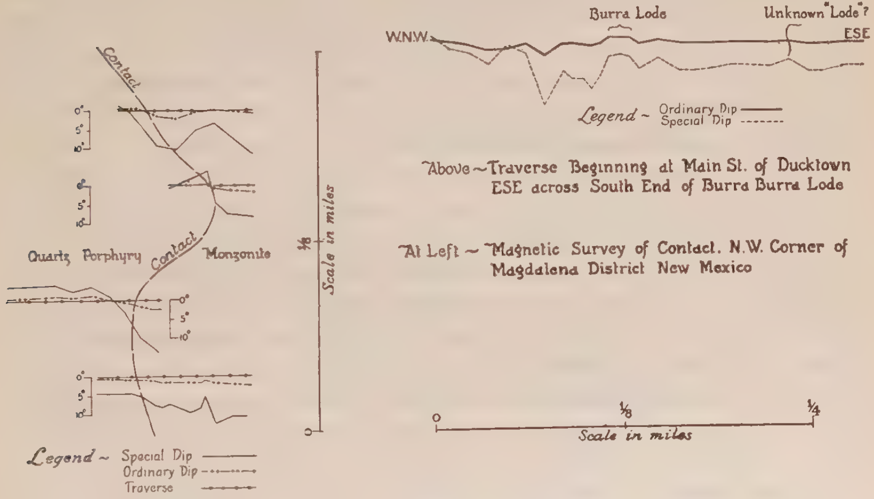
For a given set of conditions, the change in position of the dip needle produced by a given small change in the earth's field can be computed. In general, for the northern United States, the change is from 1° to 2° for a 1 per cent. change in the earth's field; with the dip needle in good condition, readings can be depended on to within ½°. This change in dip-needle reading therefore serves to detect a change in magnitude of the earth's field amounting to only ½ to ¼ per cent.

NOTE.—A change in angle of the earth's magnetic field with no change in magnitude will produce a change in the dip-needle reading, but slight changes in angle are so much less effective than slight changes in magnitude that only the more important change is considered.

The magnetite present in ordinary rocks may be sufficient to increase the strength of the magnetic field at a short distance from the rock surface to many times its normal value. As this magnetic effect decreases in proportion to the first power of the distance, even moderately magnetic rocks would have to be buried many feet before their magnetic effect would decrease from many hundred to the ½ to ¼ per cent. change in the field that can be detected with the ordinary dip needle.

For those cases where it is desired to detect still smaller changes in the magnetic field, the writer devised a special dip needle with which practically any degree of sensitiveness desired can be obtained. If the needle is balanced, as shown in Fig. 2, so that its long axis is perpendicular to the direction of the magnetic force, and its counterweight is on a horizontal line with the pivot, $Ga = Ml$ and the turning moments of gravity and the magnetic field are equal and opposite in their effects on the needle.

The difference in the action of this needle and the ordinary needle lies in the fact that the lever arms of the two forces a and l change in exactly the same proportion as the needle turns so that in any position $Ga' = Ml'$. If the needle is turned an angle b the lever arms are shortened to $a \cos b$ and $l \cos b$. Consequently the turning moments are $Ga \cos b$ and $Ml \cos b$, and both sides of the equation are diminished by the value of a common factor $\cos b$. Hence the needle is in unstable equilibrium and the slightest change in the magnetic force M, either increase or decrease, results in turning the needle 90° from the position shown in the figure. It is obvious that the only bar to infinite sensitiveness in this instrument is the lack of mechanical perfection in its making. Any degree of sensitiveness desired can be attained by varying the angle between the counterweight and the long axis of the needle. Field work has proved that this instrument may easily be adjusted so that a change in the magnetic field that will show 1° change in the ordinary dip needle will register 10° or more on the special dip needle.



Above ~ Traverse Beginning at Main St. of Ducktown ESE across South End of Burra Burra Lode

At Left ~ Magnetic Survey of Contact. N.W. Corner of Magdalena District New Mexico

FIG. 3.

Tests made in coöperation with the U. S. Geological Survey are shown in Fig. 3. The left-hand curves show magnetic variations found in four traverses across a quartz-porphyry monzonite contact. Here the contact is more weakly magnetic than the rocks on either side, except in one traverse, and the monzonite is more weakly magnetic than the porphyry. In the case of the traverse across the Burra Burra lode in the Ducktown district, the lode is more strongly magnetic than the surrounding rocks but just west of the lode there is an unusually weak attraction. Weak attraction is indicated on these curves by negative readings, these do not indicate an actual reversal of the magnetic force, but rather that the north end of the needle was above the zero (horizontal) rather than below it. A somewhat weaker downward tilt of the needle was noticed about 600 ft. east of the Burra Burra lode. To test this attraction, two other traverses were run several hundred feet farther north; it was easily detected and its course was found to be parallel to the Burra Burra lode. Its meaning can be determined only by drilling.

For use in ordinary geologic field work or prospecting, the ordinary dip needle is sufficiently sensitive to changes in the earth's magnetic field. It shows many changes that our limited knowledge of the distribution of magnetite does not permit us to interpret adequately. In beginning its use in any district, an area of abundant outcrops should be chosen and the effect of known conditions of various beds and contacts on the needle carefully determined; then the significant features into adjacent areas where the rocks are covered may be followed. Only where the local changes are usually delicate and important for exploration purposes is it advisable to use the more sensitive needle here described.

INSTANCES OF PRACTICAL USE OF MAGNETIC METHODS

Magnetic methods have been used in the work of the Wisconsin Geological Survey for many years. The work was started with the belief that the only magnetic rocks that could be traced were the iron formations. It was soon found, however, that much more general use could be made of these methods. Igneous flows and intrusives in sediments of various kinds were usually detected quite readily. In some cases the boundary between a quartzite and a slate could be followed with assurance even though covered by many feet of glacial drift. Minor faults offsetting such traceable contacts, in many cases, could be determined even though the rocks were entirely obscured and there was no surface indication of faulting. In the Wausau district, a 30–100-ft. dike of diabase was followed 12 mi. through a region of granite rock, because it possessed a weaker magnetic attraction than the including rocks. In only one place in this distance was an exposure found, which showed definitely the cause of the phenomenon.

The U. S. Geological Survey gave the writer an opportunity to try out these methods, for six weeks, in the Magdalena, Santa Rita, and Hanover districts of New Mexico and for four weeks in the Ducktown district of Tennessee, but since the war, time has not been available to continue the work.

In the Magdalena district the object was to see if the zinc orebodies, which contained some magnetite, would show any effect at the surface. They were found to be too weakly magnetic to give any indication at the surface through the 400 to 500 ft. of cover. In doing this work, however, it was found that the contact between the monzonite intrusive and the rhyolite flows was sufficiently magnetic for it to be followed for 3000 ft. across the wash-covered slope at the foot of the mountains. The accuracy of the work was proved by tracing the contact into a small, and previously unvisited, outcrop in a small gully. Other contacts were found to have a readily traceable magnetic effect. In the northwest corner of the special topographic sheet, the contact of a quartz-porphyry, a tuff, and a monzonite intrusive showed an attraction sufficiently strong to swing the ordinary dip needle several degrees from its normal reading. This contact could be traced readily as far as it was followed, but it was not followed to its end. This is shown in Fig. 3.

In the Chino district, the mineralized portion of the intrusive was observed to be more weakly magnetic than the non-mineralized part. The contact of the intrusive with the limestone at the north was strongly magnetic and could be traced without difficulty. In the Hanover district, the metamorphosed limestone containing the zinc was also strongly magnetic and easily followed by its magnetic attraction. The great limestone mass east of the Hanover valley showed little or no attraction except along the intrusive contact, where there was great variation.

In the Ducktown district, the test was made to see if the pyrrhotite ore was sufficiently magnetic to be traced. The pure pyrrhotite was not attracted even by a strong bar magnet. Only where there was a little magnetite with the ore was there any effect on the dip needle. Here, also, the unexpected was found. Certain connections between the known ore lenses could be traced by a few degrees of deviation of the dip needle; other attractions of the same kind were found in areas where there had been no exploration and no other surface indications of ore were to be found. One of these is indicated east of the Burra Burra lode in Fig. 3. Whether these are simply magnetitic beds or fault zones with no ore, or whether they are ore bands is not known. The Ducktown Co. drilled in one of the belts thus located and found ore but of too low a grade to be mined. A complete magnetic survey of this district, which could be made for the price of a few drill holes, would serve as a valuable guide for future exploration for new orebodies.

Last summer, in Wisconsin, an extensive detailed study of the Keweenawan lava flows and intrusives of the northwestern part of the state was begun. Many individual flows have been traced in detail by magnetic methods, and the solution to many of the problems of the character, structure, and extent of these rocks in a region having a heavy glacial cover is indicated by the characteristic magnetic attractions. We have found, largely by these methods, a great structural discontinuity that probably marks the position of a fault of tremendous magnitude, possibly to be later correlated with the great thrust fault known on Keweenaw Point. If exploration for copper is undertaken in this area (and there is just as good surface indication of its presence as on Keweenaw Point) the major details of the various lodes and their extensions can be readily ascertained from this work.

Another interesting piece of magnetic work is now being carried on by W. R. Crane, of the Bureau of Mines. In certain brown-ore fields that are known from drilling, Mr. Crane has strong proof that there are mild magnetic attractions that are definitely related to the orebeds and their depth. This work has not been completed but the writer is expecting from this work some interesting and important results that will be of great value in guiding the drilling of brown-ore deposits.

Sufficient has been stated to indicate that careful observation of the local magnetic attractions will tell many things about buried rocks that otherwise can be obtained only by expensive drilling or underground work. How much these observations will tell we cannot say. There have been too many new facts shown in the last few years to permit a prediction of the limits of usefulness of such methods. Their value is so great and their cost is so slight, when compared to the cost of any extensive drilling compaign, that much more extensive use of them should be made.

As an illustration of possible use the writer would suggest, what at first glance seems an absurdity, the use of magnetic methods in oil wildcatting. Although an oil sand, as such, will not give any magnetic effect that we can detect, particularly when buried a few hundred feet or more, coastal oil domes show a core of material quite different in character from the surrounding rocks. Either this core or the surrounding rocks may contain sufficient magnetite to be detected by the special form of dip needle. Not having tried this the writer does not know if this is true but we do know, in general, that a difference in material is almost a sure indication of a definite difference in magnetic permeability. If this difference is sufficient for its effect on the local magnetic field to be detected with the instruments at our command, we have a means of locating any domes that hitherto have not been observed because of lack of topographic expression. If such domes could be located by a cheap surface method, such as a magnetic survey, that method would be a valuable guide to the man exploring for oil.

Conclusion

The purpose of this paper will have been attained if it succeeds in arousing the interest and curiosity of mining engineers, explorers, and geologists in these methods of following geological features of interest and of economic value under a cover of obscuring material. There is little question but that valuable mineralized contacts and beds or lava flows can and will be followed, and will help lead to the discovery of new and valuable deposits of ore. The presence of only a small percentage of magnetite is needed to make this feasible, and ores of nearly all minerals or the associated rocks may, and in many cases do, contain this amount. If the desired interest can be stimulated we will learn many facts about the distribution of magnetite in rocks, which in turn will make possible better and more accurate interpretation of the significance of local magnetic attractions.

Porcupine Ore Deposits

By Louis D. Huntoon, New York, N. Y.

(Canadian Meeting, August, 1923)

The Porcupine gold area, located on the Hudson Bay slope of northern Ontario, has produced over $100,000,000 in gold and has paid more than $28,000,000 in dividends, since the first real production in 1912. The character of the ore has been studied by the geologists of the Bureau of Mines of Ontario, the geological staffs of the operating mines, and engineers commissioned to examine properties there. What information the Government has is free to everybody; the geological staffs of the several properties also give to the visiting engineer what information regarding ore deposition they possess, and the author wishes to express his indebtedness to the geologists of the Ontario Bureau of Mines and the staffs of the operating mines for their free discussion of geological questions. This paper is based on the examination of several properties, various petrographic reports, not only on specimens from Porcupine but from elsewhere in northern Ontario, and interviews with Government and mine staff geologists. The petrographic work has been done by Prof. R. J. Colony, of Columbia University.

GEOLOGICAL HISTORY

The geological history of the district may be summarized as follows: The outpouring of great floods of lava accompanied by beds of tuff, flow following flow; the contacts of different flows can be seen in various parts of the camp. This period was followed by an uplift and the production of the Temiskaming sediments by erosion. Later, dynamic action turned the sediments and underlying lavas on edge; this was followed by the intrusion of igneous rocks—in the Porcupine area the quartz porphyry, which is of such great importance in connection with the gold deposits. Doctor Burrows says: "The intrusions of quartz porphyry have in some way influenced the deposition or location of the gold but it is not likely the porphyry has been the source of the gold-bearing solutions." This statement of Doctor Burrows I indorse.

The formation of non-gold bearing quartz veins followed the intrusion of the porphyry. These veins occur along or near the contact of porphyry-greenstone, in the greenstone, and in the porphyry mass itself; this quartz is accompanied by chlorite, sericite, and carbonate. Prior to gold deposition, the general composition of the greenstones was greatly altered so that they now are quite similar in composition. Then came an age of geologic disturbances, which brecciated the quartz veins and opened the enclosing greenstone schists, producing thereby favorable channels for the circulation of solutions. Gold was then deposited in the quartz breccia and schists as auriferous pyrite and visible and spectacular gold. The petrographic slides showed no further geological disturbances. There have been movements subsequent to the gold deposition, as shown by the carbonaceous or graphitic fault in No. 5 vein of the McIntyre, the displacement of the No. 5 vein at depths of 300 to 500 ft., and by a fault associated with the orebodies of the Hollinger. There appears to have been no circulation of gold-bearing solutions following those later disturbances. In the examination of properties in the district this later disturbance—this later faulting—must be taken into consideration as, so far as the author's study goes, it is non-gold bearing.

TYPES OF OREBODIES

Various types of orebodies are outlined by Percy Hopkins, "Ontario Gold Deposits" 1921, two of which are of interest in the Porcupine area and are too closely related in this area to be segregated into two distinct types: (1) The auriferous pyrite quartz type where the values occur along crushed and schisted zones, exemplified by the Hollinger, McIntyre, and Dome mines; and (2) the sheet-like deposit, "simple or true fissure veins," exemplified at the Rea (now Newray) mine. Here the values occur along minute parallel planes or cracks in the quartz with smooth walls; where the quartz has been brecciated slightly spectacular gold occurs.

I am of the opinion that these two types are simply modifications of one orebody; the origin of the gold is the same. The deposition of the free gold probably took place at a different period from the auriferous pyrite and with depth and favorable geological conditions the sheet-like veins have promises of developing auriferous pyrite orebodies. My petrographic analysis, to date, has given no clue as to the age of the visible and spectacular gold; it was deposited under conditions similar to the auriferous pyrite. It was the visible and spectacular gold that led to the discovery of the camp, and it is partly because of this fact that I make the statement that, with depth, croppings of visible gold under favorable geological conditions have promise of developing valuable sulfide orebodies. Percy Robbins, in the first Hollinger annual report, 1912, says "our sampling has shown that the occurrence of visible gold does not mean payable values." The auriferous pyrite has produced the orebodies and the visible gold developed in these sulfide deposits serves as a "sweetener" only.

ORIGIN OF GOLD

Most of the geologists and engineers who have studied the Porcupine area do not think that the quartz porphyry has been the source of the gold. The quartz porphyry is closely related to the deposition of the gold by producing favorable structural conditions. The mineralization of the orebodies points to their formation at great depth and at high and intermediate temperatures. The source probably was from a deep-seated magma. There has been no secondary enrichment, which might account for the spectacular gold. The report of a petrographic analysis of a spectacular gold specimen says: "This sample supports opinions expressed in previous reports as to the origin of the gold, the distribution of the gold, and its relation to vein quartz. The gold is the latest thing to be precipitated, and it is of primary origin from probable aqueo-igneous sources."

GOLD DEPOSITION

As has been stated, the gold was the last mineral to be deposited in brecciated quartz, and schistose and brecciated greenstone. The gold occurs as auriferous pyrite, visible gold along slip planes in the quartz, and spectacular free gold in brecciated quartz. Petrographic slides show no free gold in the pyrite but etched polished specimens of auriferous pyrite show free gold under the microscope. All petrographic analyses of gold specimens showed schisting, brecciation, intense alteration of the greenstone, formation of quartz veins carrying no mineral, further crushing and brecciation, followed by deposition of auriferous pyrite and visible gold.

In places immediately adjacent to the porphyry, especially in front of the nose of the porphyry, the greenstone enclosing good quartz veins may be so highly carbonated and silicified that it did not crush just prior to the advent of the gold solutions, with the result that values are usually very low grade and most erratic; nevertheless visible gold has been seen under such conditions. This is an example of a most favorable structure containing quartz veins of little or no value. Drifting away from the porphyry influence should develop gold values.

The gold, as a rule, does not extend for any great distance into the porphyry, although the quartz veins do. The Dome report, for 1922, says: "Another orebody has been found at the eighth level in the porphyry (known as 824) . . . drift has disclosed a zone of well-mineralized porphyry and quartz veins and veinlets. The average assay value disclosed in this drift is $12.08 for a length of 160 ft."

The author visited this orebody in 1922 and was most favorably impressed; petrographic analysis of specimen from here was most favorable. The annual report of 1923, one year later, says: "Stope 824 in the porphyry . . . has proved disappointing . . . 1249 tons averaging $6.11 were drawn from this working."

Mr. Brigham advised the author a year ago that he had summarized the value of ores mined in the porphyry on the Hollinger and found them to be low grade on the average and disappointing.

In the last report of the Dome mine, mention is made of drilling through the contact porphyry (500 ft.) underlying the ore deposits in the sedimentaries and that no ore was developed in the porphyry; ore was later developed in the underlying greenstones.

The foregoing tends to confirm the opinion that workable orebodies do not extend into the porphyry; at least not to date. The coarse-grained pyrite deposited in the greenstones rarely if ever carries gold. The pyrite, although not crushed, appears to be of a different geological age.

Deposition Control

When submitting specimens to Prof. R. J. Colony for petrographic analyses, the author has invariably requested information regarding the controlling factors of deposition—was the deposition controlled by structure or by composition of the enclosing rocks, or by both?

All of the petrographic reports show the ore to be a mineralized breccia; deformation and schisting previous to brecciation; a variable original rock character such as felsitic, dacitic, andesitic, and tuffaceous schists. The petrographic reports further show that before gold deposition the formations were so highly altered that they are not very different in general composition. These reports state: "Whatever their original character, they all have been so profoundly altered by silicification, chloritization, carbonatization, and sericitization, that they now are not very different from one another in composition, nor are they greatly different from the unmineralized portions of the rocks."

The unmineralized rock specimens were from formations away from orebodies. As the general composition of the ore, enclosing walls, and formations some distance from the ore are not very different in composition, the deposition of gold values does not appear to be controlled by rock composition; that is, it is not the prime factor of control. The petrographic reports also say: "If any compositional control at all is exerted, chlorite and sericite seem to exercise it for the pyrite is usually found in strongly sheared places which are sericitized and chloritized . . . the pyrite is also associated with quartz, carbonate, and chlorite of vein origin (hydrothermal) not related to the rock at all . . . so far as any petrographic evidence indicates, the gold (*i.e.*, pyrite) was controlled in its depositions by favorable structural conditions, in connection with, very likely, places containing more chlorite than others."

U. OF I.

To summarize the foregoing, the controlling factor of ore deposition appears to be one of structure and not composition, unless the early deposition of chlorite had some effect which does not seem likely.

Favorable Geological Conditions

Present developments show that the favorable conditions for orebodies comprise a schisted, brecciated, greenstone associated with a porphyry intrusion. So far in the Porcupine area no mines have been developed in the porphyry mass itself. The porphyry appears to have resisted the schisting and pronounced brecciation just prior to gold deposition. There is every reason to believe that if the porphyry had responded to the geological disturbances and produced favorable structure just prior to the gold deposition epoch that large bodies of ore would have been deposited in such porphyry as has been deposited in the greenstone.

Depth of Ore

What controls the depth to which the orebodies will go? Will underlying porphyries cut off the values? There is no direct answer to these questions. The last annual report of the Dome mine indicates that the gold in the Temiskaming sediments is bottomed by the porphyry; a later report indicates valuable orebodies in the greenstones underlying the porphyry mass (500 ft. thick) but no ore in the porphyry. Further developments at the Dome should throw some light on these questions.

There appears to be a decided rake of the orebodies to the east, as shown by the No. 5 vein on the McIntyre. The 84-56 veins of the Hollinger, which occur on or near the surface on the Acme (Hollinger Consolidated), is presumably the No. 7 vein on the McIntyre, which apexes at a depth of 1375 ft., showing a decided rake to the east. From this it would appear that the origin of the gold solutions was in the east and that they flowed upward westerly, possibly following a deep trough in the porphyry as well as fracture zones below certain overlying tongues of porphyry.

The mineralization indicates that the orebodies were formed at depth. Various hypotheses can be deduced regarding the depth to which the orebodies will extend, but the author's opinion, based on present development work, is that the orebodies will extend to great depth and that such depth on individual zones will increase as advance is made easterly.

Grade of Ore with Depth

My data is not sufficient to draw definite conclusions regarding the grade of the ore with depth. Available data, however, leads to the belief that starting with the apex of a vein, whether on the surface or below ground, the grade gradually increases for a considerable distance. This condition does not show in the mill heads, as annually reported by the operating mines, as the development of higher grade ore permits the economic mining of lower grade ore for an average mill head. On the No. 5 vein of the McIntyre, the upper levels were low grade; with depth, the grade increased, as shown by the annual report for 1916, which says: "The grade of the ore has increased from $9.20 on the 400-ft. level to $12.62 on the 600-ft. level." A similar condition was observable on McIntyre No. 7 vein early this year; the ore lenses developed on the 1875-ft. level were higher grade than those on the 1750-ft., and they, in turn, were higher grade than those on the 1650-ft. levels.

Reports of the Dome mine, dated March 31, covering a period after the caving system of mining had been abandoned and the orebodies of the Temiskaming sediments in the vicinity of the intrusive porphyry are being mined, give the following annual mill heads; 1920, $6.96; 1921, $7.51; 1922, $8.20; and Mar. 31, 1923, $12.12.

The examination of an assay map at another mine indicated that the value of the ore at that property was also increasing with depth. Without mine-assay sheets, from which a careful study can be made, definite conclusions cannot be drawn. The indications are that the value of any individual orebody increases perceptibly with depth and beyond this enriched zone the values will probably decrease.

DISCUSSION

Frank L. Hess, Washington, D. C.—Did you find any porphyries in the Porcupine region that were as little changed as the Kirkland Lake?

Louis D. Huntoon.—The prophyry in the Porcupine area is quite different from that in Kirkland Lake. In both districts quartz veins appear to have been formed prior to the deposition of gold. In the Kirkland Lake area, the quartz veins in the porphyry responded to later dynamic action, forming favorable channels for the deposition of gold; in the Porcupine area, the porphyry containing primary quartz veins appears to have acted more as a rubber buffer and did not open to any extent just prior to gold deposition. The results are: Orebodies in the quartz veins of the Kirkland Lake porphyry and little or no ore in the quartz veins of the porphyry in the Porcupine area.

C. A. Filteau, Cobalt, Ont.—You do not mention silicification in regard to the porphyry.

Louis D. Huntoon.—I presume you refer to the alterations of the greenstones. My petrographic reports show that before gold deposition the greenstones were so highly altered by silicification, carbonization, chloritization, and sericitization that they were not very different in general composition at the time of gold deposition. In contact with the porphyry, in many places, the alteration of the brecciated, schisted greenstones has been so pronounced as to form an envelope of a compact, blocky structure which did not respond to the dynamic action just prior to gold deposition. In drifting away from the porphyry at such places, good values have been developed where the structure was favorable

C. A. Filteau.—It used to be stated that the three elements favorable to ore occurrence were silicification, schisting, and mineralization. Would you confirm that?

Louis D. Huntoon.—Not exactly. My opinion is that the favorable conditions for ore deposition comprise a schisted, brecciated formation, mineralized, and associated with a porphyry; I am also of the opinion that the primary quartz veins in the above structure may be essential; there are such indications. The porphyry appears to have assisted in producing the favorable structure just prior to the gold deposition, but is not related to gold deposition otherwise. It is not the source of the gold.

C. A. Filateau.—Does not practically all the commercial ore show mineralization?

Louis D. Huntoon.—All of the commercial ore so far developed in the Porcupine area shows pyritic mineralization—auriferous pyrite. The visible and spectacular gold acts as a sweetener to the grade of the ore. Sulfides are essential to make an orebody.

C. A. Filteau.—Do you know any theory on the origin of the graphitic schist? Did you hear it was of sedimentary origin?

Louis D. Huntoon.—It has been suggested to me that the graphitic schists were of sedimentary origin but I have seen no foundation on which to base such argument. My opinion is that the graphitic schists are the result of faulting subsequent to gold deposition. Several such faults are exposed in the McIntyre. Under petrographic analysis, the origin is obscure; it appears to be related to a later movement and my petrographer states: "can see no other source for the graphite than an igneous one."

Norman C. Stines, New York, N. Y.—Is it not true that the McIntyre is now developing ore in depth not because it has reached depth but because it has reached the same ore-bearing zone as is on the Hollinger ground, due to the easterly rake of that zone and they are developing the easterly rake of it?

Louis D. Huntoon.—That brings out a most interesting criticism of my paper where I refer to the No. 7 vein of the McIntyre as apexing at 1375 ft. The top, or apex, of the No. 7 vein was discovered about 1000 ft. easterly of the Hollinger workings. It is on the same strike as the orebearing zone on which veins 84-56 of the Hollinger are located. It may be the easterly rake of the same ore-bearing zone

Norman C. Stines.—It is only because that zone at that place comes into the McIntyre ground. The surface of the McIntyre is made up of probably 75 per cent. of porphyry, at 1500 ft. the same area is represented by less than 50 per cent. of porphyry; so that instead of the orebodies increasing in depth or instead of the orebodies coming in in depth, this mine has come into the ore-bearing zone in depth.

Louis D. Huntoon.—When we consider the No. 7 vein and those south, it is quite true that the McIntyre mine has developed the ore-bearing zone of the Hollinger in depth. This is not true though when we consider the No. 5 vein of the McIntyre. This vein is quite independent of the Hollinger ore-bearing zone and is some distance to the north.

Norman C. Stines.—Is it not also true that the original fissure along which these orebodies have formed went through not only the volcanic complex but through the porphyry and right on through the basalts; that the veins of the bodies in some cases have even gone into the porphyry; that where the vein in the Keewatin was extremely rich when it passed out of the Keewatin into the porphyry it carried enough Keewatin to carry gold with it?

Louis D. Huntoon.—The primary, or original, fissures went through the volcanic complex and the overlying porphyry; primary quartz veins were deposited in these fissures but contained no gold at time of deposition. Where the rich veins of the Keewatin pass into the porphyry they may or may not carry values in the porphyry. The ore mined from such veins in the porphyry have been most disappointing in average values. The porphyritic gold ores that I have had examined showed no Keewatin formation in the porphyry; they did show favorable structure for ore deposition. I have seen but one specimen of porphyry carrying any Keewatin formation.

Norman C. Stines.—You mentioned another example in the district where the values had increased many times upon reaching a certain depth; is it true there, as in the McIntyre, that the increase in values in depth has been due not so much to the original condition as to the mine having passed into good conditions that are dipping into them from the east?

Louis D. Huntoon.—My petrographic work so far has given me no clue as to the reason for increased values with depth on any individual orebody. The structure and mineralization appear to be identical. I have given this question considerable study and have considered the possibility of a highly pregnant solution passing upwards and depositing the values rapidly where conditions were favorable and the less pregnant solutions passing on and depositing lower values toward the top of the same orebody. This is the only explanation I have to account for increased values with depth on an individual orebody.

Norman C. Stines.—Is not that mainly because the mine is getting away from the influence of the porphyry dam itself? Is it not true that if a horizontal section is taken that as the porphyry is approached in any one fracture the values decrease?

Louis D. Huntoon.—The values are liable to be low if the orebody is close to the porphyry but so far as I can judge this is not necessarily the controlling factor. The porphyry appears to have had no influence on the No. 5 vein as a whole. Recent developments of high-grade ore at the west end of No. 5 vein expose porphyry tongues extending into the orebody from the west. This particulr orebody appears to be on the upper side of the porphyry and connected with the solution channels responsible for the No. 5 vein deposition.

Scott Turner, Toronto, Ont.—The point has just been raised as to whether or not the McIntyre found new ore in this No. 7 vein or whether it found the continuation of the Hollinger ore. The McIntyre should get full credit for that ore because the Hollinger ore had not picked up No. 7 and No. 9 veins; so as far as those ores are concerned they were new discoveries.

Another interesting thing there is the sequence of the fractures. Of course, I presume this is the author's theory too. The old Keewatin series were heavily fractured and sheared before the intrusion of porphyry. Whether or not that porphyry followed the lines of previous shearing is an interesting point. The maps show that the older shearing did not follow the intrusion, but was influenced by it; there was an angle of about 15° between the two directions. After the porphyry was in there, other shearing occurred, which extended, as you say, through the Keewatin and through the porphyry. At the same time the intrusion of the porphyry itself may have produced an independent shearing parallel to its major axis and extending from the prow toward the west.

Frank L. Hess.—By shearing do you mean something that made the rocks schistose?

Scott Turner.—Fracturing, is perhaps, a better word than shearing.

Louis D. Huntoon.—I think it is shearing followed by brecciation.

Scott Turner.—The wedging in of that porphyry mass can be explained by the fact that the porphyry must have come in at an angle and tried to push through, and was continuously turned off until it was almost parallel to the direction of the older shearing.

Another interesting thing is that the Hollinger orebodies were all overlain at one time by that porphyry; that is, it is the underside of that porphyry intrusion that is productive in the Porcupine camp, so far as these mines that we are discussing are concerned, and the No. 7 and No. 9 veins of the McIntyre, therefore, correspond with all the known Hollinger veins, and they are mineralized almost up to the porphyry.

McIntyre No. 5 vein is different from the Hollinger series. There, instead of a steep-dipping or almost vertical fracture, you have a 55° dip to the northwest. Is there anything in the theory that the No. 5 vein may be simply a fracture zone joining two independently, separate, porphyry intrusions? It may be a different type of vein.

Louis D. Huntoon.—There may be something in the suggestion that No. 5 vein is a fracture zone joining two separate porphyry intrusions. To the east, the No. 5 and No. 7 veins appear to join or at least to be very close to one another. To the west, they separate forming a wedge-shaped area that has been practically non-productive to a depth of 1500 ft.; this area may become productive with depth. The No. 5 vein has been considered a fissure vein quite independent of porphyry intrusions. Recent developments on the lower levels have exposed a porphyry at the west end raking easterly. Immediately adjacent and in front of this porphyry, on the 1375-ft. and 1500-ft. levels, a large valuable orebody has been developed. At the time of my last visit, sufficient work had not been done in this area to draw definite conclusions. My opinion is that the No. 5 vein or fracture zone may be related to this western porphyry, which, so far, is quite distinct from the main mass of porphyry associated with the No. 7 vein.

Scott Turner.—Another interesting thing is that this ore is only on the lower side of the porphyry; there is none on the upper side, although that porphyry core is supposed to have been the source of the gold. Another interesting formation is in the Schumacher, where one must go about 3000 ft. (as shown by a longitudinal section through the core of that porphyry) before one will find the place under the bottom of that porphyry where conditions will be similar to the shallower mines farther west. The pitch of the porphyry is so steep to the east that you get a tremendous problem of exploration with depth in that direction. You are faced with this problem without going far east of the Hollinger.

Louis D. Huntoon.—I am of the opinion that it is possible to have ore on or near the porphyry contact, as well as below the porphyry. If we take the pitch of the porphyry it will be necessary to go to a depth of approximately 3000 ft. on the Schumacher to get ore. The development of this property will be most interesting. In regard to there being ore on the upper side of the porphyry; I consider that the recent developments of the McIntyre No. 5 vein, west end, indicate that it is possible for orebodies to occur above the porphyry, provided such favorable structure is connected with an ore-solution channel.

SCOTT TURNER.—Porcupine contains different types of orebodies; In one case the sheared basics have all the ores and the porphyry contains none. In other places, the sheared Keewatin complex carries no ore, but the sediments carry ore. In the Paymaster, the Keewatin does not carry ore but the porphyry does.

B. L. EASTMAN.—At the Paymaster Mine, microscopic work has shown that the ore in the quartz porphyry is related to shear zones. These shear zones in the porphyry have been cemented by siliceous solutions. The strike of the shear zones is northwest–southeast 35° and practically vertical. Three of these zones have been outlined and prospected by drill holes; the ore so far as known, except for some high-grade ore on the contract, is entirely within the quartz porphyry.

LOUIS D. HUNTOON.—My opinion is that the controlling factor of ore deposition appears to be one of structure and not composition of the enclosing wall rocks. Orebodies may occur in the porphyry sedimentaries or Keewatin provided structural conditions are favorable.

CHARLES P. BERKEY, New York, N. Y.—It is clear that there has been a lot of intimate shearing of all of these rocks. Whether it belongs to two periods or to one I do not know.

FRANK L. HESS.—What do you mean by shearing?

CHARLES P. BERKEY.—The intimate mashing of rocks developing schists. I am not talking about a crush zone, which may leave adjacent country unaffected, but shearing throughout the rock. The porphyry is so badly sheared that it is now a perfectly good porphyry schist.

All, including the ancient basic rocks and also the porphyries, are immensely affected by alteration, and particularly by the transforming of the original minerals into carbonates and hydrated products, and then by the introduction of sulfides. Accompanying that, at some stage, there was a fracturing of the rocks, both the porphyry and the older rock, and in these fractures quartz veins have formed. In every case that I have examined, the first or earlier vein quartz is barren of gold. But toward the end of these mineralizing processes, the solutions carried gold; and at that stage, wherever there were weaknesses that the solutions could enter, gold came in. In those portions of the rock where quartz veins were already developed they were more or less shattered, and in the cracks healed up at that stage there is gold. In those portions, however, which were not shattered in that way I have never seen any gold. This history cannot be unraveled so readily in those places where vein quartz does not occur.

It may be that in the porphyry of a certain locality there is not any gold. For some reason or other that block may not have developed the necessary weaknesses to encourage circulation through it of the gold-bearing solutions of the period when gold was being carried in, whereas at some other point the behavior in this respect was better. Thus a quartz vein in the porphyry might be crushed and become an open channel for circulation at that particular stage, while adjacent less brittle country rock might be little affected.

It is a very interesting speculation where these ore minerals all came from. I do not believe for a minute that they were gathered from the country rock or that they came from porphyry masses that we see. The evidence seems conclusive that the solutions came from magmatic sources below, charged excessively with gases in which CO_2 abounded. That is what changed the rock, causing enormous carbonation and chloritization. Escape must have been easier at some places than others and accompanying periodic deformation must have changed these escape channels materially from time to time. The ores have all come from the surplus mineralizer content and emanations that escaped or perhaps were forced out of an underlying magmatic mass as it cooled and finally crystallized. They are a part of the end-stage products and effects as rejected solutions passed into or through these overlying rock formations.

C. M. EYE, San Francisco, Calif.—To what extent has secondary enrichment played a part?

LOUIS D. HUNTOON.—There has been no secondary enrichment at all.

CHARLES P. BERKEY.—Perhaps there has been about 2 in. of alteration on the average—it does not count at all in the ores.

Magnesite Deposits of Grenville, Quebec

By G. W. Bain, Lachute, Quebec

(Canadian Meeting, August, 1923)

The Canadian magnesite deposits are situated in Grenville township, about 8 miles from Calumet station, on the Canadian Pacific Railway, but the poor transportation facilities have hindered their development. Magnesite was discovered in the boulders near White Rock Lake by Rev. W. P. Boshart in 1900. Observing that the material of these boulders was more glistening and whiter in color than Grenville limestone, he sent a specimen to the Department of Mines, Ottawa. An analysis showed this to be almost pure carbonate of magnesia and of commercial value if present in sufficient quantities.

That summer, Boshart and McAllister found numerous outcrops, both of boulders and of bed rock, that gave promise of being magnesite of a commercial grade, and decided that this mineral deposit, if developed, would rival that at Veitsch, Austria. So they procured options on most of the land on which they knew the magnesite occurred. But the transportation difficulties were too great for the deposits to be mined at the prices offered for the product.

In 1907, T. J. Waters organized the Canadian Magnesite Co. and opened a quarry on the north half of lot 18, range IX, and later on lot 15, range IX, of the same township. A limestone kiln was erected and the calcined rock was hauled to Calumet. In 1914, the North American Magensite Co, took over the quarries and shipped both crude and calcined magnesite until January, 1923.

During 1915, S. Melkman organized the Scottish Canadian Magnesite Co. and opened a quarry on lot 15, ranges X and XI, Grenville township. A light, narrow-gage railway connects this quarry with the C. P. R. at Magnesite station. but shipments are made only during the summer months.

Numerous other quarries have contributed to the output of the district but the greater part has come from the open pits of the

North American Magnesite and the Scottish Canadian Magnesite companies.

In 1917, Wilson[1] estimated that the deposits had 686,900 tons of crude magnesite containing less than 12 per cent. of lime; this was suitable for the manufacture of firebrick, refractory materials, etc. In addition, there are 483,700 tons of magnesite-dolomite containing more than 12 per cent. of lime which was suitable for the manufacture of artificial stone for building purposes. Since this estimate was made, more magnesite has been discovered; on the other hand, much of the lower grade material is no longer merchantable as the purer material from Austria is again available.

TOPOGRAPHY

The district in which magnesite occurs is situated in the county of Argenteuil, on the border of the Laurentian highlands and at an elevation of 400 to 500 ft. above the valley of the Ottawa. Lying on the edge of the uplands, Argenteuil County is intersected by many deep valleys cut by streams flowing southward. Only one hill rises above the even skyline; this extends from Brownsburg to Magnesite station and is a monadnock of Algoman granite and syenite surmounting the pre-Cambrian peneplain. From Lachute westward, the Canadian Pacific Railway follows the margin of the highlands, which end abruptly in an escarpment, 200 ft. or more in height, and broken by the valleys of the Rouge and West rivers, and many smaller streams. Where the streams leave the highlands, there are rapids and waterfalls, which in many places are used for the development of power.

GENERAL GEOLOGY

The rocks of Argenteuil County are divided into five groups according to their age.

TABLE OF FORMATIONS

Quaternary	Champlain	Marine clays, sand
	Glacial	Boulder clay, gravel, sand
	—Unconformity—	
Paleozoic..................	Chazy	Limestone
	Beekmantown	Calciferous sandy dolomite
	Potsdam	Sandstone
	—Unconformity—	
Late pre-Cambrian Intrusives	Stocks	Trap
	Dikes	Granite, syenite diabase
	Batholiths	Granite, syenite
Early pre-Cambrian Intrusives	Dikes	Agirite Bostonite
		Aegirite pegmatite
Grenville series	Grenville	Crystalline limestone
		Skarn rocks
		Biotite gneiss

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[1] Memoir 98, Canadian Geological Survey.

A straight line joining Lachute and Calumet separates the Paleozoic sediments from the pre-Cambrian rocks. The Paleozoic escarpment lies along this line, so that both topographically and geologically the pre-Cambrian and Paleozoic rocks have a marked contrast.

The Grenville series consists of coarsely crystalline white or blue limestone, skarn[2] rock, and biotite gneiss in long parallel bands, which are followed by the river valleys. The crystalline limestones are a highly folded sedimentary series with few shale or sandstone partings. They consist largely of interlocking crystals of calcium carbonate with negligible percentages of magnesium carbonate, silicates such as wollastonite and more rarely diopside, phlogopite, scapolite and chondrodite, and small hexagonal flakes of graphite. The bedding is shown by a local abundance of graphite and silicate minerals; although in some localities the limestones are so massive and uniform that all traces of stratification are lost.

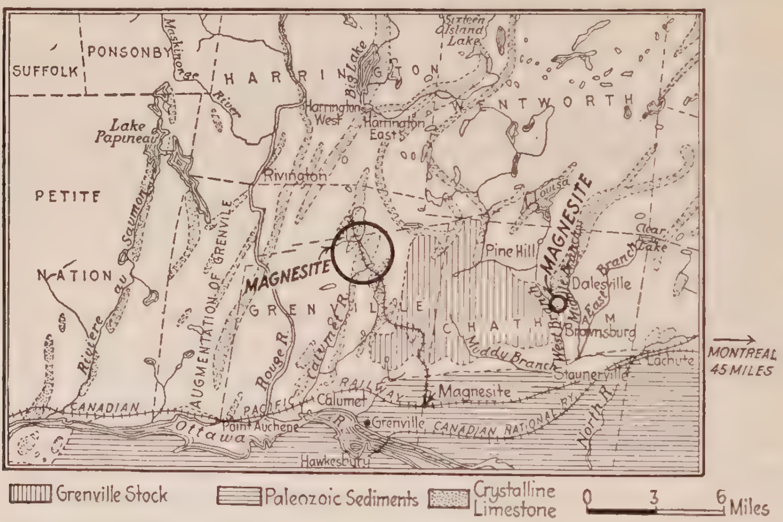

FIG. 1.—THE MAGNESITE DISTRICT.

The skarn rocks are of three more or less distinct types, as some minerals constituting one type seem to be unwilling to form in another. The types are: (1) Diopside, quartz, albite, and sillimanite, the latter in small quantities; (2) diopside, talc, albite, carbonates, and a few grains of almandite and grossularite; (3) almandite, quartz, albite, and orthoclase.

Skarn rocks of the first type are massive, white or mottled by the green pyroxene, or have a foliated or laminated appearance; they stand out as ridges or mounds along the margins of the crystalline limestones.

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[2] A term current among Swedish iron miners for the aggregates of basic silicates, especially hornblende, biotite, and pyroxene, which are associated with the magnetic ores. James F. Kemp: "A Handbook of Rocks." D. Van Nostrand Co., 1918.

Rocks of the second type, likewise, are massive but are green and have a glittering appearance, because of the small flakes of talc and the cleavage planes of the carbonate. These rocks weather readily and do not form prominent ridges. Rocks of the third type have a mottled appearance, caused by the dark red almandite garnets, ¼ to ⅓ in. in diameter, standing in relief against the white or colorless feldspar and quartz. The garnets are most abundant in parallel bands which may represent bedding planes. The rock of this type forms prominent knobs and ridges toward the edges of the bands of the Grenville series.

The skarn rocks lie between the Grenville limestone and the biotite gneisses, which have a foliated appearance as a result of the parallel arrangement of the mica; in other cases the gneiss is mottled because of the development of large crystals of colorless feldspar or almandite garnet.

Pegmatite dikes break across the gneisses and also intrude them in *lit-par-lit* fashion. Along the valleys, these rocks exfoliate under the action of frost and give steep cliffs.

The biotite gneisses occur next to and are intruded by Laurentian granite batholiths and associated dikes. These intrusive masses form ridges or divides between the limestone areas; near the contact, they are highly foliated and contain numerous xenoliths of biotite gneiss. When free from inclusions and still not far from the intruded rocks, the intrusives are pink and commonly show perthitic intergrowths of albite and orthoclase. In localities farther from the contact and nearer the center of the batholith, ferromagnesian minerals locally become more prominent and the rock becomes dark gray to almost black.

Granite and syenite of Algoman age break through the Laurentian batholith and Grenville series. This intrusive shows no foliation and has the appearance of a stock about 20 miles in diameter. It is composed of crystals of orthoclase up to ½ in. in length, a smaller quantity of colorless quartz, and a dark green ferrous hornblende. Syenite porphyries occur as small outliers, such as dikes and plugs; they are characterized by phenocrysts of orthoclase in a felsitic ground mass.

Trap dikes intrude all the rocks that have been mentioned.

South of the pre-Cambrian escarpment, Paleozoic sediments lie unconformably above the older crystallines, Quaternary boulder clay, loess and lake deposits cover the bed rocks in many parts of the area.

Crystalline Limestone and Dolomites

The deposits of magnesite and dolomite occur in the Grenville limestone and are intimately associated with the skarn rocks, biotite gneiss, and dikes of Laurentian or earlier age. The crystalline limestones, when free from magnesium carbonate, are blue and coarse grained. The crystals of calcite range in size from ⅛ to 2 or 3 in. in diameter. The grains have rounded corners and are closely interlocking. Usually they are twinned and have a hydrous magnesian silicate analogous in composition to spadaite developed along the intersection of the twinning planes. Graphite flakes with hexagonal outlines occur in parallel arrangements throughout the mass of the limestones; these flakes not only do not occur in the silicate minerals but appear to be pierced by them. Wollastonite

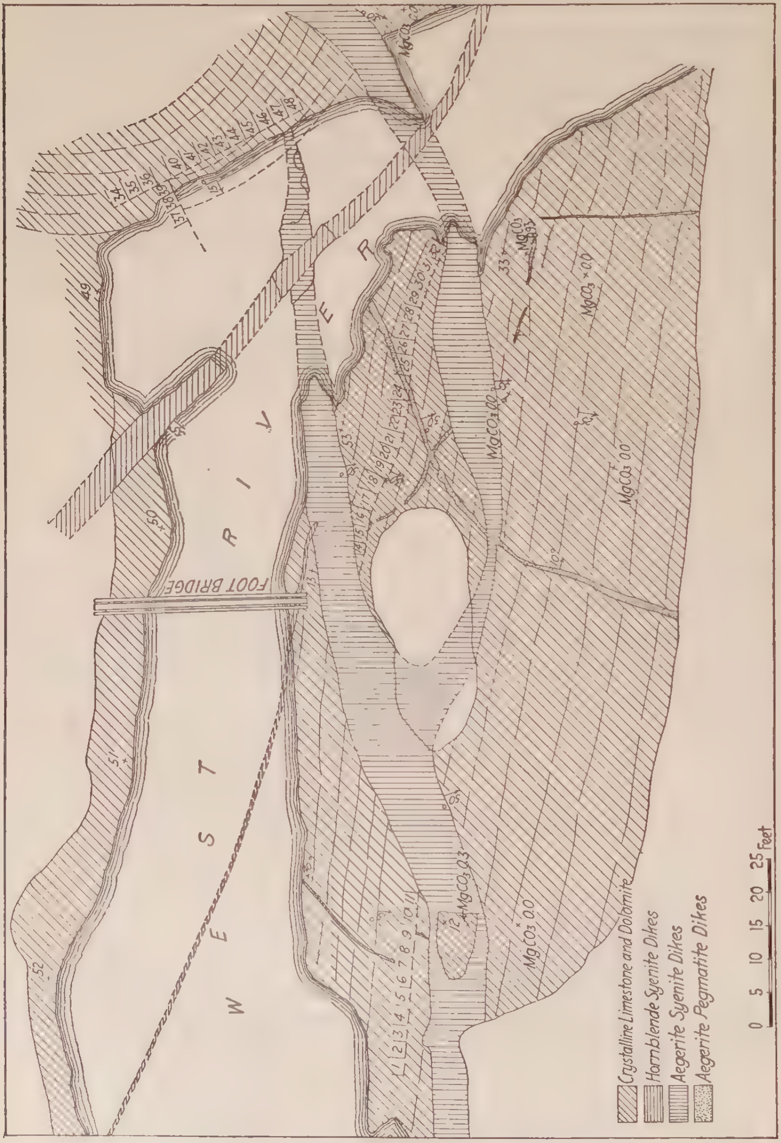

FIG. 2.—LIMESTONE CHUTE, ARGENTEUIL COUNTY.

occurs in large nodules, which weather out and form rough knobs on the calcite mass.

The limestones are intruded by dikes from the Laurentian batholith, by Algoman granite and by the younger traps. Near the Laurentian and pre-Laurentian dikes, the carbonates have a finer grain and silicates—such as phlogopite, chondrodite, scapolite, diopside, pyrope, forsterite and tremolite—are developed in the limestone. These minerals and the fine-grained carbonates occur only in the crystalline limestones lying beneath the dikes, the hanging wall remaining practically unchanged except for an increase in the size of grain and the quantity of lime silicates within 6 in. of the contact.

The carbonate rock below the dikes is more resistant to both weathering and corrosion; it is finer in grain, tougher and harder than the average crystalline limestone, and has a dense appearance. Hand specimens do not show twinning of the carbonates. Chondrodite occurs in bands as small yellow grains. Phlogopite is found as pale yellow flakes, which sometimes have a good crystal outline but more often occur between irregular carbonate grains. Scapolite forms irregular crystals both within and between the carbonates. Diopside and forsterite occur as irregular grains; while tremolite is found in long fibers. These minerals are usually altered to serpentine. Pyrope occurs as grains surrounded by a kelyphite rim. Sulfides, such as pyrrhotite and chalcopyrite, are common, together with these silicates and denser carbonates. Graphite is almost entirely lacking in this type of rock.

A detailed study and examination of the limestones along the West River, on range IX of Chatham township, was undertaken. Representative samples of fresh rock were chipped from selected areas, about 1 ft. square, both above and below, at a distance from and also near to the dikes of different ages. Analyses of these are given in Table 1; the points from which these samples were taken are marked on the map, Fig. 2.

TABLE 1.—*Analyses of West River Limestone*

Number of Sample	Insoluble,[a] Per Cent.	$Fe_2O_3 + Al_2O_3$, Per Cent.	$CaCO_3$, Per Cent.	$MgCO_3$, Per Cent.	Number of Sample	Insoluble, Per Cent.	$Fe_2O_3 + Al_2O_3$, Per Cent.	$CaCO_3$, Per Cent.	$MgCO_3$, Per Cent.
53	5.7	0.1	91.7	0.8	27	17.34	1.30	60.2	21.15
49	29.3	0.2	69.1	0.0	13	8.2	0.20	87.9	3.50
51	37.3	0.1	62.6	0.0	7	14.6	2.05	59.2	22.15
52	26.1	1.2	72.0	0.0	50	23.2	0.50	76.0	0.00
30	18.5	0.7	70.6	10.15	12	16.4	2.20	81.6	0.50

[a] Insoluble includes all the silicate minerals.

Sample 53 was clear blue limestone from above the large dikes of Laurentian age.

Sample 49 was white limestone 40 ft. from the large dikes of Laurentian age and situated on the foot-wall side.

Sample 51 was the same limestone as No. 49 but was situated about 60 ft. from the dike.

Sample 52 was the same limestone as No. 49 but was situated about 75 ft. from the dike.

Sample 30 was hard white limestone 1 ft. below the dikes of Laurentian age.

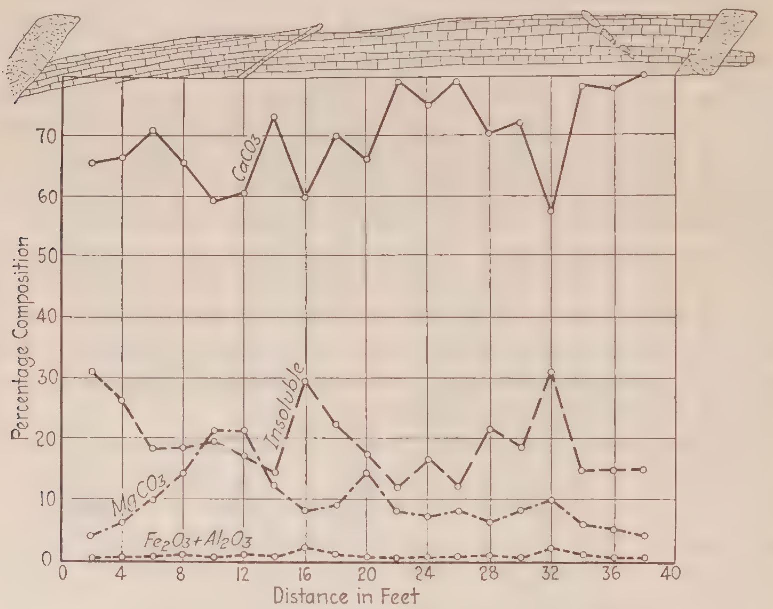

FIG. 3.—ANALYSIS OF FIRST LIMESTONE BED.

Sample 27 was the same limestone but 3 ft. below the same dike.

Sample 13 was hard white limestone taken from another bed 1 ft. below a dike of Laurentian age.

Sample 7 was the same limestone as 13 but was taken 3 ft. below the dike.

Sample 50 was a greenish blue limestone from below the hornblende syenite dike connected with the Grenville stock.

Sample 12 was an inclusion in one of the Laurentian dikes.

Two beds, which were widely separated but seemed to represent the limestones as a whole, were sampled to determine the distribution of the magnesium carbonate. The samples were cut as evenly as possible and only fresh rock was included. The chips were gathered for every 2 ft. and placed in canvas sacks to prevent loss. The first channel began beneath a large aegirite syenite dike and followed the bed across a pegmatite dike to a similar dike. The second channel began at the last named dike and ran to the edge of the stream. The samples were analyzed in duplicate and the results are given in Table 2, and are plotted in Fig. 3.

TABLE 2.—*Analyses of First Limestone Bed*

Number of Sample	Distance, Feet	Insoluble,[a] Per Cent.	$Fe_2O_3 + Al_2O_3$, Per Cent.	$CaCO_3$, Per Cent.	$MgCO_3$ Per Cent.	Number of Sample	Distance, Feet	Insoluble, Per Cent.	$Fe_2O_3 + Al_2O_3$, Per Cent.	$CaCO_3$, Per Cent.	$MgCO_3$ Per Cent.
32	0– 2	30.50	0.55	65.35	4.32	22	20–22	12.35	0.90	78.70	8.65
31	2– 4	26.19	1.04	66.10	6.70	21	22–24	16.74	1.14	74.35	7.05
30	4– 6	18.50	0.70	70.60	10.15	20	24–26	12.35	0.90	79.00	8.20
29	6– 8	18.59	1.04	65.75	14.65	19	26–28	21.80	1.81	70.20	6.32
28	8–10	19.50	0.70	59.30	21.50	18	28–30	18.80	1.20	71.50	8.70
27	10–12	17.34	1.30	60.20	21.15	17	30–32	31.20	2.10	57.40	10.10
26	12–14	14.50	1.00	73.00	12.40	16	32–34	15.10	1.15	78.00	6.35
25	14–16	29.69	2.19	59.40	8.79	15	34–36	15.16	0.96	77.50	5.64
24	16–18	19.00	1.50	70.00	9.20	14	36–38	15.00	1.00	79.00	4.80
23	18–20	17.34	1.04	66.25	14.70						

[a] Insoluble includes all the silicate minerals plotted in Fig. 3.

In studying these and the following analyses, the following points must be kept in mind: (1) The beds unless otherwise stated are of uniform thickness over the portions sampled. (2) The beds dip about 10° to 30° west, as shown in Fig. 2, and meet the dikes at an acute angle. (3) The beds are cut by numerous pegmatite and aegirite syenite dikes, which have a selvage of silicate minerals and coarsely crystallized calcite.

The analyses of samples 32, 31, 25, 24, and 17 show a high insoluble content; this is caused by the selvage of silicate minerals along the dikes. From 0 to 10 ft. and between 16 and 20 ft., the calcium carbonate increases to a maximum. In both cases, the maximum magnesium-carbonate content occurs at a distance from the dike proportional to the thickness of the dike or to the width of the selvage of silicate minerals and recrystalized zone along the edge of the dike.

Specimens corresponding to samples 28, 20 and 15 were collected and sliced for examination under the microscope. The section corresponding to sample 28 showed the rock to be inequigranular and to consist of interlocking grains of calcite and dolomite with a few more or less rounded grains of chondrodite, scapolite, pyrrhotite and pyrope and of elongated flakes of phlogopite and graphite. The carbonate grains range in size from 0.2 to 2 mm. or more in diameter. When examined between crossed nicols, the fine-grained areas are seen to consist of a number of connected grains having simultaneous extinction and twin lamellæ broken by a number of smaller grains that extinguish at various inclinations to one another. As the enclosing crystals extinguish simultaneously and as the twinning lamellæ, although broken, are in line, they seem to belong to a single grain that has been twinned and in which the smaller grains have developed at a later state. Treating a polished section of the rock with Lemberg's solution showed that the enclosing crystal was calcite and the smaller untwinned grains were dolomite.

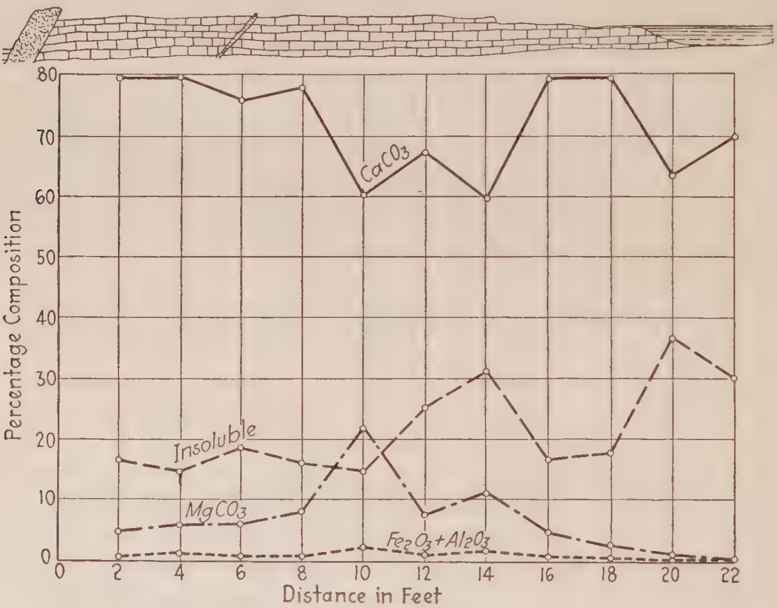

FIG. 4.—ANALYSIS OF SECOND LIMESTONE BED.

Elongated black opaque flakes of graphite and similar flakes of nearly colorless phlogopite occur between the carbonate grains and more often within them; neither of these minerals occurs within or penetrating the silicates. Chondrodite occurs in rounded grains, ½ mm. or less in diameter showing polysynthetic twinning and occurring between the carbonates. These grains are traversed by cracks filled with a pale yellow serpentine; a parting of magnetite occurs down the center of the serpentinized cracks. Scapolite occurs as irregular rounded grains between the carbonate. Pyrope forms irregular to rounded grains, from about ½ to 1 mm. in diameter, and is surrounded by a kelyphite rim equal in width to the diameter of the grains. The garnet is slightly altered to a chloritic mineral along fractures through it.

The rock corresponding to sample 20 is similar to that just described except that the calcite is coarser textured and contains fewer dolomite grains.

The section corresponding to sample 15 shows essentially the same features as the others, but the calcite grains are slightly larger and there is a decrease in the amount of dolomite, chondrodite, and scapolite, and an increase in phlogopite.

The series of analyses of the second bed, Table 3, is shorter than that of the first but the results of the two series are in general agreement. The results are plotted in Fig. 4.

TABLE 3.—*Analyses of Second Limestone Bed*

Number of Sample	Distance, Feet	Insoluble,[a] Per Cent.	$Fe_2O_3 + Al_2O_3$, Per Cent.	$CaCO_3$, Per Cent.	$MgCO_3$, Per Cent.	Number of Sample	Distance, Feet	Insoluble, Per Cent.	$Fe_2O_3 + Al_2O_3$, Per Cent.	$CaCO_3$, Per Cent.	$MgCO_3$, Per Cent.
11	0– 2	16.25	0.50	78.80	4.81	5	12–14	30.05	1.30	58.85	10.75
10	2– 4	14.50	0.65	79.20	5.68	4	14–16	16.25	0.60	78.50	4.47
9	4– 6	18.25	0.65	75.00	5.99	3	16–18	17.40	1.15	78.45	2.80
8	6– 8	15.80	0.90	76.70	7.77	2	18–20	36.60	1.40	61.85	0.80
7	8–10	14.60	2.05	59.20	22.15	1	20–22	30.00	0.25	69.70	0.07
6	10–12	24.75	0.95	66.10	7.95						

[a] Insoluble includes all the silicate minerals.

The work was now carried to the far side of the river to see if the same relations held when the dike was followed along its strike in the other direction beyond the younger dike.

Analyses showed that there was no magnesium carbonate above both the younger and the older dikes and that beneath the older dike magnesium carbonate occurred as has been shown for other beds. However a number of interesting structures were observed and studied. The limestones are folded in an anticline, which plunges at a steep angle upstream, or to the north. The beds of this anticline are quite distinct and are thicker on the crest than on the limbs in about the proportion shown in the diagramatic cross section above the curve representing the analyses, Fig. 5. The arch of the beds has been undermined by the river, giving an excellent longitudinal as well as lateral section, but the most interesting features are beds of hard massive limestone similar to the high magnesium-carbonate facies, already studied, traversed by three fissures at right angles to the bedding. These fissures extend down to a layer of biotite gneiss or schist about 6 in. thick. The first of these fissures is about ¾ in. wide and is filled by plates or scales of talc crystallized perpendicular to the walls. This fissure is about 12 ft. from the dike. The other two fissures, occurring 22 and 30 ft. from the dike, are merely gliding planes of small movement but for

⅛ in. on one side and about ½ in. on the other, they are colored green by an abundance of serpentine.

As so many interesting structures were shown and as the rocks appear to be highly magnesian, samples were taken, in the same manner as in the other localities, both from a bed across the anticline and from the same bed down the plunge of the structure. A large specimen containing one of the gliding planes and the accompanying serpentinized zone, as well as 8 in. of the carbonate, was obtained. The samples were analyzed in duplicate. The samples taken across the fold yielded the results given in Table 4 and plotted in Fig. 5. From the analyses, it will be seen that in general the greatest magnesium-carbonate values are 20 to 24 ft. from the dike, which also marks the crest of the fold. It is also to be noted that the values are higher near the magnesium-silicate veinlets.

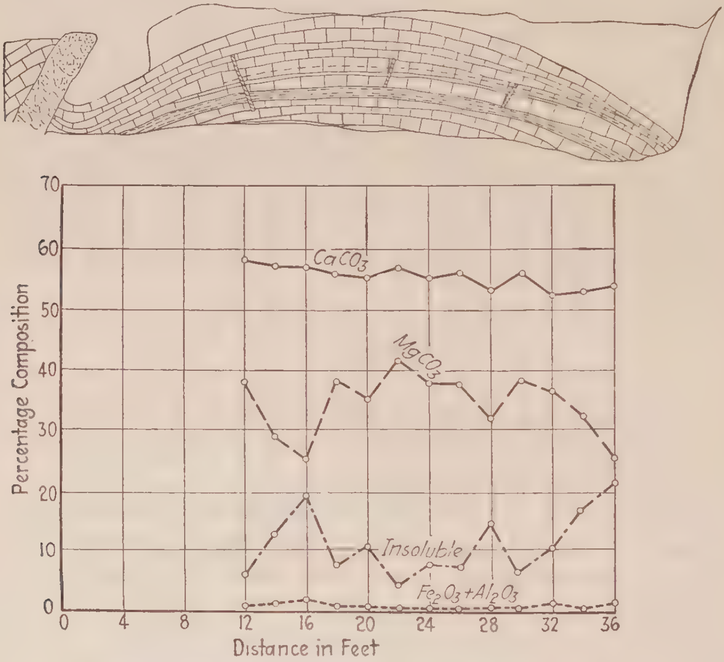

FIG. 5.—LIMESTONE BED ON FAR SIDE OF RIVER.

The next point to be considered is the variation of the magnesium carbonate down the plunge of the fold. The analyses and sampling were carried out as usual, the line of samples running from the channel sample across the fold to the water's edge. The results, given in Table 5, show a continuous decrease in magnesium carbonate and increase in insoluble as the bed is followed down the plunge of the anticline.

The last series of analyses apparently disprove, or are out of accord, with the results previously obtained; namely, that the maximum magnesium carbonate occurs 10 ft. along the bed, or about 2 ft. vertically below the dike, for here the maximum magnesium carbonate occurs 22 ft. from the dike; also the percentage of magnesium carbonate is not constant for a given distance from the dike, because the samples taken down the plunge are at equal distances from the dike but they vary greatly.

TABLE 4.—*Analyses of Limestone Bed on Far Side of River*

Number of Sample	Distance, Feet	Insoluble,[a] Per Cent.	$Fe_2O_3 + Al_2O_3$, Per Cent.	$CaCO_3$, Per Cent.	$MgCO_3$, Per Cent.	Number of Sample	Distance, Feet	Insoluble, Per Cent.	$Fe_2O_3 + Al_2O_3$, Per Cent.	$CaCO_3$, Per Cent.	$MgCO_3$, Per Cent.
48	12–14	5.40	0.75	57.00	37.20	41	26–28	7.25	0.39	55.7	37.00
47	14–16	12.69	1.10	56.10	29.10	40	28–30	14.30	0.75	52.7	31.25
46	16–18	18.90	1.70	55.50	24.50	39	30–32	6.46	0.36	55.8	37.90
45	18–20	7.70	0.50	55.00	37.60	36	32–34	10.40	1.03	51.1	36.00
44	20–22	10.60	0.75	54.62	34.50	35	34–36	16.50	0.65	51.9	31.90
43	22–24	4.14	0.24	55.50	40.95	34	36–38	20.50	1.60	52.0	25.60
42	24–26	7.85	0.65	54.60	37.10						

[a] Insoluble includes all the silicate minerals.

TABLE 5.—*Analyses Down Plunge of Fold*

Number of Sample	Distance, Feet	Insoluble,[a] Per Cent.	$Fe_2O_3 + Al_2O_3$, Per Cent.	$CaCO_3$, Per Cent.	$MgCO_3$, Per Cent.
39	0–2	6.46	0.36	55.80	37.90
38	2–4	8.66	0.66	54.00	36.80
37	4–6	12.91	1.26	54.10	31.30

[a] Insoluble includes all the silicate minerals.

As the magnesium carbonate occurred only on the foot-wall side of the dikes, perhaps the dikes formed a structure favorable to the formation of magnesium carbonate. The magnesium carbonate increased near the serpentine and talc veins, so the large specimen, containing the vein and the adjacent serpentinized zone as well as the adhering carbonate, was analyzed. The results, given in Table 6, show that the vein and adjacent wall rock contain a high percentage of silicates but that farther out the magnesium carbonate reaches a maximum and then begins to decrease. From this, it would appear that fissures mineralized by magnesium silicates cause an increase in the magnesium carbonate.

TABLE 6.—*Analyses of Specimen-containing Talc and Serpentine Veins*

Number of Sample	Distance, Inches	Insoluble,[a] Per Cent.	$Fe_2O_3 + Al_2O_3$, Per Cent.	$CaCO_3$, Per Cent.	$MgCO_3$, Per Cent.	Number of Sample	Distance, Inches	Insoluble, Per Cent.	$Fe_2O_3 + Al_2O_3$, Per Cent.	$CaCO_3$, Per Cent.	$MgCO_3$, Per Cent.
1		68.45	1.20	17.85	12.70	6	2 –2½	1.30	0.50	56.00	42.80
2	0 – ½	18.40	1.50	46.50	34.20	7	2½–4½	1.70	0.40	55.70	44.00
3	½–1	8.50	1.75	51.50	37.80	8	4½–6½	2.10	0.25	55.50	43.50
4	1 –1½	4.35	0.65	54.70	39.75	9	6½–8½	4.14	0.24	55.50	40.95
5	1½–2	2.00	0.60	55.00	41.20	10	8½–24	7.85	0.65	54.60	37.10

[a] Insoluble includes all the silicate minerals.

The serpentine veins were examined both in the hand specimen and in thin sections. One of the veins was serpentinic locally and the thin sections examined were taken from a portion low in serpentine; the second vein was largely serpentinous along the fissure. Sections of both the vein and the adjacent rock were prepared for study. The type of vein first mentioned will be described first. Three sections were prepared in each instance: first a section of the vein, second a section ½ in. from the vein, and third a section 1 in. from the vein. A section of the rock 1 in. from the vein showed that the specimen consisted of small irregular interlocking grains of dolomite and some larger twinned crystals of calcite surrounding the latter. Occasionally the twinning plane can be recognized in the dolomite crystal by the presence of a foreign silicate but cannot be caused by twinning in the dolomite, for the axes of the dolomite

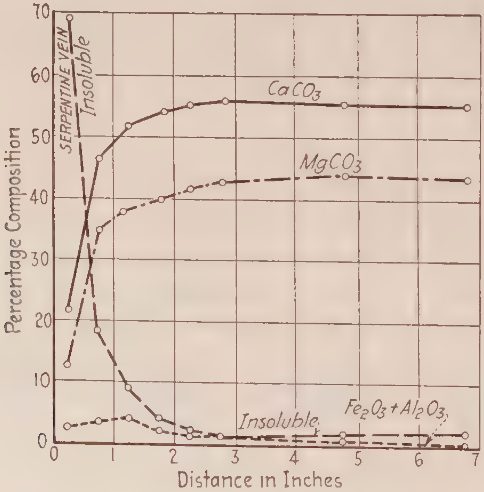

FIG. 6.—VARIATION FROM SERPENTINE VEINS OUTWARDS.

are almost invariably inclined to those of the calcite. Apparently the twinning in the calcite antedates the formation of the dolomite grains. Scapolite occurs as fresh rounded or irregular grains, from 0.05 to 0.5 mm. in diameter, intersticial to or surrounded by carbonates. Small grains of twinned chondrodite, flakes of graphite and phlogopite, and rounded grains of pyrrhotite occur intersticial to or surrounded by the carbonates. A section taken ½ in. from the vein showed essentially the same features as those just described, except that silicates are present in greater amount. A section of the vein itself shows an abundance of magnesian silicates that have been altered to serpentine.

The analyses were plotted on curves, Fig. 6, which showed that high magnesium carbonate corresponds to low insoluble and vice versa. This suggests that the silicates might be magnesian when the magnesium carbonate was low and lime silicates when the magnesium carbonate was high. Complete analyses were made, corresponding to the analyses already given, to determine whether the magnesia varied or whether the apparent variation was the result of recrystallization of the limestone.

The vein of the second series is of a more serpentinized nature and contains forsterite in addition to the silicates mentioned. The calcite and dolomite show an excellent intergrowth or pseudo-eutectic structure. Complete analyses of specimens corresponding to those of Tables 2 and 3 are given in Table 7.

A comparison of these analyses shows that the MgO and the $MgCO_3$ vary in the same way and there is no evidence, from the chemical analyses or the thin sections, that the silicate is formed by the recrystallization of the limestone; on the other hand, they tend to show a regular variation in silica, alumina, iron oxides, magnesia, and lime which could not have been possible in the original sediments.

TABLE 7.—*Complete Analyses of Selected Samples*

	First Series					Second Series		
	4–6 Ft.	8–10 Ft.	14–16 Ft.	20–22 Ft.	36–38 Ft.	8–10 Ft.	14–16 Ft.	20–22 Ft.
Sample No.	30	28	25	22	14	7	4	1
SiO_2	6.45	7.00	9.67	4.97	5.93	6.00	6.65	16.05
$FeO + Fe_2O_3$	0.62	1.11	0.62	0.31	0.51	0.42	0.42	1.54
Al_2O_3	2.60	1.95	8.44	3.49	1.96	3.33	2.63	0.14
CaO	39.60	33.10	34.35	44.40	46.40	35.83	44.91	46.65
MgO	13.55	20.84	13.61	7.51	7.32	14.90	5.00	5.25
$CO_2 + H_2O$	36.15	35.50	31.39	38.44	38.44	39.35	37.45	30.80
K_2O	1.45	0.50	1.40	0.57	0.00	Not determined		0.00
Na_2O	0.13	0.05	0.13	0.09	0.00	Not determined		0.00

An analysis of the serpentine vein gave the following composition:

	PER CENT.		PER CENT.
SiO_2	40.37	CaO	28.52
$FeO + Fe_2O_3$	0.31	MgO	17.10
Al_2O_3	3.24	$CO_2 + H_2O$	10.83

The analysis seems to show that the vein is merely a dolomitic limestone with an unusually large amount of silicates and this evidence is confirmed by the microscopic examination which has already been given.

ORIGIN OF SOLUTIONS

Chemical analyses and petrographic study show that the highest concentration of magnesium carbonate occurs on the crest of a plunging anticline, especially near serpentine veins that pass down to a thin layer of biotite gneiss or schist but do not pass through it. When followed upwards, the veins pinch out, after a few feet, in the crystalline limestone.

The biotite gneiss has a foliated appearance and, were it not for the large amount of quartz and feldspar present, might be classed as a schist. A striking feature is the mottled surface caused by the irregular distribution of garnets, about 5 mm. in diameter. Pegmatite dikes from the granite intrude the gneiss. Even microscopically the foliated appearance is marked. The texture is inequigranular, lacks uniformity, and the mineralogical composition is very irregular. An average of several sections showed that the rock contains about:

	Per Cent.		Per Cent.		Per Cent.		Per Cent.
Biotite	25	Orthoclase	12	Sillimanite	5	Apatite	2
Quartz	25	Albite	5	Carbonates	3	Diopside	
Chlorite	15	Almandite	5	Magnetite	3	Sericite	

Almandite is the oldest mineral and was once the main constituent of the rock; it is badly fractured and altered to magnetite and chlorite along the cracks. Some areas are composed largely of chlorite, garnet, and biotite, while in other places the chlorite has been largely replaced by biotite, which is occasionally bleached and has the appearance of muscovite. Sillimanite occurs in well-developed prisms 0.05 to 0.5 mm. in diameter. The orthoclase is highly sericitized and biotite, sillimanite, and quartz, which appear to have been the last formed, are the only fresh minerals in the rock.

If the gneiss was reconstructed so that the quartz, feldspar, and carbonate were allowed to remain in their present form, but so that the iron oxides, alumina, and silica of the biotite, chlorite, magnetite, and sillimanite were converted to the almandite from which they were derived, there would still remain the magnesia and alkalies of the biotite and chlorite to be accounted for. The alteration of orthoclase to sericite and garnet to chlorite usually takes place at great depths and under the influence of heated solutions. These solutions seem to have carried magnesia and alkalies, which acting on the almandite, produced chlorite and biotite until all the iron and alumina of the garnet were used up. Apparently there was still an excess of magnesia and alkalies because, when the waters moved along the fissures above the gneisses, they deposited magnesian and alkali minerals.

The biotite gneisses are found only at the contact with the intrusive granite. The granite magma was heavily charged with mineralizers; evidence of these is found in the numerous pegmatites and the injection perthites of the granite itself. These heated waters would be powerful agents of alteration but no matter how much alteration they produced they could not change almandite to biotite without supplying the necessary magnesia and alkalies. Now whether the waters and vapors acted merely as agents of transportation of the magnesia and alkalies or whether they brought them in solution from the magma itself is a question which is hard to decide. The fact that the granite contains more magnesian minerals in some places than others, especially when those places are within the interior of the batholith, may be taken to indicate that the magma contained more magnesia than is contained in the marginal phases of the granite and that this lack is due to the amount removed by the magnesia-bearing waters.

Structures of Magnesium-carbonate Deposits

Up to the present two structures have been pointed out as being associated with the magnesium-carbonate content of the limestones; these are: Inclined dikes cutting across the beds (monoclines) and anticlinal folds and possibly domes. In the first case, the deposits occur below a dike, which serves as an inverted trough. As the deposits occur on the foot-wall side, it is to be concluded that the solutions which brought about the replacement must have moved along the foot-wall side and not along the hanging-wall. If the waters were descending, they would tend to descend vertically and on the foot-wall side would get farther and farther from the dike or accumulate above the dike, in which case the deposits would be expected in the hanging-wall side. On the other hand, if the waters were rising they would tend to ascend vertically and those rising far from the dike would soon be brought near it; that is, the waters would tend to rise immediately below the dike and produce deposits along the foot-wall side. The same result is produced when the structure is an anticline instead of an inclined dike. The highest magnesium-carbonate values occur where the greatest flow of waters occurred; that is on the crest, and especially near fractures where the waters have risen more readily from the schists.

Another interpretation of the results is that instead of depositing magnesia and alkalies, the waters might have removed the lime and left the $MgCO_3$ with the silicates of the original limestone behind. Such a change would necessitate a shrinkage of the rocks proportional to the increase in magnesia. Likewise the magnesium carbonate and the silicates should increase in the same ratio. Such is not the case; not only is the rock not fractured by shrinkage but its volume has not changed and the magnesium carbonate and the silicates do not vary in the same proportion but in an inverse ratio.

As dolomite has a 12 per cent. lower molecular volume than calcite, processes of dolomitization near the surface would result in a shrinkage and destruction of the bedding accompanying a widespread occurrence of dolomitic limestone. But these conditions are not fulfilled. The beds are scarcely fractured at all and the dolomitization is highly localized. The only way in which such deposits as the one being studied are known to occur is by replacement at high temperature and considerable depth through the agency of magmatic waters with addition of the replacing material volume for volume of the material removed.

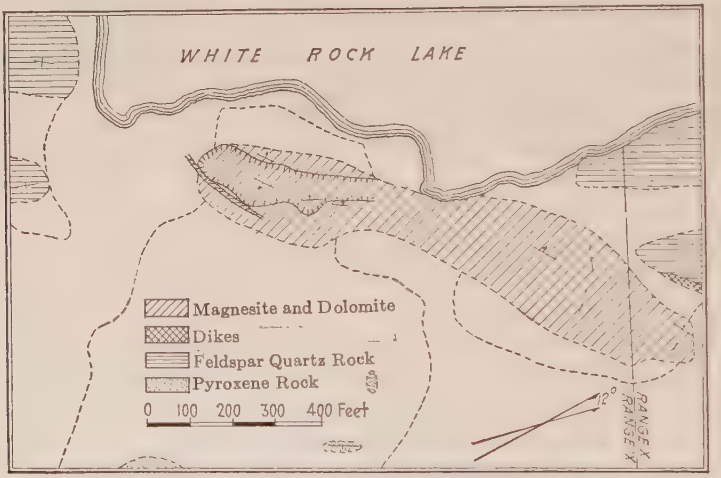

FIG. 7.—NORTH AMERICAN MAGNESITE QUARRY.

MAGNESITE DEPOSITS

In the study of the magnesite deposits, the conditions were found to be merely an advanced stage of those already described. The field work was carried on at the North American magnesite quarry at White Rock Lake, Grenville township, Fig. 7. The property is worked as an open pit about 300 ft. long, 150 ft. wide, and 80 ft. deep. Magnesite containing less than 5 per cent. $CaCO_3$ and low in silicates is obtained from two parallel bands. The tendency is for the $MgCO_3$ to decrease and for the silicates to increase with depth.

The Grenville series underlies a valley, about 1 mile wide, bounded on either side by hills of granite or biotite gneiss. White Rock Lake begins at the foot of a steep cliff of granite and overlies about ¼ mile between there and the quarry. Skarn rocks and magnesite with dolomite outcrop on the southern part of the east shore of the lake. The magnesite forms part of a stratified series bounded on either side by skarn rocks and folded into a close fan fold. The folded magnesite is intruded by a dike about 3 ft. thick which strikes N. 45° E. and dips about 7° to the southeast. East of the quarry, skarn rocks and biotite gneisses are found; while still farther eastward are the intrusive batholiths and stocks of Laurentian and Algoman granite.

The so-called magnesite is not pure magnesite but dolomite with a large number of magnesite grains. This relationship can be studied by polished sections of the calcined material, which is peculiar in that the magnesite becomes a faint pink on giving up its CO_2 while the dolomite remains white. The magnesite bears the same relationship to the dolomite as the dolomite to the calcite in the deposits that have been described. The rock also contains large grains of a yellowish serpentine, which may have been derived from the alteration of chondrodite. Grains of sphalerite, galena, pyrrhotite, and chalcopyrite are not uncommon while flakes of phlogopite can be seen in almost any part of the rock. In one part of the quarry, a beautiful pink variety of magnesite low in $CaCO_3$ is found. The magnesite is exceedingly fine grained and glistening, compared with dolomite. The workmen use these properties in separating the waste from the valuable material.

The high-grade magnesite occurs in two bands, or "veins" as they are called by the miners, but these bands are really a bed that has been sharply folded and can be traced around on the end of the quarry. As in the case of the dolomite, the highly magnesian material does not occur immediately at the dike but from 5 to 40 ft. away. The quantity of magnesium carbonate decreases as the beds are followed northward along the strike.

The dike that cuts across the fold is slickensided and shows crushing, which probably occurred during the folding. It consists of serpentine having a mesh structure and of secondary magnesite, magnetite, and chalcopyrite. The rock beyond all doubt is a dike, as it breaks across the beds and contains angular xenoliths of the carbonate rock in itself. When a thin section is examined under the microscope, the rock is seen to be composed of irregular angular grains 0.6 mm. in diameter which are completely altered to serpentine. Sometimes the serpentine forms veins of a yellowish brown chrysotile, the fibers of which vary from ⅛ to ¼ in. in length. This cuts completely across the fold, producing a combination of the two structures at the Limestone Chute, and the magnesite is found at a point where each structure would tend to produce the highest magnesium carbonate. The higher magnesia content at White Rock Lake, compared with that at the Limestone Chute, on Range IX Chatham township, may be due to two factors: (1) A greater concentration of magnesia-bearing waters resulting from the combination of structures; (2) a greater initial supply of magnesia-bearing waters.

The structure mentioned under the first heading would tend to give a greater supply of magnesia-bearing waters so that the second factor may be merely complementary to the first and not a primary cause.

Conclusions

The observations of the occurrence of magnesite and dolomite, both as a mineral and as a rock, seem to warrant the following conclusions:

1. Deposits of dolomite and magnesite of this type can be expected only in the crystalline limestones.
2. The deposits are closely associated with paragneisses rich in ferromagnesian minerals and which have been contact metamorphosed and hydrothermally altered by Laurentian granite and syenite magmas.
3. The deposits occur only beneath impervious dikes and on the crest of anticlines or for a short distance down their flanks and plunge. Magnesite deposits are more apt to occur where a combination of the foregoing structures exists. These deposits cannot be expected to continue to great depths.
4. Dolomite is finer grained and more compact than calcite. Magnesite is finer grained than dolomite. Dolomite occurs as small grains replacing calcite volume for volume. Magnesite occurs as small grains replacing dolomite. If the rocks are considered in small units, the calcite must be completely replaced by dolomite before magnesite can be formed.
5. Some beds have been more completely replaced than others. This may have been caused by catalyzers but is more probably the result of greater porosity in the replaced bed.
6. The rocks consist of definite carbonates (calcite, dolomite, and magnesite) rather than isomorphous mixtures of calcium and magnesium carbonate.
7. The occurrence of the two carbonates dolomite and magnesite rather than an isomorphous mixture of calcium and magnesium carbonate lowers the value of the material as a refractory brick, as the calcined dolomite is apt to crumble or be fluxed leaving behind an incoherent mass of grains of magnesia.
8. The magnesia-bearing waters seem to have been given off by a granite magma during its final stages of consolidation.
9. The dolomites, when free from sulfides and ferrous minerals, should furnish an excellent material for the manufacture of artificial building stone. The magnesite containing less than 5 per cent. $CaCO_3$ and low in silica should make an excellent firebrick.

Cherts and Igneous Rocks of the Santa Elena Oil Field, Ecuador

By Joseph H. Sinclair and Charles P. Berkey,* New York, N. Y.

(Canadian Meeting, August, 1923)

This paper describes the results of a visit to the Santa Elena Peninsula, Ecuador, in January and February, 1921. On account of the complicated folding and faulting of the rocks and the peculiar occurrence of petroleum, field work was possible on only a few of the many geological problems and the results here published are only a contribution to the solution of the general problem. We know of nothing published on the geology of the district, except one account, in Spanish and German, by Theodor Wolf.[1]

The Santa Elena peninsula, the most important headland of the coast of Ecuador, lies two degrees south of the equator. The chief town, Santa Elena, is about 75 miles west of Guayaquil, the main seaport of Ecuador. The nearest developed oil field is the Zorritos of northern Peru, 120 miles south across the Gulf of Guayaquil. If newspaper reports are true, *viz.* that the Anglo-Ecuadorian Oilfields, Ltd., has recently brought in a 700-bbl. well on the coast of the peninsula near Ancon, the district may become important in the production of petroleum. At the time of our visit, the best way to reach the peninsula was by a steamer of the Pacific Steam Navigation Co., southward from Panama and northward from Guayaquil. About once a week, one of these steamers anchored ½ mile off shore near the village of Salinas, an important cable station. An alternative route, in the dry season, is by automobile from Guayaquil. In February, 1921, a railway bed had been graded part way between Guayaquil and Salinas, but this was deteriorating rapidly because of lack of funds to continue construction or to keep in repair that portion already completed.

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* The field work is that of the first-named author; the second-named is responsible for the petrographic descriptions.

[1] Theodor Wolf: Geografia y Geologia del Ecuador, 1892.

The part of the peninsula here described is an irregularly shaped area extending about 22 miles eastward, from the lighthouse to the mud

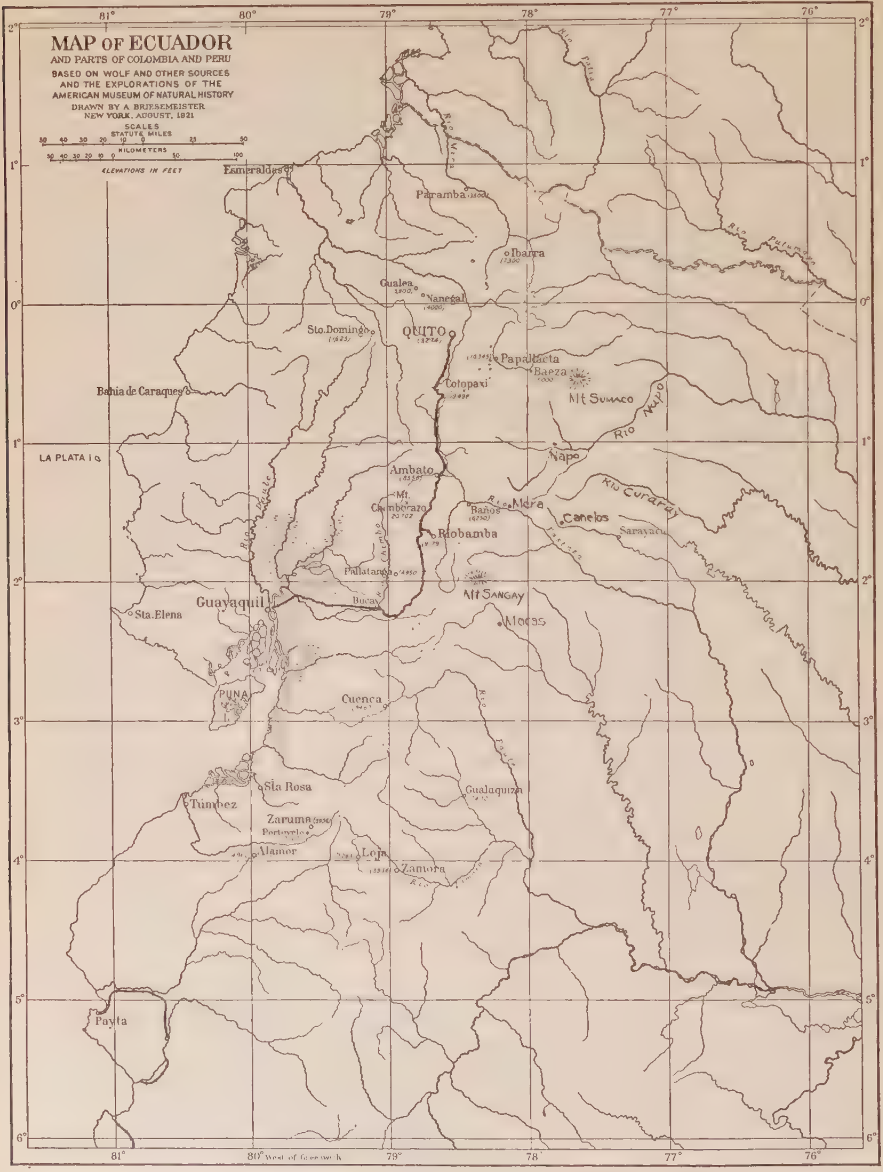

FIG. 1.

volcano 5 miles east of the village of San Vicente, and from Punta Centinela on the north coast about 12 miles to a point on the south coast about 5 miles east of the fishing village of Ancon. The most striking feature is the finger-shaped point, averaging less than 1 mile in width and nearly 7 miles in length, which extends into the Pacific. In addition, there are three minor headlands, the Ancon and Punta Carnero on the south coast, and Punta Centinela, on the north coast.

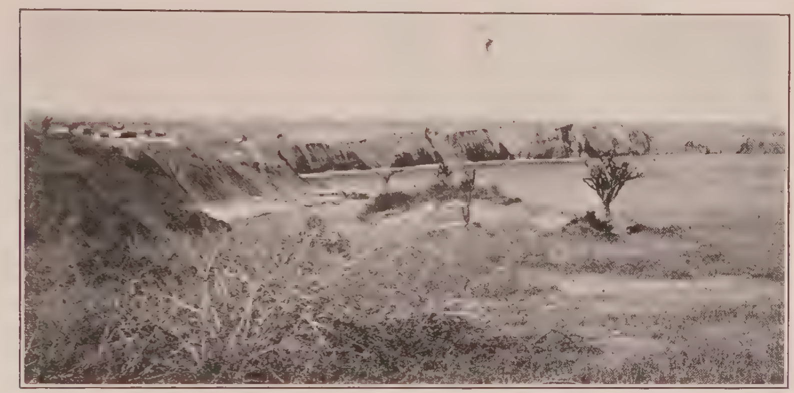

Fig. 2.—Village of Ancon; cliffs nearly 200 ft. high. Semiarid conditions shown by cactus in foreground

The district is arid, treeless, fairly cool, and healthful; there is no cultivation of the soil except where it can be irrigated from wells. The population is small and limited to the vicinity of the wells. The chief industries are fishing, the making of Panama hats, the extraction of salt from the salt pans at Salinas (a Government monopoly), the collection of petroleum from the numerous pits, and the working of the cable station at Salinas, where a number of American and English employees reside. The largest town, Santa Elena, is the seat of the local government of the district and also possesses the only church in the region. Salinas has

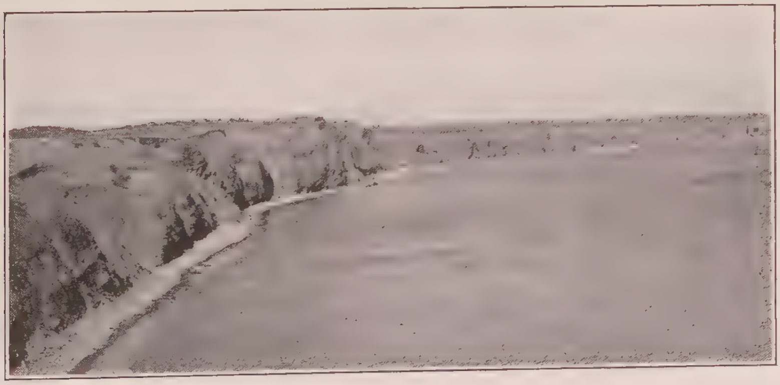

Fig. 3.—Cliffs in Tertiary sediments east of Ancon, on shore at extreme right are two wells of Anglo-Ecuadorian Oilfields, Ltd.

two hotels and a few summer homes. The population of neither Salinas nor Santa Elena exceeds a few hundred; the other hamlets consist of only a few Indian huts. There is telephonic and telegraphic communication between Salinas and Guayaquil. Automobiles can be rented at Salinas and one can drive all over the peninsula at any time of the year and as far east as Guayaquil, except in the rainy season, when the road is impassible near Guayaquil.

The western point of the peninsula is the dominating physiographic feature of the coast; it is an isolated, scarped, terraced, flat-topped hill, called La Puntilla, the summit of which, 424 ft.[2] above the sea, can be seen for many miles in every direction. Immediately southeast of this hill, the land lies almost at sea level, where Salinas and the cable station are situated. Eastward, the land rises gently, Santa Elena, 8½ miles from Salinas, being about 108 ft. above the sea and Volcancitos, 11 miles east of Santa Elena, about 250 ft.

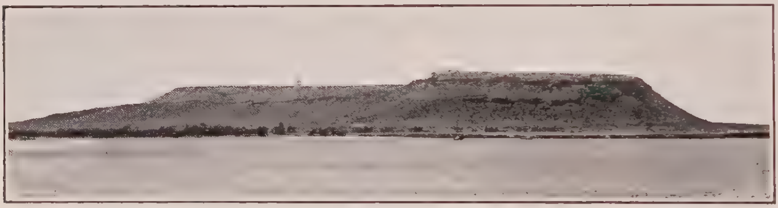

FIG. 4.—LA PUNTILLA; ELEVATION 424 FT.; ELEVATION OF LIGHT 467 FT. NOTICE TERRACE 30 FT. ABOVE THE SEA AND REMNANTS OF OTHERS ON SLOPES OF THE HILL; IN PHOTOGRAPH TERRACES RESEMBLE STRATA. THE HILL IS FLAT TOPPED AND COMPOSED OF CRETACEOUS CHERTS HIGHLY CONTORTED AND WITHOUT SIGNS OF STRATIFICATION; TOP CONTAINS SHELLS, PROVING THAT IT HAS RECENTLY BEEN UPLIFTED FROM THE SEA.

The surface is thought to be a remnant of marine erosion recently elevated above the sea. From the presence of marine Pleistocene shells on the summit of La Puntilla, it would seem that there had been at least 424 ft. of recent uplift. The generally flat surface is modified by only one valley, Rio Grande de Salada, which is very shallow and contains water only for a short time after rain. This dry stream bed enters the area in the vicinity of Volcancitos and reaches the sea 1 mile west of Punta Carnero.

As would be expected in a region of such recent uplift, where movement is probably still continuing, there is considerable development of sea cliffs. These are especially well shown on the south coast between Punta Carnero and Chanduy.

STRATIGRAPHY

The rocks of the peninsula may be divided into four groups: brecciated cherts, probably of Upper Cretaceous age; igneous rocks of limited extent intruding the cherts; a thick series of sandstones and shales of Tertiary age; and a thin formation of broken marine shells of Pleistocene age, probably the equivalent of the Tablazo of Peru.

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[2] Chart No. 1123, Hydrographic Office, U. S. N., Washington, D. C.

The Chert Series

Neglecting a thin cover of Pleistocene beds, nearly one-half of the peninsula is made up of chert. La Puntilla is entirely chert. After a break about 1 mile wide, in the vicinity of Salinas, the cherts continue eastward over an unbroken area, 17 miles long and approximately 2 miles wide, extending to a point about 4 miles east of San Vicente. It is quite possible that they constitute the prominent ridge which runs eastward as far as Guayaquil, 85 miles distant, where similar rocks occur.

Nine specimens of the brecciated cherts were collected from widely separated parts of the peninsula and from these, thin sections were made. Photomicrographs are shown in Figs. 5 to 14.

Specimen E-C, Figs. 5 and 6, from the cliffs on the seashore below La Puntilla, is a reddish dense cherty limestone. Under the microscope it is microcrystalline, extremely veined, and is essentially a cherty infusorial limestone. The rock is not badly brecciated. Microscopic fossils are exceedingly abundant and well preserved and evidently made up almost the whole rock in its original condition. It was then probably essentially a carbonate, but has subsequently become opaline and cherty. This specimen preserves the original habit the best of any of the series.

Specimen E-23, Fig. 7, from the extreme western point of the peninsula, is a fine dense, red, jaspery, infusorial chert, with many microscopic fossils, mostly infusorial. It is chiefly chert stained with iron oxide and carrying some carbonate. The minute fossils are plainly preserved, in spite of almost complete silicification. This particular specimen is less brecciated than others of the series but in all other respects it is similar to them.

Specimen E-A, Fig. 8, from cliffs at the extreme western point of the peninsula, is a dark-colored, highly brecciated, veined chert derived from an infusorial rock. It is like those already described, except that it is more broken. The earlier cementing material is siliceous, but the later healing material is carbonate. Brecciation, therefore, seems to have been going on during the change in conditions that have given it two different healing effects. This same habit has been noted in one or two of the other rocks, notably E-20.

Specimen E-20, Fig. 9, from La Puntilla, on the extreme western point of the peninsula, is a typical chert breccia cemented by carbonate. The rock, before brecciation, was a cherty type developed from a fine-grained organic deposit. From this specimen alone, it would be difficult to judge the nature of the original formation but other members of the series complete all gradations between this particular type of material and

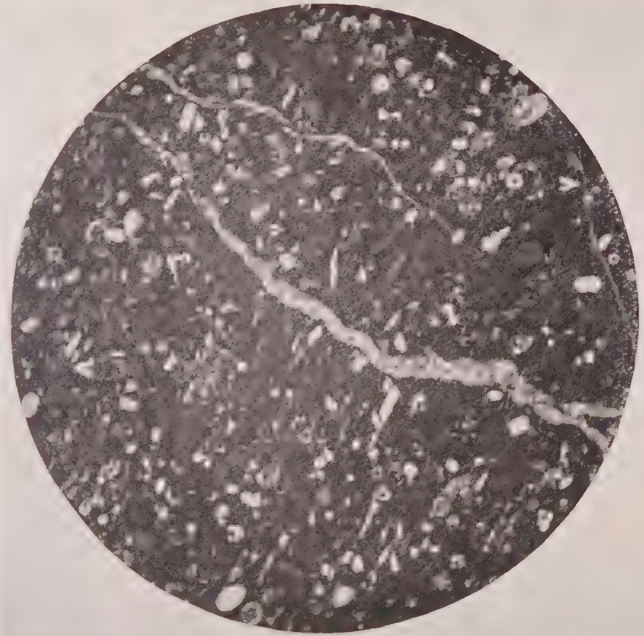


FIG. 5.—SPECIMEN E-C; INFUSORIAL CHERTY LIMESTONE; TAKEN WITH PLAIN LIGHT; × 30. STRUCTURE SIMILAR TO MANY OF FOLLOWING SPECIMENS, ESPECIALLY IN ITS EXCEEDINGLY FINE GRAIN, ABUNDANCE OF MICROÖRGANISMS, AND DEVELOPMENT OF VEINLETS; COMPOSITION LARGELY CARBONATE BUT CHERTY WHERE FOSSILS ARE BEST DEVELOPED AND IN SOME VEINLETS, BUT LATEST VEINLETS ARE CARBONATE.

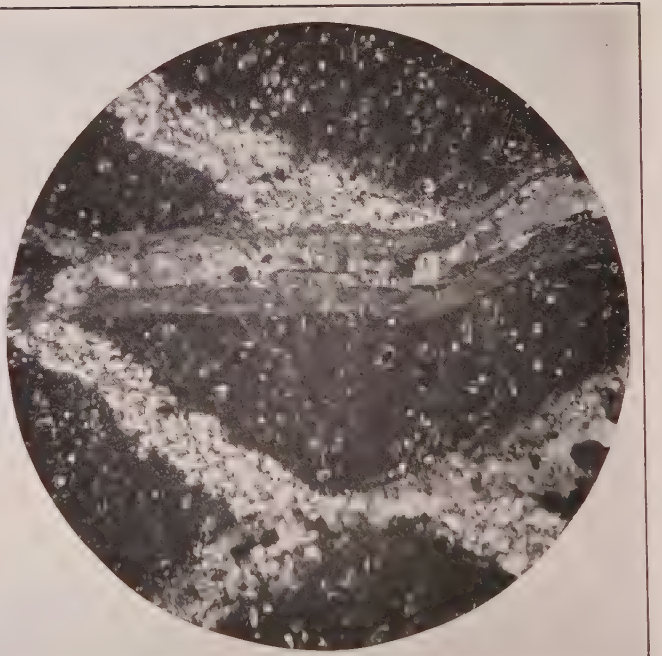


FIG. 6.—SAME SLIDE AS IN FIG. 5 BUT TAKEN WITH CROSSED NICOLS AT DIFFERENT SPOT TO SHOW TWO TYPES OF VEINLETS FORMED IN FRACTURES. CARBONATE VEINLETS CUT ALL OTHER STRUCTURES AND FINER GRAINED SILICA OR CHERTY VEINLETS ARE CUT BY CARBONATE. IT IS SIMILAR TO HISTORY SHOWN IN E-A. × 30.

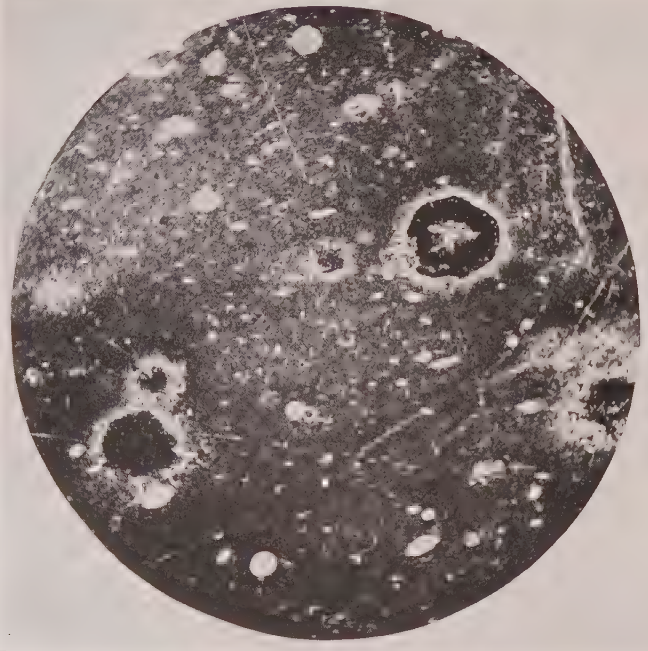


FIG. 7.—SPECIMEN E-23; JASPERY INFUSORIAL CHERT; TAKEN WITH PLAIN LIGHT; × 30. SHOWS VERY FINE-GRAINED JASPERY CHARACTER OF ROCK WITH NUMEROUS MICROÖRGANIC FORMS SCATTERED THROUGHOUT.

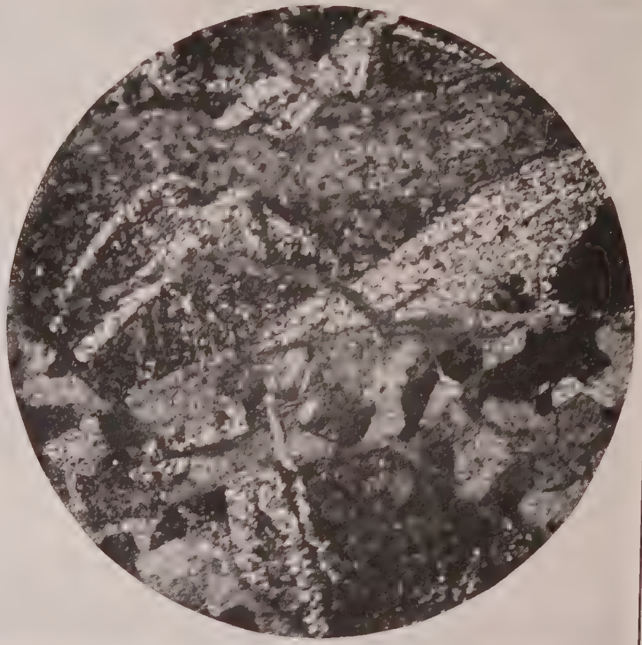


FIG. 8.—SPECIMEN E-A; BRECCIATED CHERT PROBABLY DERIVED FROM INFUSORIAL ROCK; TAKEN WITH CROSSED NICOLS TO SHOW DOUBLE HISTORY OF ROCK; LARGER UNIFORM ANGULAR AREAS ARE BRECCIATED CHERTY ROCK AND REPRESENT ORIGINAL CONDITION AS NEARLY AS IS PRESERVED; IT IS CUT BY VEINLETS OF VERY FINE AGGREGATE CHERTY QUARTZ, WHICH STOP ABRUPTLY AT MARGINS OF PRESENT BRECCIA FRAGMENTS, WHICH ARE BOUND TOGETHER BY CARBONATE. × 30.

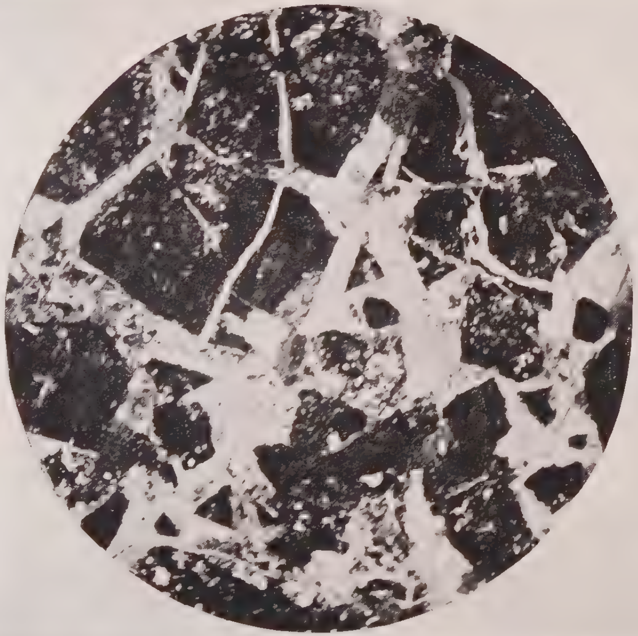


FIG. 9.—SPECIMEN NO. E-20; BRECCIATED CHERTY ROCK WITH ABUNDANT LIME VEINING; TAKEN WITH CROSSED NICOLS; × 30. SHOWS BRECCIATED CHARACTER OF ROCK; PRINCIPAL MASS IS FINE CHERTY MATERIAL, FRACTURE FILLING AND VEINLETS ARE CARBONATE; CHERTY PORTIONS SHOW BRECCIATION WITHIN THEMSELVES, AS IF BROKEN WHILE CHERTIFICATION WAS GOING ON.

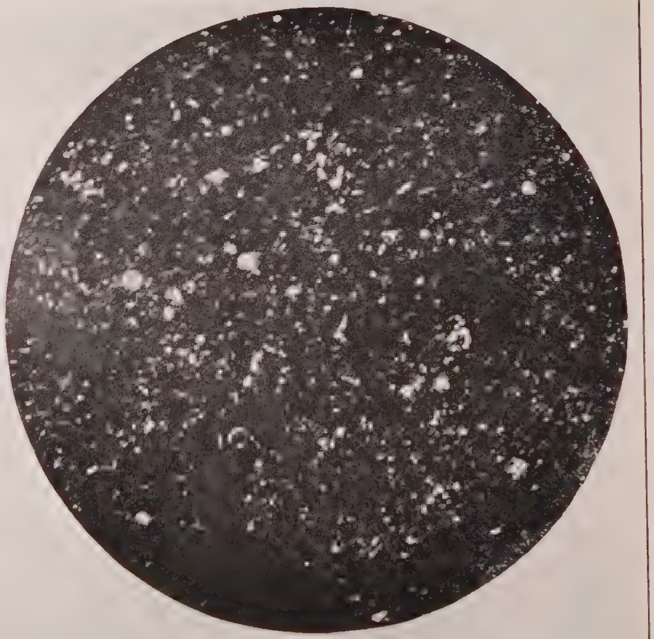


FIG. 10.—SPECIMEN NO. E-29; CHERTY INFUSORIAL ROCK; TAKEN WITH CROSSED NICOLS; × 30. SHOWS EXTREMELY FINE GRAINED ROCK; VISIBLE CONSTITUENTS ARE MINUTE SPECKS AND FLAKES OR RODS, BUT SCATTERED DEFINITE ARRANGEMENTS OF THEM SUGGEST A PRIMARY CONTROL, AS IN REPLACING AN OLDER STRUCTURE; SOME ARE CIRCULAR OR RING-LIKE INDICATING FORMER PRESENCE OF MINUTE FOSSIL FORMS.

others in which microörganisms are prominent. In this specimen, the organisms are almost wholly obliterated by chertification and brecciation but otherwise it is like the other specimens. The rock probably was undergoing brecciation before chertification was completed. Thus, some of the fragments are complex, being themselves composed of breccia fragments cemented with chert. Later, carbonate became the healing material, rather than silica. All the later veinlets are carbonate.

Specimen E-B, also from the cliffs at the extreme western point of the peninsula, is a white, dense, weathered chert, of microfine texture. Its organic origin is much plainer than that of E-A and it shows a variety of fossils, chiefly foraminifera. The only difference between this sample and some of the more obscure cherty specimens is that the fossils are somewhat better preserved. The secondary structure is pseudomorphic and amorphous by replacement. Silica was the introduced substance and the rock may now be classified as an infusorial chert.

Specimen E-29, Fig. 10, from the isolated hill called "Carnero" on the south shore, is a white, exceedingly fine, close-textured, brecciated rock clearly of organic origin. The primary essential minerals were chiefly siliceous infusorial fragments. The rock is completely chertified but preserves multitudes of microörganic fragments of which it was originally composed.

Specimen E-44-D, Figs. 11 and 12, is from the second outcrop of cherts along the shore east of Salinas. Oil is obtained from neighboring pits in the same kind of rock. It is a dense, white, infusorial chert, of microfine texture; its original structure was organic. It is indurated, veined, and brecciated. The organic origin is proved by well-preserved fossils making a good connecting link between the specimens showing prominent fossil content and those fine cherty members in which the fossils have been destroyed.

Specimen E-22, Fig. 13, taken from an oil pit, is a greenish, brecciated, veined, fine-grained rock of microfine texture. Originally it was probably clastic, either ash or shale, now thoroughly silicified. The original minerals were quartz and probably a variety of other fine material, with pyrite. After deformation, carbonate, fluorite, and pyrite were introduced. This rock appears to have been more fragmental than most of the others but the only parts distinctly preserved are exceedingly fine-grained; the rest of the rock is cherty with veinlets of carbonate. It is thought that this was possibly an ashy shale that has been silicified in the same manner as the other rocks; but whether organic material was present in any considerable amount cannot be determined. The rock has many minute pyrite specks and is brecciated and extensively veined with carbonate. Some veinlets carry a colorless mineral, thought to be fluorite, which has not been noticed in other specimens.

Specimen E-40, Fig. 14, also, is from the oil pits and is a bluish or greenish-gray shaly, fine-textured, originally chiefly clastic rock

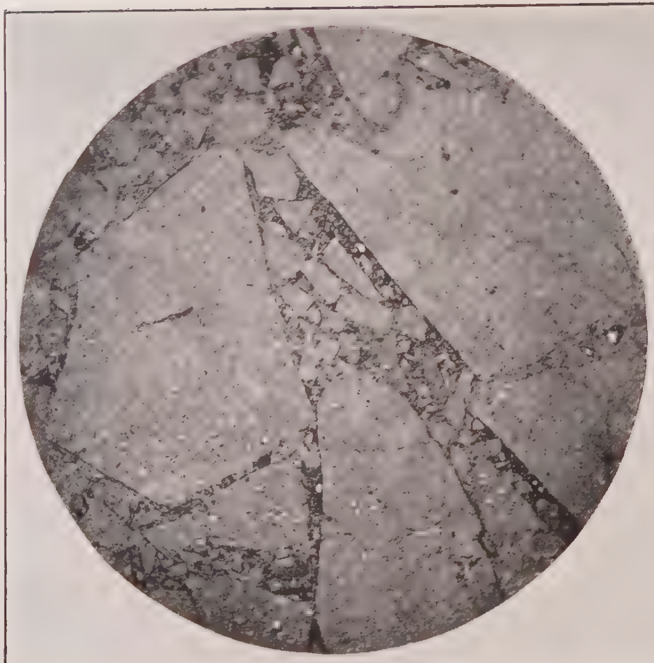


Fig. 11.—Specimen E-44-D; a brecciated rock essentially an infusorial chert; taken with plain light; × 30. Shows fine uniform texture of rock and its brecciated structure. Smooth angular patches indicate brecciation since rock became solid; original structure, judged to have been organic, is largely obliterated by chertification processes.

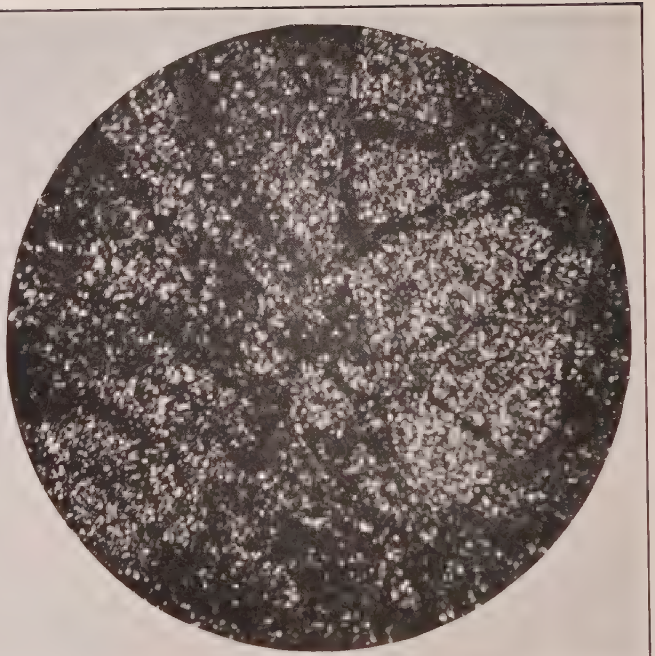


Fig. 12.—Same field as shown in Fig. 11 but taken with crossed nicols to show extremely fine texture of rock; brecciated structure is not shown as well as in plain light. × 30.

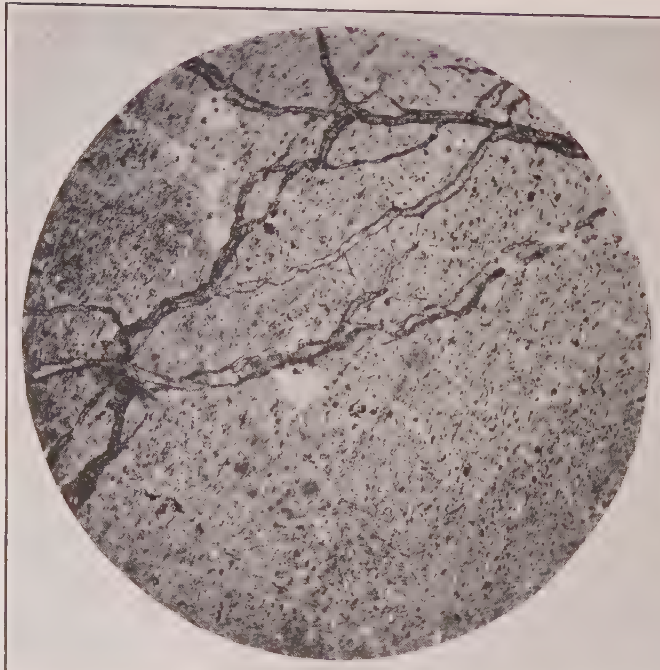


FIG. 13.—SPECIMEN E-22; SILICIFIED BRECCIATED ASH OR SHALE; TAKEN WITH PLAIN LIGHT; × 30. SHOWS FINE-GRAIN OF ROCK AND BROKEN OR VEINED STRUCTURE; THIS IS ONE OF MOST OBSCURE TYPES.

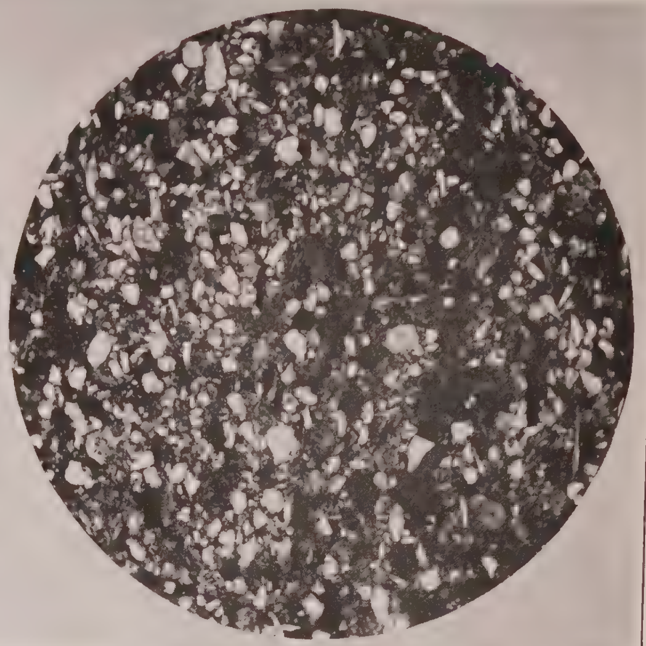


FIG. 14.—SPECIMEN E-40; FORAMINIFERAL SHALY GRIT PROBABLY CARRYING ASH; TAKEN WITH PLAIN LIGHT; × 30. SHOWS COMPLEX COMPOSITION OF ROCK INCLUDING ANGULAR MINERAL FRAGMENTS (CLEAR), CARBONATE MATERIAL (GRAY GROUND MASS), MINUTE FOSSIL FORMS, LESS IMPORTANT CONSTITUENTS CHIEFLY A SMALL AMOUNT OF LIMONITE (VERY DARK), AND A LITTLE GLAUCONITE.

with some organic remains. The primary minerals were quartz fragments, carbonates, mica plates, and fragments of fossils, especially foraminifera, glauconite and pyrite. This rock carries many mineral fragments, especially feldspar and quartz; the matrix is chiefly carbonate. It represents probably almost the extreme of clastic make-up of the series. Most of the other specimens are dominantly organic; the clastic material suggests the possibility of its derivation from an ash.

Some of the specimens shown are fine cherts in which original structures are almost wholly destroyed, whereas others well retain the original organic and clastic forms. There is no doubt that all had the same origin, being sediments composed essentially of siliceous and calcareous organic remains. In some specimens, there is much angular clastic material, which may be ash. The organic content varies widely. The tendency of the rocks is to chertify; nearly all are brecciated and veined with carbonate, which in some cases tends to replace part of the chert. Two or three are more obscure and carry finer fragmental material. The whole series represents beds of microscopic organisms, with some fragmental material largely silicified and greatly brecciated. In one case a little fluorite was noted in the veins; this is thought to indicate some igneous influence.

We have seen only microscopic fossils in these cherts. The series of slides examined by Dr. H. N. Coryell, of Columbia University, contains the following fossils: E-23, Spumellaria and Nassellaria; E-B, Spumellaria, sponge spicules, and Radiolarian spikes; E-C, Dentalina, Spumellaria, Rotalia, Orbulina universa and Textularia; E-40, Textularia cf. globifera, Orbulina universa, Globigerina and Rotalia; E-29, Spumellaria (very abundant), Radiolarian spikes, sponge spicules and Orbulina universa; E-44, Spumellaria, sponge spicules and Radiolarian spikes.

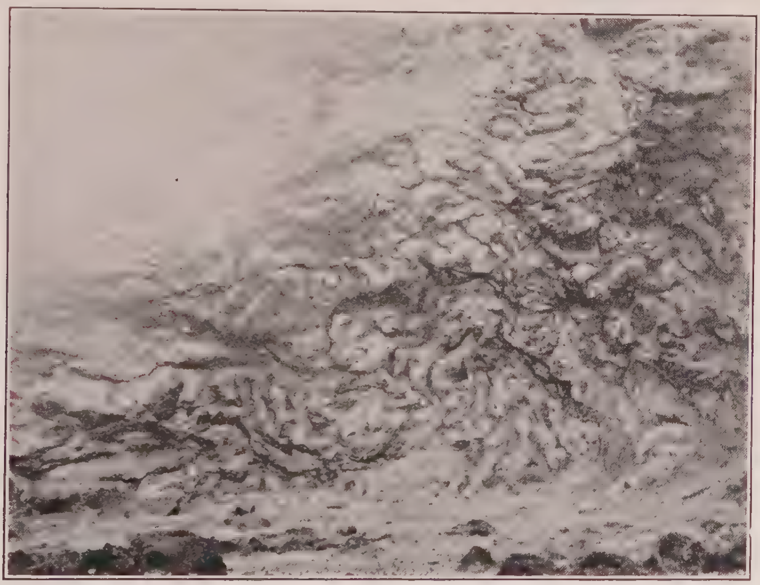
Doctor Coryell states: "The fauna consist of single to many-chambered, one-celled lime-secreting foraminifera and single-chambered silica-secreting radiolarians. Animals of this nature are pelagic, and move about near the surface of the sea in large numbers. The siliceous radiolarian remains are found in deep and shallow waters; the calcareous foraminifera and siliceous sponges are inhabitants of shallow seas. . . . The forms are consistent for Cretaceous age, some of them being equally appropriate for the Eocene. On the whole we judge that the series is Cretaceous."

Structure of the Cherts

As the cherts are everywhere crushed there is no trace of stratification (see Fig. 15) and hence we have no idea of their thickness. The single place in the entire peninsula where some stratification is evident is immediately next to the diabase sills. Through the protective influence of this tough rock a few feet of stratification is preserved. Also the fact that the igneous rocks are not associated with the Tertiary sandstones and shales tends to prove that they were intruded before the Tertiary rocks were laid down.

Oil Pits in the Cherts

The pits from which petroleum is being obtained are in cherts. As shown in Fig. 16, there are six groups of petroleum pits, each group consisting of from 20 to 40 pits, varying in depth from 10 to 72 ft. and in diameter from 5 to 10 ft. These groups, beginning on the north coast, are: The Chilean Syndicate group, 1 mile west of Salinas; the Carolina Refinery group, 4 miles east of Salinas; the Republic Refinery group, a little over 3 miles west of Santa Elena; the Santa Paula group near the south shore, 5 miles southeast of Salinas; a group halfway between the Santa Paula and the Republic Refinery groups; two pits on the north shore northeast of the lighthouse.



(Photograph loaned by Theron Wasson.)

FIG. 15.—CHERTS OF SANTA ELENA PENINSULA; NOTE CONTORTED ATTITUDE.

In all these pits the oil oozes up through the cherts and collects at the bottom where it is gathered in buckets, lifted to the surface by windlasses, and placed in barrels which are rolled by burros to the refineries.

Igneous Rocks

The areal distribution of the igneous rocks is negligible. They were found[3] only on the north shore east of the Carolina refinery, and (as reported by Theron Wasson)[4] on the railway grade between Santa Elena and Volcancitos. At the Carolina refinery, five igneous dikes cut the cherts. In one place, they appear to be infolded with the cherts as though they had been first intruded as a sill into the cherts and later folded, faulted, and crushed with them.

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[3] A geologist who recently visited this region thinks that there are igneous flow rocks capping the upturned cherts and older igneous rocks and says that the reservoir for the village of Salinas is situated on this impervious lava capping. We have not seen this occurrence.

Only one specimen of this lot is described, Fig. 17; this was taken from the dikes or sills near Cautivo (Carolina refinery). It is a dark crystalline rock of medium texture whose original structure appears to be diabasic and somewhat porphyritic. The primary essential minerals are labradorite, augite, and some altered olivine. The primary accessory minerals are magnetite and apatite. The secondary minerals are chlorite, uralite and questionable serpentine, and some limonite. The rock is comparatively fresh, a fact which in the tropics does not mean that it is of recent origin. Only the olivine is altered. The rock has a diabasic structure and porphyritic habit, which is much more prominent than in the usual diabase. We have tried to indicate this double habit by calling it a porphyritic diabase. On account of its structural relation to the cherts we believe these igneous rocks also are of Cretaceous age.

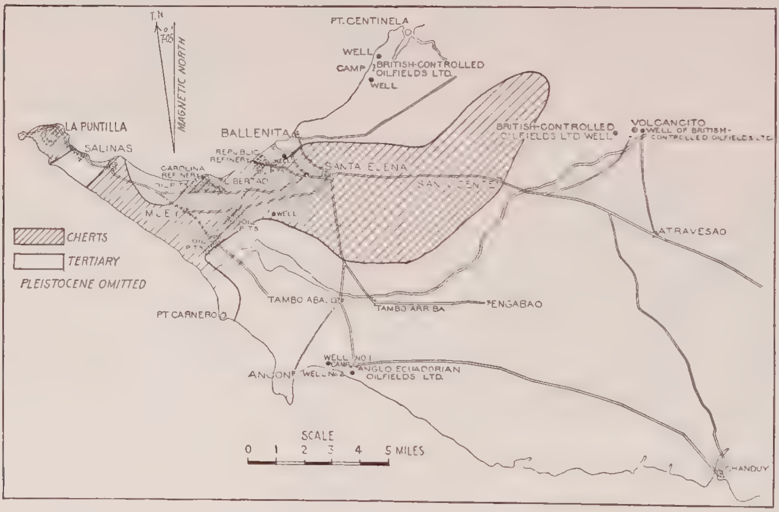

FIG. 16.—SANTA ELENA PENINSULA.

Tertiary Sandstones and Shales

Disregarding again the areal extent of the Pleistocene, about one-half of the peninsula is made up of a thick series of sandstones and sandy shales with which no igneous rocks are associated and whose age we think is Tertiary, from the fact that they seem to overlie the Cretaceous cherts and certainly are older than the Pleistocene deposits. In the examination, we did not find any fossils except in the vicinity of Centinela, on the north coast, and unfortunately these were few and so badly broken that we did not attempt to study them. These sandstones and shales

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[4] Theron Wasson, private report.

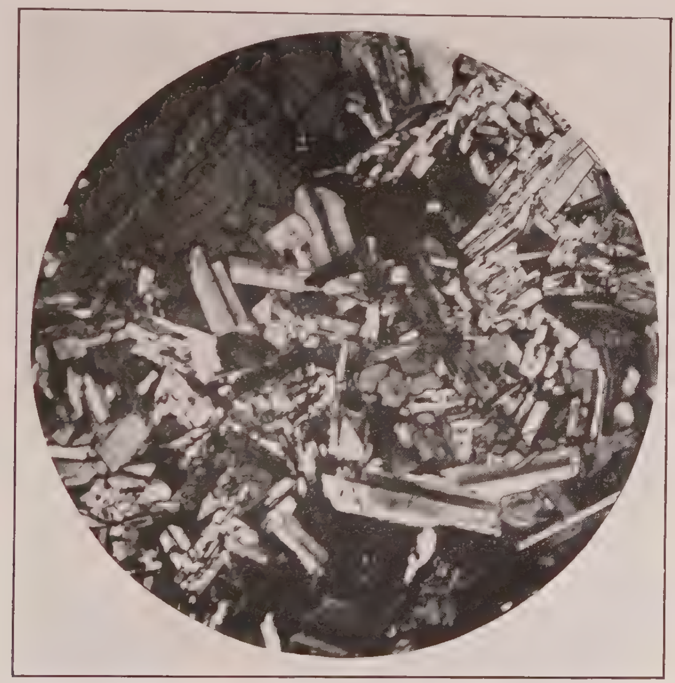


FIG. 17.—SPECIMEN E-46; A PROPHYRITIC DIABASE; TAKEN WITH CROSSED NICOLS; × 30. SHOWS DIABASE CRYSTALLINE STRUCTURE OF ROCK; ONE OF LARGER PHENOCRYSTS OF PYROXENE EXTENDS INTO FIELD FROM ONE SIDE BUT FIELD IS MADE UP MOSTLY OF WELL-FORMED LATH-SHAPED PLAGIOCLASE CRYSTALS, WITH PYROXENE AND ALTERED OLIVENE AND MAGNETITE OCCUPYING INTERSTITIAL SPACES.

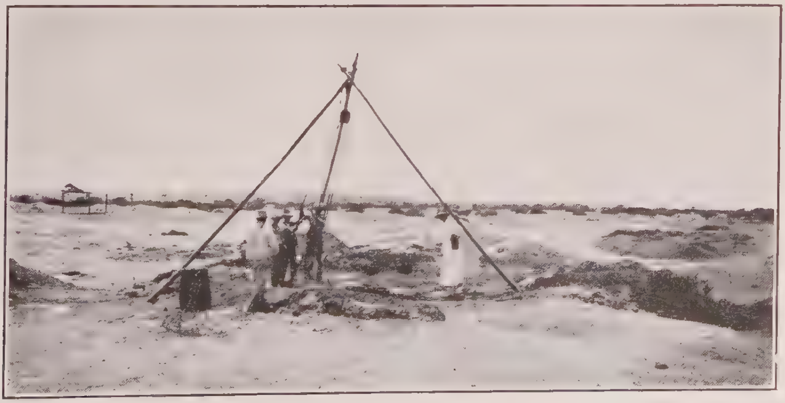


FIG. 18.—TRIPOD WITH PULLEY FOR LIFTING BUCKETS OF OIL AND WATER FROM PITS, ON THE SANTA ELENA PENINSULA.

are probably several thousand feet thick and although on the south shore near Ancon they are discolored by a small seepage of petroleum, no beds of black carbonaceous shale occur in them. In a few places we noticed, especially near Centinela, on the north coast and near Ancon, small amounts of lignite and gypsum in the conglomerate which is found here and there. They are always highly contorted and faulted in the vicinity of the cherts but away from these the strata become less disturbed and, on the south shore, lie nearly flat although in them occur faults which appear to be of considerable throw, as on the Ancon headland. In general, the determination of anticlines is difficult. Two sharp folds occur on the north shore near Centinela and a broad fold is believed to

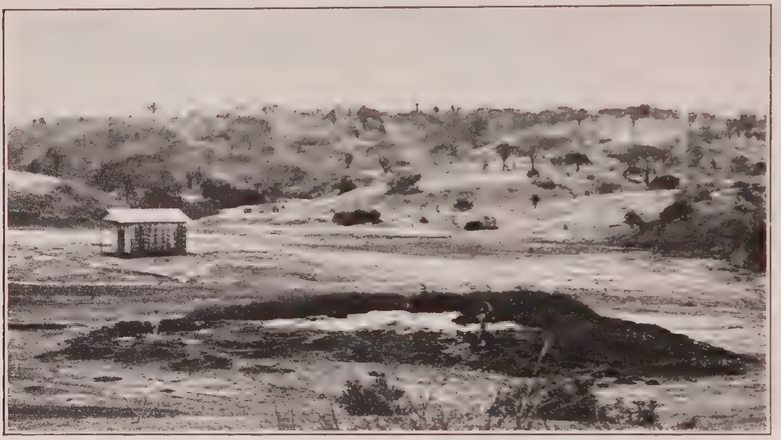


FIG. 19.—MUD VOLCANO NEAR SAN VICENTE; SHED COVERS ONE OF THE HOT MEDICINAL SPRINGS WHERE SICK BATHE.

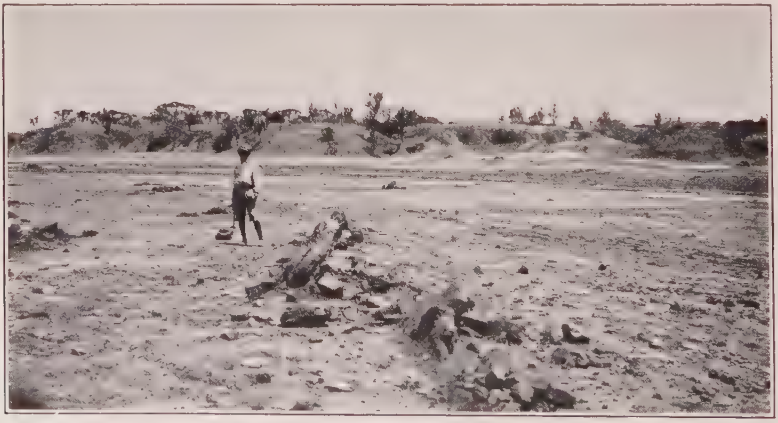
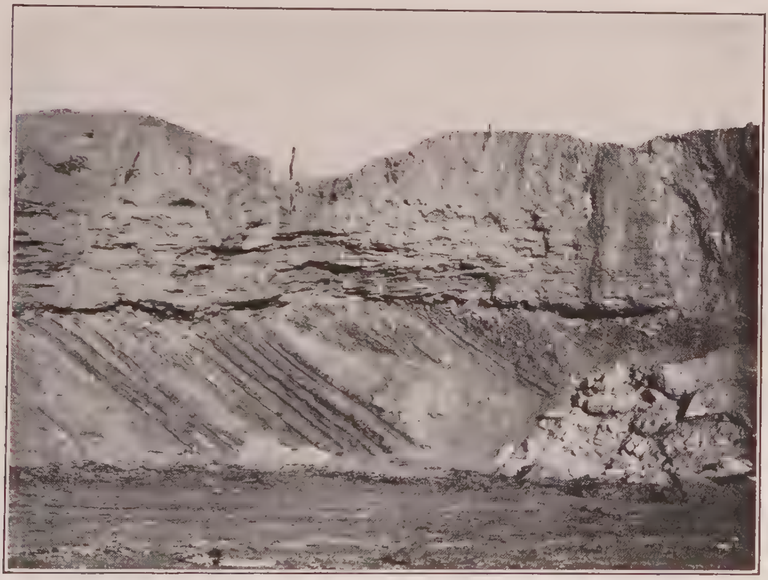


FIG. 20.—OUTCROP OF TERTIARY SANDSTONES AND VALLEY NEAR SECOND WELL OF BRITISH CONTROLLED OILFIELDS, LTD., VOLCANCITOS.

exist in the vicinity of Ancon, where the Anglo-Ecuadorian Oil Co., Ltd., is drilling. In the upturned beds of the Tertiary rocks, 11 miles northeast of Santa Elena, is a mud volcano (see Fig. 19) 10 ft. high which has a cone about 100 ft. in diameter, while the top diameter is about 20 ft. It is built up of mud and tar. Gas constantly breaks through the central orifice, often with considerable force. Nearby are thermal springs which are considered medicinal.

Pleistocene Formation (Tablazo)

This formation occurs as a thin covering over the Cretaceous and Tertiary rocks and is somewhat widespread, see Fig. 21. It is made up of marine shells and contains, in places, bones of vertebrates. Of the four classes of rocks found on the peninsula, this is generally regarded as of the most certain age. We have, however, seen no list of fossils from it. Wolf,[5] in describing the vertebrates of this formation, says that in the vicinity of Santa Elena are found large bones that have been determined to be of Mastodon Andium and Equus Andium, the same as are found on top of the Andes in the tuffs and alluvial beds near Riobamba, Ecuador.[6]



(Photograph loaned by Theron Wasson.)

FIG. 21.—PLEISTOCENE OVERLYING IN ANGULAR UNCONFORMITY UPTURNED TERTIARY SEDIMENTS ON NORTH SHORE OF PENINSULA.

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[5] Theodor Wolf: Geografia y Geologia del Ecuador, 1892.

[6] W. Branco: Ueber eine fossile Saugethier Fuana von Punin bei Riobamba in Ecuador nach dem Sammlungen von W. Reiss und A. Stubel. Geol. und Palaeont. Abhandlungen, 1, 2, 166 pages, 1883.

Secondary Enrichment at Eagle Mine, Bonanza, Colo.

BY C. ERB WUENSCH, E. M., CHICAGO, ILL.

(Canadian Meeting, August, 1923)

THE Eagle mine is situated in the Kerber Creek mining district, Bonanza, Saguache County, Colo. The climate and topography of this district are similar to those of mining camps of the Rocky Mountain region that lie 9000 to 12,000 ft. above sea level. The district is at the northeasterly edge of the San Juan volcanic region, in which successive Tertiary lavas have built up a high plateau resting upon earlier sedimentary formations, or the underlying pre-Cambrian complex. The latter was exposed in places by erosion previous to the eruptions. In the Bonanza district, proper, only volcanic formations, andesites and latites, are found. Dr. H. B. Patton,[1] in his survey of the district, discriminates between several varieties of lavas and maps these as distinct flows. They are local variations within the same lava and it is impossible to determine any distinct line of demarkation between them. Structurally they are members of the same flow.

There are three general types of vein occurrences:

1. Lead-zinc-copper-silver-gold veins in well-defined fissures with a quartz gangue; in these, the various sulfides show a zonal arrangement.
2. Massive gray copper ores high in silver, in irregular fractures in the form of small lenses and pockets, in which little or no quartz is found.
3. Manganiferous quartz veins very low in silver and gold at the surface which, with depth, become important silver veins containing subordinate amounts of gold. These veins have a quartz gangue with an abundance of rhodochrosite. The silver is associated with small amounts of sphalerite, galena, and pyrite; fluorite also is present.

The Eagle is the only mine that has been developed in the manganiferous quartz type of vein. It was discovered in the fall of 1882, but little more than assessment work was done on the property for a number of years. A prominent lens of quartz containing an abundance of manganese oxides outcropped, but its silver and gold content were practically nil. At a depth of about 40 ft. a small pocket of native and horn silver ore was found, which netted $1200 from approximately 1200 lb. This stimulated development and at a depth of 90 ft. the water level was reached without finding any more ore. Desultory operations were conducted by various lessees for several years and, on the 134-ft. level, a few small "bunches" of residual primary ore, which had escaped leaching, were found; because of their low grade (average 12.0 oz. silver and 0.02 oz. gold) operations proved unprofitable. In 1898, the owner, a local merchant, sank the shaft to a depth of 200 ft., where the top of the zone of secondary enrichment was encountered. The rich sulfide streaks were carefully sorted out and the low-grade vein material used to fill the stopes. The ore shipped averaged about 130 oz. of silver and 0.30 oz. in gold. Occasionally the gold content was as high as 2 to 3 oz. Up to the present time, the total production of the mine has been about 600,000 oz. of silver and 3000 oz. of gold from 35,000 tons of ore.

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[1] Colorado Geol. Surv. *Bull.* 9.

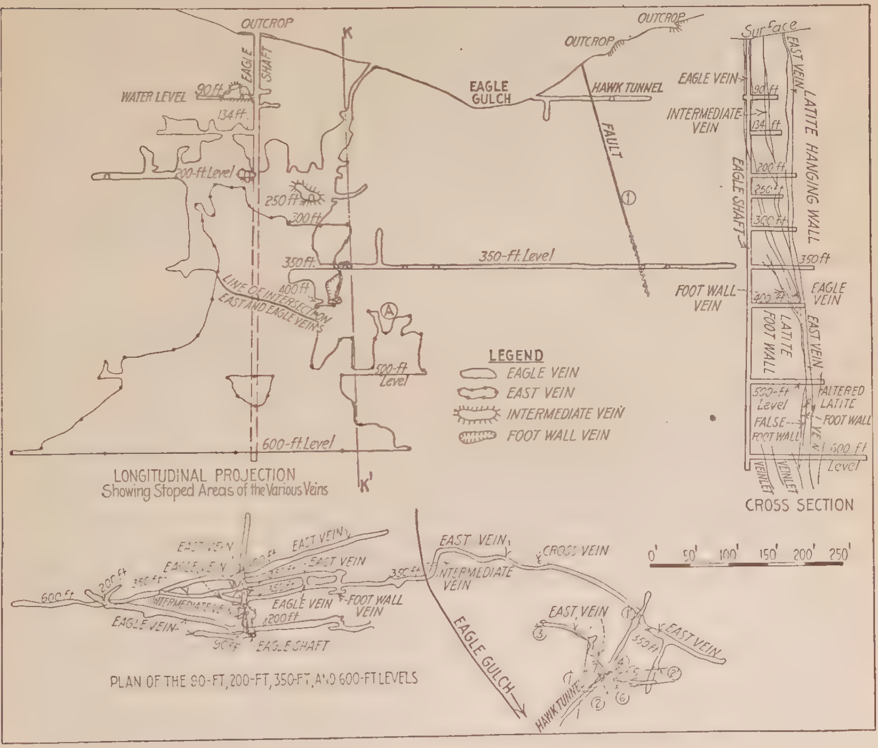

Fig. 1.—Plan and sections of Eagle Mine.

Structural Characteristics of Deposit

The Eagle mine constitutes a linked vein system of fissures in latite which have a general north-northwesterly trend. These fractures are roughly parallel and dip to the east at angles between 75° and 90°. The direction of strike is, roughly, at right angles to that of the gulches. The veins seem to have a preference for depressions, such as draws and saddles, rather than the ridges. Because of this tendency, and because of the heavy mantle of wash that fills these depressions, few outcrops are found.

There are three veins of consequence in the Eagle mine. The first, the Eagle, outcrops prominently in two places and contains an abundance of manganese oxides in a quartz gangue; small amounts of limonite and fluorite are also present. The outcrops of the other two, the Intermediate and East veins, are scarcely discernible. The outcrop of the East vein consists of a mixture of barren quartz, silicified and kaolinized porphyry with only a slight manganese stain. The Intermediate vein does not seem to outcrop; only a small manganese stained fracture is found on the surface.

The oxidized zone is very porous; the sulfide zone not so much, but both are of such an open texture as to permit rapid and deep circulation of the surface waters.

The Eagle and East veins are well defined. They vary from 2.5 to 17 ft. in width, with an average of 4 to 5 ft. The Intermediate vein is very erratic; it pinches and swells both laterally and vertically. Numerous other small feeders and fractures extend into the hanging wall and some into the foot wall. These terminate in relatively short distances and join either of the two major veins. The veins occupy well-defined fracture planes, as well as erratic fractures suggestive of shearing. The foot wall is especially well developed in all the veins, the hanging wall not so well.

Between the fifth and sixth levels, a false foot wall is found in addition to the real foot wall. This has proved most disastrous to the mining of the orebody by the shrinkage system between these levels. It was wholly unexpected because, except for two small lenses of latite found lying along the foot wall of the vein in the upper parts of the mine, the walls were very firm and well adapted to this system of mining. However, this false foot-wall material, which is from 2 to 8 ft. in width (with an average of perhaps 3 ft.) is uniformly present between these levels, and because of its highly altered and propylitic condition, it sloughed off and seriously diluted the grade of the ore. The false foot wall appears to be the downward extension of the foot wall of the Eagle vein, whereas the real foot wall is the downward extension of the East vein, see cross-section. In general, the wall rock is comparatively fresh and unaltered, except for a few feet next to the vein, where it frequently is partly silicified and pyritized.

CHARACTER OF MINERALIZATION

Roughly, the upper hundred feet of the various veins are completely oxidized, although locally oxidation may extend to between 250–300 ft. The vein material consists of quartz, psilomelane, pyrolusite, wad, limonite and fluorite. The fluorite usually is completely decolorized or has a black to purplish-black tint; a decided difference from the deep green in the primary ore. Even though quartz predominates, the vein is so thoroughly stained by the manganese oxides as to give the appearance that it is the most abundant mineral. This part of the vein is practically devoid of precious-metal content. It contains from a trace to 0.01 oz. gold, and from 0.3 to 6.0 oz. silver. The higher content is found only in small residual portions where sulfides are sparingly disseminated. In these instances, the rhodochrosite can usually be observed in the last stages of alteration into psilomelane.

Below the oxidized zone, the vein consists of quartz, rhodochrosite, and fluorite containing subordinate amounts of sphalerite, galena, and pyrite. Occasionally a few specks of chalcopyrite are found. The rhodochrosite is coarsely crystallized in beautiful pink warped rhombohedrons, and the fluorite in roughly developed octahedrons of a deep green. These are made up of a number of parallel growing smaller crystals whose real form is the cube modified by the dodecahedron.[2] The combination of pink rhodochrosite, green fluorite, and white quartz, spotted with the various sulfides, makes the ore occurrence unique as well as beautiful. In the upper part of the sulfide zone, argentite coats the sphalerite and, to a less extent, the galena. This is unusual because galena is usually the more effective precipitant of silver from secondary solutions. There is some evidence that chalcopyrite, which is visible in intimate association with the sphalerite, may account for this irregularity. The sphalerite is mostly of the resinous variety, although some marmatite is found. Most of the sphalerite, however, has the appearance of being of the black jack variety, because of the sooty coating of secondary argentite. In places, native silver is found in the form of delicate wires in vug holes perched upon the base sulfides. Doctor Patton says wire silver is the predominant ore mineral in the Eagle.[3] This is not the case, however. At the time of his visit to the district, the mine was full of water so that he probably derived his information from sources other than observation. Below this argentite zone, the ruby silver minerals, proustite and pyrargarite, are found in association with the other sulfides. The former appears to be present slightly in excess of the latter. Occasional crystals of polybasite have been found. The ruby silver minerals are invariably associated with secondary pyrite, which has a greenish tinge and is present in very minute crystals. Some of this pyrite seems to be of later origin than the ruby silver minerals. One crystal of polybasite found was completely encrusted by this finely divided secondary pyrite.

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[2] *Idem*, p. 116.

[3] *Idem*, p. 116.

In the southern part of the mine, a yellowish-white material, resembling kaolin, is found in great abundance; this is a part replacement of the rhodochrosite by nontronite, a hydrous silicate of iron. It will be described in detail later.

In vugs, in the sulfide zone, crystals of sphalerite and pyrite with subordinate amounts of galena are deposited upon rhodochrosite crystals, indicative of their secondary origin. Secondary quartz crystals are frequently found upon rhodochrosite, and in turn concentric oolitic coatings of secondary rhodochrosite are found deposited upon the quartz as well as the other vein minerals. Negative pseudomorphs of quartz after rhodochrosite are found and, very rarely, quartz pseudomorphs after rhodochrosite. Imbricating quartz blades, which have preserved the cleavage angles of the rhodochrosite rhombohedrons, are frequently found. In the lower part of the oxidized zone, occasional pseudomorphs of psilomelane or pyrolusite after rhodochrosite are found. In the upper part of the oxidized zone there are no pseudomorphs to suggest the origin of the manganese oxides.

In certain parts of the vein, brecciated and partly silicified fragments of country rock (latite) are found. In these instances the vein is invariably non-productive and rhodochrosite conspicuously lacking or present in very small quantity.

In the lower levels, rhodochrosite, of apparently secondary origin, is found so intimately associated with secondary quartz as to appear to be in the incipient stages of metamorphism into a variety of rhodonite. This material will not effervesce in any condition of hydrochloric acid.

In places on the 134-ft. and 200-ft. levels, an interesting segregation of secondary quartz from the manganese oxides is found. The material appears to be composed of barren quartz, but close inspection shows that the quartz is merely a thin coating over the black manganese oxides.

Two distinct periods of mineralization are evident. The first is characterized by an abundance of barren quartz with subordinate amounts of fluorite and containing a small amount of pyrite. Later, along the same lines of fracturing, or roughly parallel fractures, the second period of mineralization is found. This was characterized by an abundance of quartz, rhodochrosite, and subordinate amounts of fluorite; the amounts vary considerably in different parts of the vein. It was with these mineralizing solutions that the argentiferous sulfides[4] were introduced into the vein by ascending solutions. Invariably these sulfides are absent if there is no rhodochrosite, although in some parts of the vein where rhodochrosite predominates but little or no sulfides are found.

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[4] These also contained a small amount of gold.

SECONDARY ENRICHMENT

It has been stated that secondary processes have impoverished (leached) the silver and gold in the oxidized zone and enriched the portions below the water level. The present ground-water level is at a depth of 90 ft., or roughly at the level of the creek. Oxidation, however, extends on the average to a depth of 100–150 ft. The oxidized zone is almost devoid of precious metals. The longitudinal projection shows that no stoping of any consequence was done above the 134-ft. level. In the little stope at the north end of this level, a small shoot of residual unoxidized ore, averaging about 12 oz. in silver, shows rhodochrosite with rosin zinc blende and galena. The pure sulfides will average only about 75 oz. a ton in silver and 0.30 oz. in gold. The change from this material to the barren oxidized material is abrupt—within 10 to 15 ft. The sulfides are remarkably fresh and are probably characteristic of the original primary ore. Not much could be seen of the oreshoot on the 200-ft. level, because of its being quite thoroughly stoped. The small pillars left show that oxidation, except for that which has taken place superficially since the vein was opened (the walls of the drifts become superficially oxidized within two months after exposing the vein), was only local. The Intermediate vein, on this level, is completely oxidized; the East vein, only slightly; and the Eagle vein scarcely at all. As far as opened on this level, the East vein, though only partly oxidized, is very low grade. The northern part of the Eagle vein is practically barren; it consists almost entirely of the early period of quartz, with small amounts of fluorite and scarcely a trace of rhodochrosite. The oreshoot gets progressively longer with each successive level in depth.

On the 300- and 350-ft. levels, the primary sulfides are coated with a film of secondary argentite, and in places, in vugs, these base-metal sulfides occur as a drusy coating on the rhodochrosite, suggestive of a secondary origin; they are but slightly coated with secondary argentite. A small amount of ruby silver was found on this level. The 400-ft. level is not very different from the 350-ft. level. The intersection does not appear to have exerted any particular influence on the ore deposition. The following table of assays, representative of the average grade of ore in the oreshoot, shows the value of the vein material on the different levels:

LEVEL	AU	AG	MN	FE	PB	ZN	SiO_2	Al_2O_3	S	CO_2
90 ft.	0.01	2.4	9.6	3.4	0.15	tr.	62.4	...	0.4	
134 ft.	0.01	9.8[a]	4.8	2.4	0.09	0.10				
200 ft.		7.0		2.0	0.10	0.09				
300 ft.		22.7								
400 ft.		15.2								
500 ft.	0.02	15.5	7.6	3.1	0.50	0.20	46.6	1.7	1.7	
600 ft.	0.02	10.6	13.6	3.0	0.10	0.20	48.8	1.2	1.1	15.5

[a] 2.5 oz. excluding lens of residual primary ore.

These samples are quite representative of the vein material on the respective levels, with the exception of the 200-ft., 300-ft., and 400-ft., which were so largely stoped out that an average could not be obtained, although the samples given are the average of what remains. Samples from the 500-ft. and 600-ft. levels were obtained from the floor of the respective levels, samples being taken every 10 ft. If the samples of the residual lens of primary ore are omitted from the samples taken on the 134-ft. level, the average silver content will be about the same as on the 90-ft. level.

On the 500-ft. level, the vein averaged 4.5 ft. wide. On the 600-ft. level, the samples were taken over an average width of 5.25 ft., but the vein is probably nearer 7.0 ft. wide. The crosscut from the shaft was extended 100 ft. into the hanging wall. Here the vein is 17 ft. wide; only the 6 ft. next to the foot wall were included in the samples, the remaining 11 ft. consist of the early period of barren quartz, which here will average about 3 oz. in silver, probably due to scattering of downward moving secondary solutions. In the upper levels this barren quartz runs less than an ounce. Multiplying the respective widths and average values of the vein material on the 500-ft. and 600-ft. levels, a foot-ounce value of 70 is obtained for the vein on the 500-ft. level, whereas only 56 ft.-oz. are obtained for the 600-ft. level. From this, it would appear that there was an impoverishment of about 20 per cent. in the vein from the 500-ft. to the 600-ft. level. However, if 4 ft.-oz. are added to allow for the excess width of the vein, the difference between 5 ft. sampled and 7 ft. width multiplied by 2 oz., plus 3 additional ft.-oz. for the 3 ft. of propylitic latite, averaging about 1 oz., which lies between the false and real foot wall of the vein between these levels, there is an impoverishment of only 7 ft.-oz. or 10 per cent. On the 500-ft. level, a number of the samples across the vein contained from 50 to over 100 oz. in silver, whereas on the 600-ft. level none of the samples across the vein ran as high as 50 oz. The fact that the oreshoot is so much longer on the lower levels than on the upper would lead one to believe that the descending enriched solutions would not produce any great enrichment as they would be deposited over a much greater volume of vein material. Considering all the facts, it would appear that the lower limit of the secondary zone has about been reached, and that with further increase in depth the primary ore will be reached, and this, if the vein is not too wide, should prove profitable to mine.

Salient Physical and Chemical Factors Involved in the Enrichment

The porosity of the vein enclosed within rather tight and well-defined walls (except for the lowest level and the area in the vicinity of the creek where faulting has shattered the vein) in conjunction with the deep and rapid circulation of the vadose waters have been most effective in bringing about the chemical processes. The rhodochrosite gangue and the small amount of sulfide minerals have played the leading roles in the chemical aspect of the enrichment. The sulfides, upon oxidation, were converted into sulfates and the rhodochrosite into manganese oxides. The oxidation was undoubtedly stimulated by the presence of the soluble sulfates of iron and zinc. Manganese oxides, especially the manganic oxide, is an effective oxidizing agent. William H. Emmons[5] has emphasized the role of manganese in liberating nascent chlorine in acid solutions of such dilutions as are found in mineral deposits. This chlorine is effective in dissolving the gold and causing it to migrate.

In the Eagle mine, there has been, undoubtedly, a leaching of the gold from the outcrop, because it contains less than 0.01 oz. on the average, whereas in the upper part of the sulfide zone it averages 0.05 oz. and 0.02 oz. on the lower levels. During the early life of the mine, some of the shipments of silver ore contained from 0.10 to 2.0 oz. of gold. This gold is the result of secondary concentrations; most of this ore was obtained in the vicinity of the 200-ft. level. The presence of the higher manganese oxides would inhibit the reduction of ferric sulfate to ferrous sulfate in the oxidized vein material, so that precipitation of the gold would be retarded until the ferric sulfate encountered the rhodochrosite, when neutralization and reduction would effect its deposition. There has been little enrichment of gold below the 300-ft. level; the upper part of the zone of secondary enrichment.

If nascent chlorine were present to effect the solution of the gold, we would expect it to precipitate the silver as the insoluble chloride, cerargyrite. This mineral, however, was of rare occurrence. In the case of the silver, the ferric sulfate was the active solvent. It has been demonstrated[6] that ferric sulfate, in the presence of manganese oxides in acid solutions, increases the solubility of silver even in the presence of chlorides. This doubtless accounts for the leaching of the silver. The open texture of the vein, which permitted rapid circulation, coupled with the fact that rhodochrosite will not precipitate silver from dilute sulfate solutions,[7] and in addition the small amounts of base sulfides present, none of which are especially active in causing the precipitation of silver,[8] would account for the relatively great vertical extent of the zone of secondary enrichment. Because the original oreshoot increased in length with each successive level in depth, the volume of vein material leached must have been small compared to the volume of material subjected to enrichment; hence, though the enrichment was important in increasing the grade of the secondary ore, the secondary ore was not enriched as much as is usual in such enrichments.

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[5] The Agency of Manganese in the Superficial Alteration and Secondary Enrichment of Gold Deposits, *Trans.* (1912) **42**, 25.

[6] H. C. Cooke: The Secondary Enrichment of Silver Ores, *Jnl. of Geol.* (1913) **21**, 9.

[7] L. G. Ravicz: Experiments in the Enrichment of Silver, *Econ. Geol.*, **10**, 368–92.

[8] Chase Palmer and E. S. Bastin, Metallic Minerals as Precipitants of Silver and Gold, *Econ. Geol.* (1913) **8**, 140.

The various analyses of the oxidized vein material show that small amounts of lead, zinc, and sulfur are present; except the iron, they are present in smaller amounts than in the sulfide zone. This would imply a slight migration. In the presence of the abundant carbonate gangue, the small amounts of lead and zinc are probably due to their forming their relatively stable carbonates. The sulfur is probably retained with the iron as a basic sulfate, which is an intermediate mineral in the hydrolysis and conversion of ferric sulfate into limonite.

Silica (quartz) has acted normally in the leaching. Although small amounts migrated in depth, a larger amount remained in the oxidized zone. This is evident by comparing the analyses of the material from this zone with that of the lower levels and results from the relative insolubility of quartz in acid solutions of such dilutions. The resistance of fluorite to oxidation and leaching probably is because of the mild acidity of the sulfate solutions.

NONTRONITIZATION[9]

Clarke,[10] in his data ofgeochemistry, speaks of nontronite, $H_4Fe_2Si_2O_9$, as being the ferric equivalent of kaolin, $H_4Al_2Si_2O_9$. In this mine, nontronite is developed on a large scale as a replacement of the rhodochrosite. In appearance, this replacement material is easily mistaken for kaolin. It is yellowish-white in color and tends to form clay-like earthy masses. It has had a significant influence on the oreshoot, because when the nontronite material appears, the ore minerals are absent. On each level, the ore abruptly terminates as it reaches the locus of the line KK' on the longitudinal projection. Fresh, unaltered rhodochrosite changes into this nontronitic material and any sulfides present disappear. On the 500-ft. level, however, there is some rhodochrosite ore, which averages about 7 oz. in silver in the vicinity of A. In all directions, this material blends into nontronite in such a manner as to suggest that the sulfides have been leached out, and that this small lense is residual.

On the 350-ft. level, a long drift has been driven to explore the prominent outcrop on the south side of the gulch. This was considered to be more promising territory than the proved territory in the vicinity of the Eagle shaft. The old Hawk tunnel was reopened and the vein sampled systematically. The gold content varied from a trace to 0.01 oz., the silver from 0.3 to 7.7 oz., and lead and zinc from a trace to 0.3 per cent. A composite of all the samples had the following analysis: Gold 0.01 oz., silver 2.3 oz., manganese 10.8 per cent., iron 4.8 per cent., lead 0.07 per cent., zinc 0.08 per cent., insoluble 68.0 per cent., sulfur 0.5 per cent. A composite of the outcrop samples, taken from 50 to 70 ft. vertically above the tunnel level, gave the following results: Gold 0.01 oz., silver 1.0 oz., manganese 16.7 per cent., iron 4.1 per cent., lead 0.05 per cent., zinc 0.10 per cent., insoluble 65 per cent., sulfur 0.6 per cent.

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[9] The author wishes to acknowledge the assistance given by A. W. Warwick in the identification of the mineral nontronite.

[10] U. S. Geol. Surv. *Bull.* 616, p. 491.

In the Hawk tunnel, a few residual nodules of rhodochrosite showing sulfides were found. One of these contained 0.01 oz. gold and 17.1 oz. silver. This vein material was identical with that of the 90-ft. level of the Eagle in mineralogical as well as chemical composition. The wall rock was also extensively silicified. Indeed, the indications were favorable enough to justify the expectation of finding a richer oreshoot under this outcrop than was found beneath the Eagle outcrop; but no oreshoot was found on the 350-ft. level, the yellowish-white nontronitic replacement material was universally present in various stages of the replacement. In a few places, small bunches of sooty sulfides were found in both the nontronite and the quartz. Picked specimens of this material contained from 2 to 5 oz. in gold and from 200 to 500 oz. in silver. Out of about 500 ft. of drifting on the vein only three or four daily face samples contained more than 2.0 oz. of silver and a trace in gold, although the gold content of these was much higher on all samples than from the same grade of silver from other parts of the mine. The average of the whole vein would not contain as much as did the average on the Hawk tunnel level, 220 ft. vertically above.

Where the cross vein joins the East vein, rhodochrosite containing a few disseminated crystals of the sulfide minerals, characteristic of the deposit, is found. The rhodochrosite shows various stages of the conversion into nontronite. The sulfide particles are rather sooty and give the appearance that they, too, are in the incipient stages of solution. This seems to indicate that a selective solution may have taken place, that is, that the character of the downward circulating waters was such as to have effected the solution of the sulfides before the rhodochrosite. If this is so, there is a possibility of finding an important secondary enrichment in depth. On the other hand, there is the contradictory evidence that no important amounts of sulfides were ever present. In a few isolated parts of the veins, small areas of rather fresh rhodochrosite, with an abundance of quartz and some fluorite are found in which little or no nontronite is present, but the sulfides are absent. This is, however, only a local condition, because within short distances ore material is again found. It seems incredible that this nontronitized portion of the vein should never have contained sulfide minerals, because of the clearly established genetic relationship between the rhodochrosite and the silver-bearing solutions. Then, too, the strong character of the mineralization, as evinced by the abundance of rhodochrosite, quartz and fluorite, discredits any belief that the locus of the line *KK′* is the original (previous to nontronitization) southern limit of the oreshoot. The northern limit of the oreshoot is clearly defined by a gradual transition from the characteristic mineralization into a partly silicified wall rock or barren quartz.

An analysis of the pure nontronitic material, which apparently was representative of the most advanced stage of the replacement (it was identical in appearance to kaolin, not a vestige of its having any relation to the rhodochrosite could be detected by a 20× hand lens), gave the following results:

SiO_2	24.2	
Al_2O_3	0.8	
CO_2	23.5	
Fe_2O_3	12.9	
MnO	31.5	
Zn	0.2	
Pb	tr.	
CaO	5.3	
S	none	
Total	98.4	
H_2O	2.8	Calculated necessary to form nontronite.
	101.2[a]	

Assuming that the manganese and the calcium are both present as carbonates, this would require the presence of 23.6 per cent. CO_2. This checks the total amount of CO_2 present within 0.1 per cent. Pure nontronite, $H_4Fe_2Si_2O_9$, contains 51 per cent. Fe_2O_3, 38 per cent. SiO_2 and 11 per cent. H_2O. This material only contains one-fourth this amount of ferric oxide, which would only require 9.5 per cent. silica to form nontronite. There is, therefore, an excess of silica amounting to 14.7 per cent. present, not taking into consideration the 0.8 per cent. alumina. This uncombined silica must be present in a gelatinous form, which is soluble in hot concentrated or dilute hydrochloric acid, because when pure specimens of this replacement material, free from visible quartz, are dissolved in acid no appreciable amount of gritty material, indicative of the mineral quartz, is present. The whole specimen, except the fraction of a per cent. which might be associated with the alumina, dissolves readily with the formation of an abundance of gelatinous silica. It would appear from this that some form of hydrolysis, in which a hydrous metallic silicate was present, took place. The fact that the replacement is not complete, complicates the problem. Nontronite is easily decomposed by either acid or alkaline solutions leaving limonite as one of the products. But there is a deficiency of ferric oxide, so that if it was formed, because of its stability, we would expect it to remain behind and not be leached, especially as nontronitization is incomplete. We would also expect the rhodochrosite to be further replaced before the nontronite undergoes decomposition.

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[a] These analyses were made by a commercial assayer and hence are hardly accurate enough to check exactly; all computations were made on a slide rule.

Patton[11] gives the following analysis of the Eagle latite:

SiO_2	67.16	Na_2O	4.23	P_2O_5	0.05
Al_2O_3	16.03	K_2O	5.78	SO_3	0.03
Fe_2O_3	1.67	H_2O	0.64	Cl	none
FeO	1.38	TiO_2	0.56	S	none
MgO	0.77	ZrO_2	none	MnO	0.12
CaO	1.62	CO_2	0.06		
				Total	100.21

The K_2O and Na_2O contents are extremely high. Could it not be possible that the unusual amounts of alkali silicates were effective in bringing about the nontronitization and that they themselves may have been hydrolyzed and deposited at least a portion of this excess gelatinous silica? It is known[12] that solutions of alkali silicates precipitate solutions of all metallic salts with the formation of insoluble silicates, which are decomposed by acids with the separation of silicic acid, a gelatinous precipitate, soluble in hydrochloric acid. This process might also have been effective in the formation of the excess silica. This assumption necessitates an explanation of the manner in which the solution of the sulfide ore minerals, of the original vein, was brought about; this is beyond the scope of this paper.

An important enrichment can reasonably be expected in depth, notwithstanding certain indications to the contrary. It is hard to imagine the chemical reactions involved in bringing about this leaching of the sulfide mineral content. Where the rhodochrosite is visible and the nontronite inconspicuously developed is it not possible that the sulfide particles could have been dissolved before nontronitization manifested itself? Perplexing and as impossible as the chemistry of this may appear, I am inclined to give the structural and mineralogical evidence preference over the chemical aspect.

If this leaching has taken place, it is unusual in that it illustrates a unique example of solution and migration beneath the water level. This seems preposterous in the light of our present knowledge of the enrichment of ore deposits, but when the results of such a phenomenon are so pronounced, we can hardly set aside this evidence in favor of what appears to be an impossible cause, especially when our knowledge of the cause, geochemical activity, is so meager and where we have so frequently encountered incongruities of chemical processes in the face of observable facts.

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[11] *Op.cit.*

[12] Prescott and Johnson: "Qualitative Chemical Analysis," 291.

Unusual Example of Simultaneous Faulting and Mineralization

In the Hawk tunnel, on the south side of Eagle Gulch, an interesting example of simultaneous faulting and mineralization is found. The fractures 5 and 6 are small veinlets, 6 to 12 in. in width, containing manganese oxides, quartz, limonite, and fluorite, characteristic of the mineralization of the leached oxidized zone. These are faulted by similar fractures 4 and 7, which also contain small amounts of the same minerals. Veinlets 5 and 6 end abruptly against faults 4 and 7 and, in addition, a small faulted segment is found between the latter. Fault 2 cuts faults 4 and 7 and is itself faulted by a post-mineral fault 1. All these faults, except 1, contain the mineralization common to all the veins. This fault has displaced fault 2 from its lateral extension 3. These two fractures form the foot wall of the East vein. A perplexing feature is the fact that faults 4, 5, and 7 do not cross the East vein, but are terminated by that portion of the East vein which deviates from faults 2 and 3, which form its foot wall, at the extreme southerly end of the drift and to the north of fault 1, respectively. In this vicinity, the East vein has no well-defined walls. Fault 6 forms a false foot wall to the East vein for a short distance before it joins into 2 to the south.

On the 350-ft. level, fault 1, shown as 1, has no well-defined wall as was the case on the Hawk tunnel level, but is rather a crushed and brecciated zone. This illustrates how a well-defined fault plane may give way to a brecciated zone. The horizontal displacement is considerably less on this level than on the Hawk tunnel level, indicating a slight rotation and also suggesting the origin of the several branches of the vein system, which join into one main vein at depth.

In sympathy with this faulting, the area beneath the gulch is much more crushed than the area in the region of the shaft. It apparently offered a much freer and deeper line of circulation for the percolating vadose waters. This may account for the abundance of the yellowish-white nontronite replacement material in this area.

Possibilities in Depth with Change of Formation

On the 600-ft. level of the Eagle, which in point of elevation is the lowest mine working in the district, the normal flow of water is only between 30 and 40 gal. a min. and is obtained almost entirely from the vein itself. Judging from the general geological structure of the district, it seems logical to suppose that with a relatively short increase in depth, within the next 300 to 500 ft., the underlying sedimentaries or, if they have been removed by erosion, the basal complex should be reached. If limestone should be found, there is a possibility of large replacement orebodies occurring along the contact of the overlying porphyry, as at Leadville, or in fault fissures in the limestone, near the granite contact, as are found at Monarch and Garfield, 20 miles to the northwest. This is admittedly a "long shot" of a development chance, but one worthy of note. From the small amount of water encountered, it would appear that the veins and faults extend throughout the volcanic series, so that there may have been a relatively rapid circulation through the veins to some deep-seated artesian channels related to the sedimentaries. If the sedimentaries are absent and granite is found directly beneath the latite, the veins may continue in the granite but they will undoubtedly be nonproductive, as has been the case elsewhere in Colorado.

Helium, a National Asset

By Richard B. Moore,* D. Sc., New York, N. Y.

(Canadian Meeting, August, 1923)

The successful commercial production of helium during the last few years has added greatly to its scientific interest. When the quantity of an element available for experimental purposes increases within a few years by 200,000 per cent., both public and scientific interest in the element are greatly stimulated.

Before the war, probably, there was not more than 10 or 15 cu. ft. of helium in existence. The larger part of this was owned by Prof. Kamerlingh Onnes, and was used by him in connection with his low-temperature work at Leiden. By experimental work during the war and immediately afterwards, the Canadians, under Prof. J. C. McLennan, produced about 60,000 cu. ft. of helium of various grades of purity. Up to Dec. 1, 1921, when the plant at Fort Worth was shut down because of lack of funds, there had been produced during the experimental work by the U. S. Government, during the war and afterwards, 2,300,000 cu. ft. of helium averaging between 92 and 95 per cent. purity.

After the new appropriation was made in 1922, the production plant was again started and at the time of writing was making about 15,000 cu. ft. of helium a day. Only three of the five units were being operated because of the inability to get sufficient gas to run at full capacity. The gas supply is owned by two companies and on account of a disagreement between them, it has been possible for some time to get gas from only one. It is expected that when all five units are running steadily, the daily production will be between 30,000 and 40,000 cu. ft. of gas averaging from 93 to 95 per cent. purity.

All three experimental plants used during the war have been shut down. The Air Reduction plant, using the Claude system, and the Linde plant were both dismantled before the production plant was built. The Jefferies-Norton plant at Petrolia, Texas, was shut down permanently in July, 1921, and is being dismantled. As the Linde process was most successful during the experimental period, this process is used in the present plant. This plant was constructed by the Bureau of Yards and Docks of the Navy, and has been operated under the immediate supervision of the Bureau of Steam Engineering of the Navy. The Linde Co. has furnished the technical force and has coöperated with the Navy in the running of the plant.

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* Manager, Development Department, the Dorr Co., late chief chemist, U. S. Bureau of Mines.

It was early recognized that present commercial methods of liquefaction were not sufficiently economic in character to be permanently satisfactory for the production of helium, as in the extraction of oxygen from the air. The actual cost of such extraction of oxygen is only a small part of the total cost at which the gas is sold. This is because of the large overhead, large number of cylinders used, and the periodic running of the plants. Any process that would cut the actual cost of production in half would have little effect on the final cost of the gas. In helium plants run continuously, the situation would be different, the actual cost of operation being the main item and any reduction in such operation figures largely shows up in the final cost of the gas. Therefore, it has been extremely important to do everything possible to get more efficient methods of refrigeration and to apply these to the production of helium.

With this object in mind, research work has been carried out on fundamental problems for the Bureau of Mines at Harvard University, at Massachusetts Institute of Technology, and at the Bureau of Standards. In order to coördinate all of this work and get it under one organization, the cryogenic laboratory of the Bureau of Mines was started about two and a half years ago; this laboratory now has about fifteen research men and a competent force of mechanics and draftsmen. Fundamental data are being obtained and as they are obtained they are being translated into practice through a consulting board of engineers, which has been designing a refrigeration cycle especially adapted to the extraction of helium from natural gas. A small unit built and operated in the cryogenic laboratory has produced helium of 96 per cent. purity at one operation. The production plant at Fort Worth requires two operations to get this purity, the first step producing gas between 60 and 70 per cent., which is reprocessed, and stepped up to 92 to 94 per cent.

The next step is the construction of a semi-commercial unit at Fort Worth, where an adequate supply of natural gas is available. This unit will be about twenty-five times the size of the present one, and the final commercial unit will be ten times the size of the semi-commercial one. Three, four, or five large units will consitute a full-sized plant.

In addition to the work on the actual production of helium, the Bureau of Mines has been constructing two repurification units for the Army. One of these is at Langley Field, Virginia; this plant uses a combination of high pressure and refrigeration. In general, the gas contaminated through use in a balloon or dirigible is led through a liquefier surrounded by liquid air and, under the pressure on the gas and at the temperature obtained, the nitrogen and oxygen are liquefied and the helium passes on and is recovered in the original purity of 93 to 95 per cent.

The other plant is on two railroad cars and can be transported to wherever it is to be used. It is smaller than the one at Langley Field, having a capacity of only 1000 cu. ft. of helium per hour. However, it is unique in that it uses charcoal at low temperatures for the actual purification. Charcoal has the property of adsorbing heavy gases, but at low pressure does not adsorb helium; therefore, if a mixture of helium-oxygen-nitrogen is passed through a tube containing charcoal at relatively low temperatures, the oxygen and nitrogen are adsorbed by the charcoal and the helium passes on. By having several adsorbers available, while one is being used two or three can be cooling down, and another one can be outgassed or heated so that the adsorbed oxygen and nitrogen are given off. In this way, a continuous process is possible. The great advantage of the charcoal method of purification is that it produces a much higher grade product. A gas of 99.9 per cent. purity should be obtained continuously. This has a great advantage over the lower grade product obtained by the other method.

The gas in a dirigible, whether it be hydrogen or helium, is slowly reduced in lifting power by the diffusion of air into the bag and of helium into the air. When the grade of gas drops to about 84 or 85 per cent., the lifting power is so reduced that it is necessary to replace the old gas with new. It can be readily seen that if the original gas put into the bag is practically 100 per cent. pure instead of 94 or 95 per cent., the time the dirigible can remain in the air is greatly increased, and the range of operations largely extended. Inasmuch as it will not be necessary to purify the gas as often under such conditions, the actual cost of the use of helium in lighter-than-air craft is considerably reduced.

A third method of repurification, now being tried in the cryogenic laboratory, involves the actual expansion of the impure helium through an expansion engine. The necessary refrigeration for the liquefaction of the nitrogen and oxygen in the helium is thus obtained by the expansion of the impure gas.

The Navy is planning to build a repurification plant at Lakehurst, N. J. A dirigible with a capacity of over 2,000,000 cu. ft. of gas is being constructed in the hangar there, so that it is advisable to have a repurification plant on the ground. The Bureau of Mines has designed and is building this plant for the Navy.

The general plan, therefore, calls for the construction and operation of three repurification plants (two stationary and one mobile); the operation, for a time at least, of the production plant at Fort Worth, Texas, using the Linde process; the construction and thorough testing of a semi-commercial unit for extracting helium from natural gas, using a method developed by the Bureau of Mines and its consulting engineers, and, if this is successful, the erection of one or more full-size units after the method has been approved; the continuation of research both in pure science and the application of results obtained to the engineering projects, with the object of continuous improvements tending toward efficiency and reduction in cost. Such a program will ultimately put the helium problem where it belongs, namely as a great asset to this nation both in time of war and in time of peace. Already a strong organization is interested in the commercial construction and operation of dirigibles in this country. Its program is conservative but progressive; and this country should have in operation at least one line of commercial dirigibles within a few years. From this beginning it should be possible to build an organization that will connect not only the principal cities in this country, but the United States with Europe and, possibly, South America and the Far East.

As this country has the only large sources of natural gas containing helium in the world, it is of great importance, from the national defense standpoint, that our monopoly be retained; therefore, the exportation of helium is inadvisable. The matter should be handled without great difficulty as there are no commercial plants in operation at the present time and, therefore, the control of helium does not interfere with capital already invested. A provision of this character also will tend to develop commercial airships in this country rather than abroad. To take care of this phase, and to give the Helium Board, which controls the Government's activities in helium, an opportunity to run the project on the most efficient possible lines, a bill was introduced into Congress at the last session by Representative Kahn, of California. It was H. R. 11549, authorizing the conservation, production, and exploitation of helium gas, a mineral resource pertaining to the national defense, and to the development of commercial aeronautics, and for other purposes. The bill gave in complete form the authority necessary for government officials to carry out the helium program in the most efficient manner. The Interior Department, the Army, and the Navy have the authority outlined in several of the sections, but additional authority was called for under the bill, such as the prohibition of exportation of helium without permission of the President; the sale of byproducts, such as natural gas, etc., produced in connection with the extraction of helium; the renting of helium to commercial companies when the Army and Navy have more than they need for their operations during any given year. The bill had the backing of the President, and the Secretaries of War, Navy, and Interior. It also called for an appropriation that would allow the Government to control the resources of helium-bearing gas in some one field in the United States, in a manner somewhat similar to the Government oil reserves. In this way, the Army and the Navy would be guaranteed a supply of helium for the next 20 or 25 years, and, therefore, be independent of possible wastage of this valuable national asset.

The bill was referred to the Committee on Public Lands, which, after public hearings had been held, decided that the bill was outside its jurisdiction and asked Congress to refer the bill to the Military Affairs Committee. This was done, but as it was too late for this Committee to hold hearings, action on the bill was postponed until the next session of Congress.

Natural Sources of Helium

Helium exists in the atmosphere in a proportion of 1 part in 185,000 by volume. It is found in very minute quantities in sea and river water, and in a large number of older rocks and minerals. It undoubtedly exists in some of the fixed stars and in many nebulae, as well as in the sun. Until its presence in natural gas was noticed, the principal source of helium was from minerals containing uranium and thorium. The percentage of helium in any mineral depends on the amount of uranium or thorium in that mineral, its age, and the percentage of the helium formed that is retained in the mineral. Naturally soft pulverulent minerals, like carnotite, give up the helium more readily than hard dense minerals like monazite or thorianite. As much as 6 or 7 cc. of helium may be obtained from 1 gm. of the latter mineral by solution in acid. If a mineral is soluble in acid, the helium can readily be obtained by this method; if it is not soluble in acid, the gas may be liberated by fusion with a mixture of sodium and potassium carbonates. A certain proportion of helium in a mineral can also be obtained by heating the mineral direct without any chemical treatment. It requires, however, a high temperature and a long period of heating. On heating monazite to 500° C. only a small percentage of the helium is evolved; at 900°, 24 hr. heating is required to liberate all of the gas. For thorianite, 7 per cent. is obtained at 300° C. after 5 hr. heating; 8.5 per cent. at 500° after 8 hr.; and the whole at 1000° after 30 hr.

Radioactive minerals constituted the principal source of helium for scientific work until its commercial production. Helium is also found in the gases evolved from a large number of mineral springs. Moureu and Lepape have done considerable work in determining the presence of the rare gases in the gases evolved from the mineral springs in France. Some of these gases contain from 5 to 8 per cent. helium, but as the total volume given off during the year is small, this source has no commercial value; it has been easier to get helium from minerals for scientific purposes than from gases from springs The total volume of helium given by some of the springs in question are of interest. The largest is that of Neris (Allier), which gives 1200 cu. ft. of helium per annum; the Bourbon-Lancy (Saone et Loire) gives about 350 cu. ft.; and Santenay (Cote d'Or) 180 cu. ft. The boracic-acid fumaroles of Lardarello, Tuscany, Italy, give off steam, which is used for generating power. There is also present carbon dioxide with traces of hydrogen, methane, hydrogen sulfide, nitrogen, and a minute quantity of helium. After the steam is condensed, the gas consists mainly of carbon dioxide and contains from 0.1 to 0.15 per cent. helium.

In 1907, Cady and McFarland found that certain natural gases, particularly from southeastern Kansas, had a low British thermal unit and that they were high in nitrogen. An examination of the nitrogen showed some of the gases contained as much as 1 to 1 ½ per cent. helium.

During the war, it became advisable to know just what our sources of helium really were; therefore, the Bureau of Mines started a field survey. G. Sherburne Rogers was delegated, by the Director of the U. S. Geological Survey, to assist in this work, and the report was afterward published as Professional Paper No. 121 of the Geological Survey. A large number of samples were taken in different fields and analyzed by chemists of the Bureau of Mines, and the results showed that this country had larger potential supplies of helium in natural gases than one would have thought from the work of Cady and McFarland.

After the death of Mr. Rogers, the writer supervised a more detailed survey of the field work, and every natural-gas field in the United States has been sampled and tested for helium. It was necessary to keep a force in the field for nearly two years and a laboratory was devoted exclusively to helium analyses. The result of this work has shown that considerably more than 500,000,000 cu. ft. of helium is annually going to waste in connection with the ordinary use of natural gas. Of course, the larger part of this helium could not be extracted for commercial purposes; there is a wide variation in percentage of helium in natural gas and whereas it would be practical to extract the helium in some cases, it would not be practicable in others. However, it would be quite possible to extract yearly 50,000,000 cu. ft. of helium from gas which is being regularly used, if sufficient plants were available. This amount could be considerably increased, if necessary, by forcing certain fields. There is no question that if the United States had a protracted war, so that there would be time to build the necessary plants, 150 or 200 large dirigibles could be kept in the air over a period of four or five years and perhaps longer. This would give this country a unique weapon in case of trouble with a foreign nation and one that could not be duplicated by any other nation, because of the lack of helium within their borders.

At the present time, outside of the United States, there are only two known localities where helium could be produced in commercial quantities. Canada has two sources of natural gas containing 0.34 and 0.33 per cent. helium, one in Ontario and one in Alberta; the boracic-acid fumaroles in Italy would probably give a small commercial supply. During the war, the British, through Prof. J. C. McLennan, of the University of Toronto, used the Alberta gas as a source of helium for dirigibles. At the end of the war, however, the plant was shut down after producing about 60,000 cu. ft. of helium of various grades.

The natural-gas resources of the United States cover about a dozen states, beginning with Texas. The general trend of helium-bearing gas runs through Oklahoma, particularly the Osage, southeast Kansas, southeast Illinois, Indiana, Ohio, Pennsylvania, and New York; the belt also embraces West Virginia, Kentucky, and Arkansas. The richest gas is found in Texas, Oklahoma, and Kansas. In general, however, the Kansas fields, although high in helium, are old and the supply of gas is limited. Some of the largest gas fields in the United States are low in helium, notably those of Louisiana and Wyoming. Helium appears to be always associated with nitrogen in a natural gas, but high nitrogen does not always mean high helium. If the gas contains from 10 to 30 per cent. nitrogen, it is reasonably certain to be high in helium, but if it runs 70 or 80 per cent. of nitrogen, or higher, the chances are that it contains no helium. An ordinary analysis of a natural gas will, therefore, give at least strong indications as to whether helium is or is not present.

The volume of helium actually given by a gas field is interesting. The Petrolia Field, near Witchita Falls, Texas, from which the government has been obtaining its helium is a small but rather old field. It has an area of about 10 sq. mi. with a possible drainage area of 40 or 50 sq. mi. The natural gas produced from this field has contained about 80,000,000 cu. ft. of helium, which would supply the initial gas for twenty large dirigibles and keep them in the air for several years. As this field is only one of numerous locations favorable for the commercial production of helium, the use of helium in balloons and dirigibles is quite practical in so far as the supplies of the raw product are concerned.

Mr. Rogers, in Professional Paper 121, of the U. S. Geological Survey, has described a number of helium-bearing gas fields. A much larger amount of field work has been done since that time, but the results of this work cannot be published in detail as the Helium Board desires that a definite government policy as regards helium be adopted before the analyses are given out. Government analyses, as well as reliable analyses by others, show a helium content of from a mere trace up to 1.94 per cent.

One analysis, which has not been checked, showed one gas to contain a little over 2 per cent.

Gas is considered commercial in grade if it contains over 0.5 per cent. helium, with a sufficient volume to justify commercial operation. Table 1 gives data concerning twenty-three states containing natural gas, with the number of counties sampled in each state, and the maximum helium analysis. The figures under the column headed "average helium content," of course, give only a rough estimate of the relative importance of the fields as two or three high analyses would be greatly reduced by a number of low ones. The names of the states are not given and it is regretted that further details cannot be published at this time.

TABLE 1

Number of Counties Sampled	Highest Helium Content, Per Cent.	Lowest Helium Content, Per Cent.	Average Helium Content, Per Cent.
3	0.26	trace	0.13
7	0.11	none	none
5	1.04	none	0.20
2	0.12	0.04	0.08
9	0.61	0.11	0.34
17	2.13	trace	0.76
15	0.43	none	0.15
9	0.19	none	0.04
2	0.04	none	0.02
4	0.27	0.01	0.08
1	trace	trace	
13	0.31	none	0.11
1	0.17	0.17	0.17
15	0.55	none	0.30
24	1.04	trace	0.23
10	0.17	none	0.07
2	0.04	trace	0.03
1	0.36	0.35	0.35
26	1.73	none	0.43
2	trace	none	
3	0.12	none	0.05
2	0.06	trace	0.03
10	0.10	trace	0.01

Out of 183 localities tested in the early stages of our work, from Texas, Oklahoma, and Kansas, seven samples contained over 0.8 per cent. helium, 9 contained 0.5 to 0.8 per cent., 59 contained 0.2 to 0.5 per cent., and the remaining 108 localities showed less than 0.2 per cent. Natural gases containing over 0.5 per cent. helium are not very common, although the number of samples between 0.2 to 0.5 per cent. is rather large. This latter statement applies to a number of the other states, a large percentage of the samples analyzed carrying from 0.1 to 0.3 per cent. helium with an occasional sample around 0.4 or 0.5 per cent. or higher. More recent field work has raised considerably the number of samples in Texas, Oklahoma, and Kansas carrying more than 0.5 per cent. helium.

By a proper distribution of five or six plants, it would be quite possible to extract 50,000,000 or 60,000,000 cu. ft. of helium annually in the United States. In case of an emergency, this amount could be raised to 100,000,000 cu. ft. or more. On the other hand, natural gas cannot be used and conserved at the same time. Whereas we have available large potential volumes of helium for war or commercial purposes, natural gas is like oil and in the end the supplies will give out. A conservative but progressive policy will give to this country an adequate supply of helium for many years, but it will be necessary to use reasonable conservation measures to prevent unnecessary exploitation of gas high in helium. Where it has been possible to sample different sands in the same locality, it is usually the gas from the more shallow sands that is richer in helium. In such cases, instead of mixing the gas from all of the sands, the shallower supply should be reserved mainly for the helium content; the gas from the deeper sands could be used first for commercial purposes that do not involve the extraction of helium.

Origin of Helium

The presence of helium in such large quantities in natural gas is of great scientific interest. Helium is associated in the air with argon, neon, krypton, and xenon. It was stated by Rutherford, during his early work on radioactivity, that the alpha particle given off in radioactive changes was probably a charged helium atom. Later, Ramsay and Soddy, by actual experimental work, showed that this was true, and that helium was formed during radioactive changes involving alpha rays. From that time, it was generally accepted that the helium in the atmosphere came from the disintegration of the radioactive elements. This theory received the necessary support from the fact that the rocks constituting the earth's crust all contained minute traces of radium. In fact, Strutt has shown that taking the average radium content of the rock-forming minerals on the surface of the earth, it would only require the same average content to a depth of 40 miles to take care of the loss of heat of the earth by radiation. The radioactive explanation of the origin of helium would also give a satisfactory explanation for the presence of helium in certain springs and mineral waters, in so far as the quantity of helium alone is concerned. The richness of some of these gases is difficult to explain, as a few of them carry as much as 8 or 10 per cent. helium in the evolved gas. The large quantity of helium in natural gas, however, throws some doubt on a radioactive origin. A ton of uranium element, with all of its disintegration products including radium, will only give about 110 cubic millimeters of helium per annum. If the Petrolia Field has up to the present time given 80,000,000 cu. ft. of helium, it is easy to calculate the amount of uranium necessary for the origin of such helium, and the time required for its accumulation. There are no indications of any kind to suggest that the necessary quantities of uranium and thorium minerals are present in the earth's crust at even one locality where rich helium-bearing natural gas is found. If this statement is true, and a radioactive origin cannot be proved in connection with any one locality, what can be said concerning the helium resources in a dozen different states? If we try to explain the presence of the helium by the radioactivity of ordinary rocks, and without the aid of local concentrations of uranium and thorium minerals, it would be necessary to assume that all natural gas is strongly radioactive. The average rocks and minerals in Wyoming are just as active as the average rocks and minerals in Texas, and yet there is practically no helium in the Wyoming gas.

In addition, so far as we have been able to find out, the natural gas in Russia and Mexico contains no helium; therefore, it is not possible to assume that the presence of the rare gas is due to the radioactivity of ordinary rocks and minerals, and if we do accept a radioactive origin, it must be in connection with large and localized deposits of thorium and uranium minerals. This is an extremely difficult view to accept.

Another possible origin of helium involves acceptance of the nebular hypothesis. The chromosphere surrounds the sun to a depth of about 10,000 miles and consists of incandescent hydrogen and helium. If the earth was thrown off from the sun, it would have to come through this atmosphere, and it would be possible for the revolving plastic mass to occlude a certain amount of the gases through which it came, and to retain these permanently in the interior. As changes went on in the earth's crust, helium and hydrogen would take the path of least resistance and, therefore, would accumulate where other gases were. This is a plausible, if not a correct, explanation for the presence of helium in natural gas, and would also explain the richness of some of the gases from mineral springs. The difficulty in connection with this theory, however, lies in the fact that the helium seems to be more or less localized in the United States, although this is not a vital objection.

Another explanation might be called a modified radioactive origin. It has been shown that if the velocity of the alpha particle drops below 5000 miles per second, the alpha particle will not ionize gases and, therefore, the methods now used for the recognition of radioactivity would be inadequate, as such methods depend on the ionization of the air by the rays from the radioactive substance. The possible radioactivity of common elements that normally are not considered to be radioactive has, therefore, received some attention. Such a common element might be radioactive with the elimination of alpha particles with a velocity of 3000 or 4000 miles per second, and such radioactivity could not be recognized at the present time.

Potassium is very slightly active, due to slow beta rays; no alpha rays have yet been detected. If potassium minerals give rise to alpha rays there should be an accumulation of helium in the minerals. This has been tested by Strutt in connection with the potash deposits of Stassfurt, where he found that the potash minerals contained a much larger amount of helium than should have been present based on the radioactivity of the mineral. The sodium and magnesium minerals do not show this anomalous condition. Table 2 indicates some of the results he obtained. It would seem easy to settle this question by determining the amount of helium in potash minerals, such as potash feldspar, leucite, etc. Unfortunately the amount of material necessary is so large, and the quantity of chemicals, such as sodium and potassium carbonates required to decompose the minerals is so great, that it would not be possible to be certain that any minute quantity of helium obtained actually came from the mineral itself. It is only in case of soluble minerals, such as Strutt used, that a reasonable certainty of the origin of the gas can be assured.

TABLE 2

Mineral	Composition	Helium, Cubic Millimeters per 100 Gm.	Uranium Oxide (U_3O_8) per 100 Gm., Grams	Helium per Gram Uranium Oxide, Cubic Centimeter
Rock salt..........	$NaCl$	0.0233	7.1×10^{-6}	3.3
Sylvite............	KCl	0.55	2.15×10^{-6}	256
Carnallite..........	$KMgCl_36H_2O$	0.151	3.23×10^{-6}	47
Kieserite...........	$MgSO_4H_2O$	0.0179	6.47×10^{-5}	0.277

Recent discoveries, in western Texas, of soluble potash minerals seem to give some small evidence for the possibility of such an origin as has been described. Something more concrete and more definite, however, will be necessary before we can believe that potassium minerals have any real connection with the origin of helium in natural gas.

DISCUSSION

R. L. LLOYD, New York, N. Y.—Is it not possible, as a matter of economy, to use certain minimum quantities of hydrogen?

R. B. MOORE.—Yes, you can safely use 14 or 15 per cent., but the presence of that hydrogen is objectionable in connection with repurifying. In other words, it makes the repurification process rather dangerous, and we have decided that it is not worth while putting in hydrogen for that purpose.

R. L. Lloyd.—But in case of emergency you could use it.

R. B. Moore.—Yes.

C. M. Eye, San Francisco, Calif.—Is there any danger of the supply becoming exhausted?

R. B. Moore.—Yes indeed. This year, Congress is going to be asked to appropriate $5,000,000 with which to acquire one of these large gas fields and hold the gas in the ground

C. T. Young.—From what part of Canada does helium come?

R. B. Moore.—Alberta and Ontario. There are two sources of supply. The gas is about 0.3 per cent. in grade. Some of the gas geologists with whom I have talked on this trip say the supply is not very great.

J. K. Anderson, Charleston, W. Va.—From what parts of West Virginia did you obtain helium?

R. B. Moore.—Most of the gas in West Virginia has a small amount of helium in it, but none of it is very high.

R. D. Hall, New York, N. Y.—What process was used at the last where you got 18,000 cu. ft.?

R. B. Moore.—The Linde process. Of the three plants we used during the war, which were put into competitive action, the Linde Plant did the best work; therefore the big plant was built using the Linde Process.

R. D. Hall.—That is not the No. 1, is it?

R. B. Moore.—It was built from the design of the No. 1 plant. The No. 1 plant was a small experimental plant; the big plant is about eight times as large.

C. M. Eye.—The table showed Kansas considerably higher in helium than Texas or Oklahoma but lower in gas. How do you account for that?

R. B. Moore.—Most of the Kansas fields that are high in helium are declining rapidly in gas production; in other words, they are old fields. Unfortunately, we did not visualize the use of helium ten years ago, because then these fields were splendid fields for production.

Sherwin F. Kelly, Lawrence, Kans.—Would you consider the presence of helium in the deposits of those salts as proof of radioactive elements that we cannot detect by other means?

R. B. Moore.—One ton of uranium will produce in one year 110 cu. mm. of helium; you can figure how long it will take to produce 500,000,000 cu. ft. Therefore, knowing the amount of radioactivity and knowing the rate at which helium is produced, you can tell whether you are getting the right amount within limitations. Strutt has figured about 50 to 100 times as much helium in the Stassfurt potassium salts, as should be there, theoretically, but in the sodium salts about the normal amount.

J. K. Anderson.—How much would the initial rock pressure there increase your production?

R. B. Moore.—The cost depends partly on the rock pressure. On the newly designed semi-commercial plant we will run with a compression of 600 lb. High initial rock pressure helps very much.

C. T. Young.—Have the commercial quantities of uranium been tested for radioactivity?

R. B. Moore.—If there is any nickel in a uranium ore it is radioactive; and if it is radioactive, there is helium in it. Uranium is producing helium at a definite rate; whether it is in a gold ore or any other ore, the amount of helium is specific; for every ton of uranium metal, there is produced certain quantities of helium. Helium is produced both in thorium ores and in uranium ores because alpha particles are one of the things always associated with radioactivity

Pyritic Smelting and Basic Converting at the Kosaka Copper Smelter, Japan

BY KENZO IKEDA, DR. ENG., KOSAKA, JAPAN

(New York Meeting, February, 1923)

THE Kosaka smelter is situated in the extreme northern end of Hondo (the main island of Japan) 15 mi. east of Odate, on the government railroad, to which it is connected by a private railway. It contains ten blast furnaces having a maximum daily capacity of 1600 short tons of ore, conducting a slow pyritic smelting entirely without coke or limestone, and with a charge carrying over 50 per cent. of briquetted and agglomerated fines. The matte (35 per cent. Cu) is blown in two basic-lined barrel converters, the blister copper (98.8 per cent. Cu) being cast at once into anodes. The adjoining refinery contains 600 tanks with a maximum capacity of 1300 tons of cathodes per month, the byproducts being gold, silver, and bismuth. At present, working at about half capacity, the whole plant consumes 2000 kw. of hydro-electric power, and employs 850 men and women.

The ores are derived from two principal sources: The relatively new Hanaoka mine, 20 mi. from the smelter, yielding copper pyrites by underground mining, and the old Kosaka open-cut mine, 1½ mi. from the smelter, yielding three different ores: (*a*) *kuromono*, a compact fine-grained mixture of barite (32 per cent.), zinc blende, galena, and pyrite, the whole carrying about 2.5 per cent. Cu; (*b*) cupriferous pyrites with 2.4 per cent. Cu; (*c*) siliceous ore with 2 per cent. Cu. The Hanaoka ore now constitutes approximately one-half of the charge, almost replacing the Kosaka *kuromono* which was the ore chiefly smelted until 1913.[1]

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[1] Lack of space prevents the publication of a large number of details relating to the earlier operations with the Kosaka ores alone, which were particularly notable for their successful pyritic smelting of a charge carrying 17.6 per cent. $BaSO_4$, 5 per cent. Al_2O_3, 24 per cent. SiO_2, 2.3 per cent. Cu, 19.6 per cent. Fe, 6.2 per cent. Zn, 1.2 per cent. Pb, 19.6 per cent. S. This was accomplished without the use of any coke in the charge, but with the injection of 2.8 per cent. of coal through the tuyeres. Matte fall (30 per cent. Cu) was 6.6 per cent. and slag (0.32 per cent. Cu) was 70 per cent. of the charge.

Of the operations as now conducted, the most striking features are the abnormally small height of the blast furnace and its large tuyeres, the introduction of raw coal through the tuyeres, whereby the necessity for coke in the charge is almost completely avoided, the use of magnesite lining in the converters (its first application in Japan), and the novel method of placing this lining.

KOSAKA BLAST FURNACE

Of the several blast furnaces, varying somewhat in size, those having dimensions (at tuyere level) of 24.5 by 4 ft. (750 by 122 cm.) have given the best results. As shown in Fig. 1, it is a sump furnace with a forehearth at one end. Matte is siphoned continuously into a sand bed at one side, while the slag enters a removable settler (2 × 2 × 2.6 ft.—61 × 61 × 79 cm.) and flows thence to a granulating launder. The furnace has an open top with sectional sheet-iron covers, and a flue on each side. The water jackets are 2.8 ft. (77.4 cm.) square; on each side are seven flat ones with two tuyeres apiece and two curved ones at the corners, each with one tuyere; the two curved end jackets have no tuyeres. The tuyere, of 6 in. (15.24 cm.) diameter at its inner end, tapers only on its upper side, its lower side lying horizontal; opposing tuyeres are offset 4 in. (10.2 cm.) on centers. Other furnace dimensions are:

Total height	17.65	ft.	(538 cm.)
Ore column above tuyeres (max.)	9.0	ft.	(274 cm.)
Tuyere center to hearth bottom	3.0	ft.	(91.4 cm.)
Area at tuyere level (4 × 24.5 ft.)	98.0	sq. ft.	(9.1 sq. m.)
Total tuyere area (32 of 6-in.)	6.3	sq. ft.	(0.585 sq. m.)
Tuyere area to hearth area	6.4	per cent.	
Forehearth, 2 × 8.5 × 2.7 ft. deep, inside			(61 × 259 × 82 cm.)

The firebrick lining above the water jackets is boshed to a 7-per cent. pitch.

PRESENT SMELTING PRACTICE

The average charge contains the following, in percentages:

Hanaoka pyrites	46
Kosaka siliceous	20
Kosaka *kuromono*	3
Kosaka pyrites	2
Furokura copper ore	8
Misc. siliceous ores	3
Total ores	82

Hanaoka waste	8.5
Converter slag	6.0
Return matte fines	2.5
Blast-furnace flue dust	1.0
Total fluxes, etc	18.0

Of the above total, 45 per cent. is in the form of briquettes and 10 per cent. in blast-roasted sinter, which large proportion of fines has been

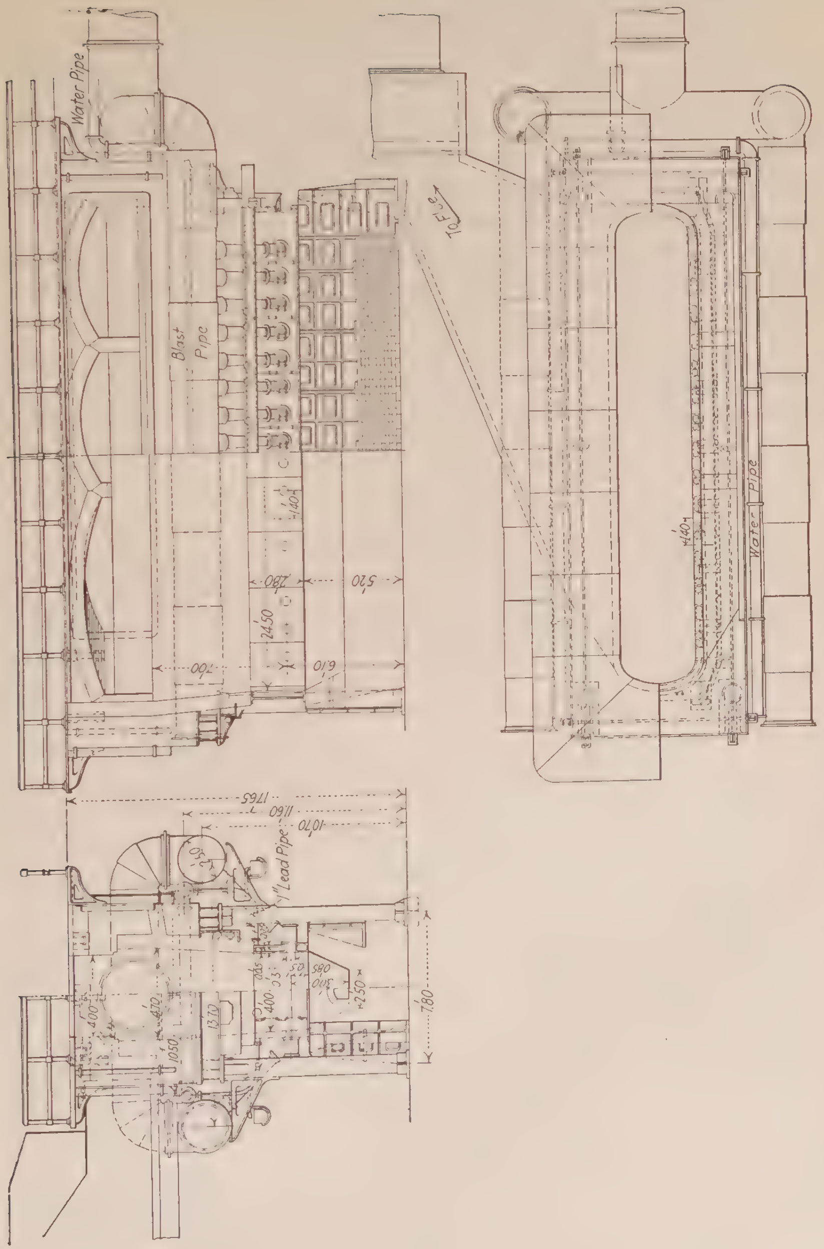

FIG. 1.—KOSAKA 24.5 BY 4-FT. BLAST FURNACE.

successfully smelted largely by reason of the low furnace burden and relatively weak blast (1 lb. per sq. in.).

The metallurgical results are as follows:

	Cu	Fe	Pb	Zn	SiO_2	Al_2O_3	$BaSO_4$	CaO	S	Total
Charge........	4.1	26.0	0.5	2.2	26.0	3.8	4.2	2	25.0	93.8
Matte........	24.0	42.0	1.5	2.8					25.0	95.3
Slag..........	0.29	29.8	0.2	2.5	42.0	6.1	4.6 (BaO)	3	0.4	88.9

Matte fall, 16 per cent. of charge, or 19.5 per cent. of ore; slag, 62.5 per cent. of charge, or 76 per cent. of ore.

The heat balance works out about as follows, it having been observed that only one-half of the total iron and one-third of the total sulfur contribute heat to the operation:

Evolved	Kg.-Cal.	Per Cent.	Consumed	Kg.-Cal.	Per Cent
Combustion of fuel........	285	42.2	In reaction...........	155	23.0
Oxidation of sulfide........	334	49.6	In fusion.............	246	36.5
Other sources.............	55	8.2	Heating and losses....	273	40.5
Totals.................	674	100.0		674	100.0

It is interesting to note that the combustion of fuel supplies almost exactly the amount of heat required for heating the charge and the escaping gases and providing for losses by radiation, cooling water, etc.

While the amount of blast theoretically required per 1000 kg. of charge works out at 947 kg., or 25,800 cu. ft., it has been found necessary to supply actually 100,000 cu. ft. (based on blower displacement), the losses being attributable mainly to the practice of feeding coal almost continuously through tuyeres, at least one of which will be open practically all the time.

This feeding of coal through the tuyeres is one of the unique features of Kosaka (and other Japanese) practice, and requires considerable skill and long experience. The most suitable coal is a compact, non-coking bituminous, high in calorific power and low in ash (18 per cent. ash has been found applicable at Kosaka); it should be of about nut size and comparatively dry, though not necessarily so dry as coal intended for pulverizing. The feeding instrument is a metal tube 2 ft. long and of 3.5 in. diameter (61 × 9 cm.) cut off obliquely at one end; through the other end is inserted a plunger, a disk on the end of a rod. When filled, it holds 3 lb. of coal. The usual charge at each feeding of a tuyere is 6 to 8 lb., being repeated at intervals of 5 to 10 min. One feeder usually attends eight tuyeres, and can feed about 2500 lb. (1134 kg.) per 8 hr., at which rate the four men to a furnace-shift can feed 16 tons of coal per day, or nearly 6 per cent. of the weight of charge. The present normal rate is about 4 per cent. of the charge, which seems about the minimum allowance to afford a satisfactorily fluid slag. Before feeding a charge of coal, the tuyere is punched; this has to be done carefully to avoid breaking the tuyere noses, which should normally extend 8 to 12 in. into the furnace. To assist in maintaining these noses, plastic fireclay is skilfully applied at the top and sides of the inner end of the tuyere, building up a foundation for a naturally formed nose; this is done only once or twice in each shift, and requires about 20 lb. of clay per tuyere per day.

As compared with the use of pulverized coal, this method of injecting nut coal has the metallurgical advantage, in addition to its freedom from the well known technical difficulties attending the use of coal dust for this purpose, that the fuel is consumed precisely at the zone where its heat will do the most good. The ability to smelt such an unusually low ore column without any appearance of a hot top is largely the result of this unique method of supplying fuel. Our experience has shown that, as compared with a 4-in. tuyere, the 6-in. tuyere not only facilitates the coal-feeding operation, but affords numerous other metallurgical advantages resulting in greater capacity, easier regulation, and smaller consumption of fluxes and coal per ton of charge.

Basic Converting

The first attempt at basic converting in Japan was made at Kosaka in July, 1920, and was successful from the start; whereas siliceous linings, owing partly to their poor quality, would not stand more than two blows, the magnesia linings of shell *B*, Fig. 2, for example, were serviceable after treating 15,100 tons of matte in 613 working days, or 5848 actual blowing hours.

Converter *A*, Fig. 2, was made from an old acid shell, which was only slightly remodelled by enlarging its mouth, providing an expansion joint between drum and end plates, and increasing the tuyere diameter to 1¼ in. (31.7 mm.), using tuyeres 7⅜ in. (27.3 cm.) apart. (Converters *B* and *C* were made in a similar manner.) One unique feature of its construction was the use of standard rectangular brick exclusively (2.5 by 4.5 by 9 in.) obtained from the Magnesia Industrial Factory at Amagasaki; these analyzed 80 per cent. MgO and 5 per cent. SiO_2, were of dark brown color, compared favorably with imported brick, and were usually harder and more accurately dimensioned than specially shaped brick. Some cutting, of course, was necessary. The brick were set in a mortar composed of fine dead-burned magnesia mixed with one-third its weight of boiled coal tar to which 1 or 2 per cent. of slaked lime had been added; the admixture of lime was found to accelerate the setting of the mortar. The bricks were laid with great care, the joints averaging not more than 0.025 in. (0.6 mm.) thick; in all, 2700 brick were required.

The backing, 3 in. (7.6 cm.) thick at the bottom and as thick as necessary to fill space elsewhere, was composed of dead-burned magnesia fines

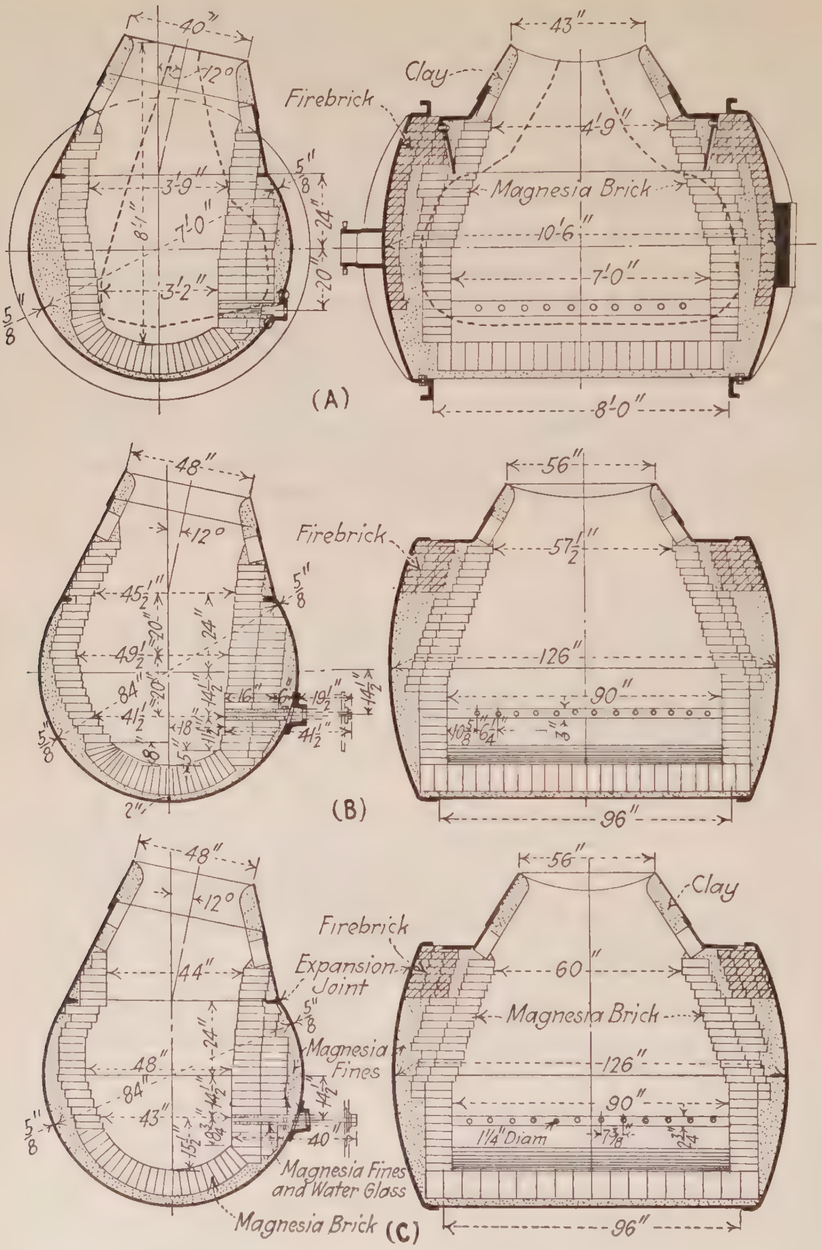

FIG. 2.—KOSAKA BASIC CONVERTER.

and grains moistened with about 7 per cent. its weight of 25° Bé solution of crude magnesium-chloride crystals; this made a plastic mass that could be rammed firmly into place behind the bricks as they were laid. The tuyere pipes were embedded in a layer 18 in. wide and 5 in. deep (45.7 by 12.7 cm.) of fine magnesia wetted with 17 per cent. its weight of a 40 per cent. solution of water glass.

Converter *A* as thus finished (in 33 days; at total cost of 5000 yen) had inside dimensions at tuyere level of 84 by 38 in. (213 by 96 cm.) or 22 sq. ft., and could hold 6 tons of matte when filled 1 ft. above the tuyeres; the depth of 10 in. (25 cm.) below tuyeres proved to be too small for the best operation, so this dimension was increased to 16 in. (41 cm.) in converters *B* and *C*. These two converters are in daily use, while a slightly modified converter *A* (12 tuyeres instead of 10) is held as a spare.

The operation of converting does not differ much from American practice. As a rule, about 8 tons of matte (37 per cent. Cu) are treated at each blow, starting with 4 tons and adding two installments of 2 tons, followed by scrap, cement copper, etc. The usual cycle is about 3.4 hr., yielding 3.5 tons of anode copper, or 45 min. net blowing per ton. The flux, weighing about 20 per cent. of the matte, is a siliceous ore (SiO_2, 53 per cent.; Fe, 18 per cent.; Al_2O_3, 4 per cent.; Cu, 1.8 per cent.; S, 20.5 per cent.) crushed to 1 in. size. Slag averages 57 per cent. the weight of matte treated, and carries 4.5 per cent. Cu, or 5.4 per cent. of the total copper charged. The direct recovery of copper in anodes (98.8 per cent. Cu) is about 93 per cent. of that charged. Blast pressure is 12 lb. per sq. in. (0.844 kg. per sq. cm.) and volume 315 cu. ft. (8.9 cu. m.) per min. per tuyere (10 of 1⅜ in. diameter). Based on the volume of air theoretically required to perform the reactions, and the volume actually supplied, as computed from motor input, the blast efficiency works out at 76 per cent., covering the entire blowing time.

The life of the basic linings is remarkable. Converter *B* was blown in on Oct. 6, 1920, and *C* on Feb. 14, 1921. To date (Aug. 10, 1922) *B* has converted 6750 tons of anode in 2131 blows during 654 days of continuous operation (excluding holidays) and is still in service with its original lining. *C* has produced 5700 tons of copper in 1650 blows during 531 days; the brick lining on the tuyere side is gone, but the backing is still serviceable.

"MAGNETIC" COATING

The unusually long life of the magnesia lining, in spite of its recognized liability to spall with changes of temperature, is due in large measure to the persistency of the iron-bearing coating; this may reach a thickness of 1 ft., although, theoretically, a very thin coating is sufficient. An analysis of this coating is compared with the average composition of the converter slag in the accompanying table:

	Magnetic Coating, Per Cent.	Average Slag, Per Cent.
SiO_2	3.79	15.76
Al_2O_3		1.05
MgO		0.22
FeO	4.75	39.67
Fe_3O_4	74.75	31.72
FeS	2.01	
S	(distrib.)	0.95
Cu	2.15	3.00
CuO	3.50	
Cu_2O	3.15	
Cu_2S	2.92	
Pb		1.18
Zn		3.34
	97.02	96.89

The freezing point of the slag has been determined as 1100° C., and the usual temperature of the converter bath is believed to be about 1200°; the melting point of the coating is close to 1275°, its iron-bearing constituent corresponding approximately to the formula $3(FeO).(Fe_2O_3)$, and hence not being true magnetite, though highly magnetic. All indications point to the fact that the usual working temperature is higher in basic-lined than in acid-lined converters; at Kosaka, the temperature is always only slightly below the fusion point of the protective coating, and momentarily, as during the white-metal stage, it may exceed that temperature. Thus the refractoriness of the protective coating against these temperatures, and its ability to re-form after dissolution, depend largely on the relatively high conductivity of the magnesia lining, which is purposely made as dense and conductive as possible. From this point of view, the conductivity of magnesite brick, which is about twice that of bauxite or chromite, is a distinct advantage instead of a detriment, as has frequently been supposed.

At the same time, loss of heat by radiation has not proved objectionably large; it actually figures out at about 25 per cent. of the heat evolved in the converter, compared with 31 per cent. as computed from data on acid converting at Globe.[2] In any case, no difficulty has been experienced in maintaining a satisfactory working temperature of 1200° C. without adding any fuel, which was formerly required with the acid converters, and at the same time a much smaller proportion of solidified copper remains behind in the ladle.

DISCUSSION

E. P. Mathewson, New York, N. Y. (written discussion).—This paper leads one to consider why the practice of introducing fuel to blast furnaces through the tuyeres has not been more generally used. Many years ago, in England, the iron industry tried introducing broken coal in blast-furnace smelting. In recent years, pulverized coal has been used to the extent of about one-third of the fuel required for the smelting operations, both in lead and copper smelting. The writer has heard that at Copper Cliff, in smelting nickel ore, the plan was tried of introducing all the fuel to the blast furnace through the tuyeres in the form of pulverized coal, but the trial was not kept up long enough to get sufficient data for future work.

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[2] Hofman, "Metallurgy of Copper," 329.

There is no doubt that pulverized coal introduced through tuyeres tends to carry the heat up the shaft to the top of the charge, and the same objections can be made to its use in this way as to the old-fashioned method of putting the fuel in at the top. Would it not be well to introduce at the tuyeres, instead of crushed coal, coke crushed to about 1-in. cubes? This ought to make for quicker smelting, as it is well known that the use of coal as fuel in a blast furnace slows down the furnace. The method described by the author is particularly applicable to pyritic smelting. The paper emphasizes the fact that high pressure is not necessary in pyritic smelting; also, that pyritic smelting can be carried on with a cold top.

It is interesting to note that the converting of copper matte in basic-lined shells has become the general practice in Japan. The Japanese usually take up new ideas very quickly, but, in this instance, it was left for Doctor Ikeda to introduce this important improvement in converter practice. He places the date of its introduction in Japan as July, 1920. Doctor Ikeda was fully conversant with the value of the "magnetic" coating as a protection to the magnesite lining and as an indicator of temperature in the converting operation. Temperature control in basic converting of copper matte is most important. The analyses of magnetic coating and average converter slag are very interesting and conform closely to those in this country.

W. R. Ingalls, New York, N. Y. (written discussion).—This paper is a model of what a metallurgical contribution of its own kind ought to be. It describes a new practice in a lucid way and with great condensation. Much is accomplished in such a paper by avoiding the presentation of unnecessary details.

In the smelting of Kosaka from 4 to 6 per cent. of carbonaceous fuel is used and one of the objectives of the new process is to make better use of it. The plan of action is to get raw coal of nut size into the furnace through the tuyeres, so that the fuel will be consumed precisely in the zone where its heat will do most good, and concentrate it more than if the fuel were introduced from the top of the furnace, or as powdered coal blown in through the tuyeres.

The author gives an illuminating heat balance, which in a few lines conveys a perfectly understandable picture. About 42 per cent. of the heat generated in the furnace is ascribable to the combustion of carbonaceous fuel. The thermal efficiency of the furnace appears to be about 60 per cent., but undoubtedly it is even higher, for the heating of the charge, which is not separately reckoned, is nevertheless useful work. Anyhow, the figures indicate a high efficiency, which is the more noteworthy when it is considered that about four times the quantity of air is required for blowing the furnace than is theoretically estimated. This is one of the penalties of feeding coal almost continuously through the tuyeres, at least one of which will be open practically all of the time. Attention may be drawn also to the labor that is required, four men per furnace-shift being needed to feed 16 tons of coal per day. If the coal is 5 per cent. of the charge, this means 12 man-days for this purpose in the smelting of 320 tons of charge. Clearly therefore the wage scale for labor is an important consideration in the application of this process.

J. Parke Channing, New York, N. Y.—While in the United States, the author visited Tennessee. The difference in the operations at Kosaka and Tennessee is not very great. He succeeds in operating with a column 9 ft. high; in Tennessee we operate with a column 12 ft. high. The total height of our furnace from tuyeres to feed floor is 18 ft. We have tried numerous heights of column, but have found 12 ft. most satisfactory.

In the last 15 years, we have tried all the combinations, and are now using what is known as straight Burra-Burra ore (from Burra-Burra mine) carrying about 25 per cent. sulfur. It is a fine mixture of pyrite, pyrrhotite and chalcopyrite, with a siliceous gangue, carrying probably 16 per cent. silica and a small amount of lime, magnesia, and some alumina.

The object in Tennessee is, primarily, to make a gas sufficiently high in SO_2, sufficiently low in CO_2, and containing enough oxygen to permit its conversion into sulfuric acid. The secondary object is to conserve the copper in the form of a matte. When we put fines into the charge, the furnace does not run regularly and we get a low-grade gas. By taking out the fines and running the coarse material, we get a gas, when the converters are operating at the same time, which will run about 7 or 7.5 per cent. SO_2 and 2.5 per cent. CO_2. In order to smelt properly, it is desirable to keep the fuel as low as possible; otherwise there will not be the necessary amount of free oxygen in the gas to convert the SO_2 into SO_3. Last week we were running on about 4 per cent. coke, which was dumped in at the top. We charge consistently Burra-Burra ore, field quartz, and coke—nothing else. We smelt, roughly, about 900 tons of Burra-Burra ore per day in a furnace that is about 56 in. across at the tuyeres and 32.5 ft. long. There are two Nordberg blowing engines, each delivering 30,000 ft. of air to a furnace.

The furnace operations could be much improved, as far as the matte is concerned, if we added more field quartz and barren limestone. But the present method is the most economical. The slag runs about 36 per cent. silica. It is high in FeO, as it should be; consequently the slag loss is high. I think the slag will run at least 0.3 per cent., and the matte averages about 12 per cent. copper. That matte we treat directly in a large Pierce-Smith converter and the gases from that converter are most excellent, averaging about 6 per cent. in SO_2. They combine with the blast-furnace gases and go to the acid chambers.

Some years ago, when Doctor Ikeda was there, we tried putting coal into the tuyeres; but the labor cost was prohibitive. The results, however, seemed so satisfactory that we tried the use of pulverized coal. Almost simultaneously the people at Copper Cliff did the same thing. We erected a pulverizing plant, put a series of tuyeres in one side of the furnace, and blew in pulverized coal for two or three days, when the furnace did run with fine, pulverized bituminous coal. During this time there came off a certain amount of carbonaceous material, probably very finely divided, which went through the flues, through the Glover tower, into the acid chambers, and remained in the acid. As a result our acid was very dark. This impurity did not in any way reduce the value of the acid for the purpose of making fertilizers, but other customers strongly objected to that black acid, so we had to abandon that method.

I agree with Doctor Mathewson that if the coal were introduced in the furnace in a larger size, or if we introduced coke, we would get it exactly where it is wanted, right down in the zone of fusion; and we could get along with less than we are using now. But we have not been able to develop a method for introducing the coal or coke mechanically, in medium size without excessive cost. So we get as good and strong a coke as we can, dump it in at the top, and trust that all of it won't burn up before it reaches the zone of fusion.

Doctor Mathewson said that in Scotland some years ago they introduced coal at the top of the furnace. They are still doing it on a very large scale and are obtaining as one of the products from that operation an oil known as blast-furnace oil, which is a most excellent flotation reagent.

We are converting the 12 per cent. matte in a large Pierce-Smith converter, which is 12 by 25 ft., using for flux, London ore, which is a very siliceous ore, and a little field quartz. In addition, we are trying the experiment of putting in the flotation concentrate with a Garr gun. During the two days I observed it, it seemed to be going successfully. This large Pierce-Smith converter is only tapped for copper once a day. Our ore runs about 1.75 per cent. copper, and if our extraction is about 1.35 per cent. that means 27 lb. to the ton, or 24,300 lb. of copper daily from green ore, to which is added the copper from concentrates.

The smelter superintendent told me that for each ladle-full of matte put into the converter, he takes out a ladle-full of slag. The slag from the converters is very fluid; it runs about 21 per cent. silica and is poured into the furnace settler. When I was there in the early days of smelting roasted ore, we carried on that practice and were unable to find that the slag from the settler was increased by pouring in the converter slag. Since the plant has been running on low-grade matte, the indications are that the slag in the settler increases from 0.05 to 0.1 per cent. in copper over and above the normal slag from the blast furnace by pouring the converted slag into the settler. From a mechanical point of view, it is an easy way to get rid of the converter slag and in our case it is an economical way.

In a personal letter, Doctor Ikeda says: "The plant is doing as fair work as ever and I am going to handle 25-per cent. copper matte in our basic converters, which are now treating 37-per cent. matte, by eliminating matte concentration in the near future." You can see from his report that he makes in the neighborhood of 25-per cent. matte, and although he does not refer to it, I presume he wants to run one furnace as a concentrating furnace and bring his 25-per cent. matte up to 37-per cent. matte. We did that in Tennessee at first but found that a properly designed basic converter can handle a 10-per cent. matte economically.

So far as Tennessee is concerned, elimination of the iron and sulfur is really in two stages, the first being the blast furnace and the second the converter. Doctor Ikeda's rate of concentration is six to one; he is running on a higher grade ore than we are, our rate of concentration being seven into one. If Doctor Ikeda will put in a large converter, I am sure he can get on without the necessity for any concentration of his matte. In Aguas Calientes, some twenty years ago, they did treat a 33-per cent. matte, much to their astonishment.

C. P. Linville, Elizabeth, N. J.—In southern Ohio, in the making of high-silicon irons, they are putting in, along with their ore charges, fully as much coal as they do coke. The blast-furnace owners down there produce a non-coking coal from their own coal mines.

E. P. Mathewson.—Pulverized coal has been successfully introduced into the tuyeres of blast furnaces in several places. In lead smelting, the most successful installation was at Midvale, Utah, where they were able gradually to replace up to 33 per cent. of the fuel with pulverized coal, without materially reducing the speed of the furnace; above 33 per cent., however, they reduced the speed of the furnace beyond the economical point.

The great value of pulverized coal added to the blast furnace in that way was the perfect control and the ability to correct rapidly the reduction in the furnace; getting an acid slag and finding the reduction was not sufficient, an additional amount of pulverized coal was added at once and everything was cleaned up.

The arrangement at that particular plant is peculiar. The coal is carried from a distributing box by a rubber hose, to a funnel just above the tuyere, and it is forced into the furnace by a high-pressure blast. A little space between the funnel and the rubber hose makes it possible to observe the stream of coal; when it is desirable to determine how much coal is going into a particular tuyere, a scale pan is placed under the hose and the whole stream of coal is cut for a few seconds. This is weighed on an apothecary's scale, the amount is easily calculated and the correction readily made, because the amount of coal can be varied by opening or closing the valve. It is estimated that in the case of these small lead furnaces this method of introducing fuel effects a saving of $1000 a month.

In semi-pyritic copper smelting, this method was tried, at the Garfield plant of the American Smelting & Refining Co., with fair success. About the same results were obtained; that is, it was possible to replace 33 per cent. of the coke by the same weight of pulverized coal. For some reason, however, the company thought that the saving in fuel did not justify extending the practice. Further, its amount of ore suitable for blast-furnace work is rather limited, and it did not want to extend its operations in the matter of pulverizing. It has used pulverized coal in its reverberatory furnaces, however.

At the International Nickel Company's plant, the introduction of pulverized coal in the tuyeres was tried, just before the plant shut down, owing to the failure of the nickel market. The one who was trying to introduce the process got permission to replace all of the coke on the charge, for about 48 hr. During that time, the furnace went faster than before. American metallurgists have been discouraged because the practice was abandoned at the plant in Tennessee, which was one of the first to introduce it. We have just learned that there was a good reason for discontinuing the practice, but that reason does not apply to plants smelting copper and not trying to make acid. So I believe there are big possibilities for pulverized coal.

Another place where pulverized coal is being used in this manner is in the Cerro de Pasco smelter at Casa Palca, Peru. I understand that those furnaces have been started and that the results to date are very satisfactory.

My idea in suggesting broken coke at the tuyeres was to maintain the speed of smelting. When introducing coal with coke, in the charge of a blast furnace, I have always found that the furnaces slowed down; the same thing occurs when pulverized coal is put through the tuyeres, but if coke is used in that manner, this does not occur. Of course, coke costs a little more than coal, and the one great advantage claimed for pulverized coal was that it was a cheaper fuel. If the speed of smelting is considered, it would be well worth while to consider also the use of broken coke.

Owing to the high labor cost of introducing the coal or coke through tuyeres by hand, that method becomes possible only where labor is cheap. But there should be little or no difficulty in devising an apparatus for introducing broken coal mechanically.

Arthur L. Walker, New York, N. Y.—In the Lake Superior region, they have been charging broken coal through the tuyeres of the copper cupola in order to obtain better reduction. More attention has been paid to this practice in Japan than in this country; at the large plant of Furukawa Mining Co., at Ashio, 1-in. coal is charged through the tuyeres.

R. C. Canby, Wallingford, Conn.—R. Nenninger, in 1890, fed a large portion of the fuel through the tuyeres of his furnace at the Balbach plant. I do not remember the details of the tuyere, but if the coal was not pulverized, at least it was fine fuel. Unfortunately, the deal that was made with the mine was such that Mr. Nenninger's plan was not continued.

E. P. Mathewson.—There seems to be a misapprehension of one remark I made in my earlier published discussion. It is thought, apparently, that I refer to the introduction of coal in the top of the furnace. When I say, "Many years ago, in England, the iron industry tried introducing broken coal in blast-furnace smelting," I should have added, "through the tuyeres." A patent was issued in the early 70's for introducing fuel by that method.

Heap Leaching at Bisbee, Arizona

By A. W. Hudson, Bisbee, Ariz., and G. D. Van Arsdale, Los Angeles, Calif.

(New York Meeting, February, 1923)

Heap leaching, as practiced at Rio Tinto, Spain, while one of the oldest, and probably one of the cheapest, methods of extracting copper from its ores, has not had, until recently, other than experimental application in this country. This has been due partly to a mistaken idea that the Rio Tinto ores possess some obscure and mysterious quality that renders them alone suitable to the process, partly to the fact that tests properly made meant the expenditure of large sums of money and several years time, and partly to the fact that favorable commercial and other conditions are not often found.

This paper gives some views on the chemistry and mechanism of the method, an account of the preliminary experiments, and the final plans adopted for a large-scale installation of the method by the Phelps Dodge Corpn. for treating low-grade ore from Sacramento Hill, Bisbee. If results comparable with those obtained from the test heap are to be had from the large plant, this plant will operate on the lowest-grade sulfide ore now being treated commercially in the country.

The method has several advantages: The installation cost is low. Interest charges must, of course, be made both on the cost of the ore delivered to the plant and for the plant itself, but as these are low, there is considerable advantage in the ability to regulate output by more or less complete shutdowns in accordance with market and other conditions. The amount of labor needed is small, and the cement copper produced may be considered nearer a finished product than the sulfide concentrates from a mill, and certainly means less installation cost per ton to put into final form by present methods.

At Bisbee, conditions were favorable for a trial of the method for, in the preparation of the Sacramento Hill orebody for steam shoveling,[1] a large amount of material below concentrating grade had to be moved to a suitable dumping ground, so the cost of delivering this to leaching beds meant practically no extra expense. The only risk was the cost of the plant which, it was thought, could be amortized by an actual extraction sufficiently less than that obtained experimentally to make the proposal reasonably safe. Up to October, 1921, approximately 380,000 tons, from stripping operations, averaging 0.92 per cent. copper had been delivered to the first heap, but since that time, because of the suspension of stripping and other conditions, no deliveries have been made, and plant construction has been postponed. That part of the paper, therefore, dealing with proposed plant construction represents recommendations and plans made up to that date; whether they will be followed will depend on the future policy of the corporation.

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[1] H. M. Ziesemer and George Mieyr: Steam-shovel Operations at Bisbee, Ariz., *Trans.* (1923) **68,** 251.

The plant was designed to take care also of the mine water formerly treated at the cementation plant at Bisbee; it embodies certain improvements locally worked out as the result of experience there, which are expected to result in economy.

Factors Involved in Heap-leaching Process

While the process of heap leaching is a simple one to operate, so little is known accurately of the chemistry involved that it is reasonable to hope for modifications in operating methods that will shorten materially the time required for treatment.

Sponge iron, if it is produced commercially at a sufficiently low cost, will be a much more satisfactory precipitant than the tin cans and scrap iron now used; either this product or pig iron will have to be used eventually, for the present supply of tin cans and scrap is not sufficient to cover much extension of cementation in the southwest.

We can easily define heap leaching in terms of its flow sheet and operations, which are possibly simpler and require less expensive and complicated apparatus than any other method of getting copper from ores. It cannot be said, however, that we know exactly, or even approximately, the reactions in the heaps by which the copper is converted into forms soluble in water.

There is no doubt that the kind of copper mineral, its physical character, and the manner of its original deposition are important factors. We can be quite sure that some ores cannot be leached, if it is known that they are quite free from porosity. This follows from the fact that it seems certain that extraction of copper by this method depends on each piece of rock having pores, or microscopic channels, either open or filled with material that can be acted on by the solutions, and which will permit the solutions both to penetrate and to leave the interior of the piece. It would seem probable, therefore, that ores in which the copper is of secondary origin (that is, enriched by the precipitation of copper from solution in microscopic channels, pores or fractures) will be amenable to heap leaching; and that if an ore, in which the copper is of primary origin, is lacking in pores leading from the outside of a lump to each particle of copper mineral, it cannot be treated by the method. We are, however, dealing in reactions and operations that require at least several years; and while small-scale tests can be made to show definitely that an ore can be treated, negative results in a small way are not conclusive evidence of large-scale results because of the time factor and the difficulty of reproducing large-scale conditions.

From the foregoing, there can be deduced one necessary factor in the method of large-scale operation, namely, that each piece of ore in the piles should alternately be wet completely and dried thoroughly. If any individual piece having the necessary pores or channels and thoroughly dry, is wet on the surface, in a reasonable time each passageway, by reason of capillary action, will be filled with the solution wetting the surface, providing the channels are open at both ends. No reasonable amount of washing will remove the solution from the interior; but if the surface is dried, the reversal of the capillary action will bring the solutions to the surface of the piece, bringing with them any salts dissolved on the way. This action can be observed easily on the outside of a pile or an individual piece, which when dry becomes coated with quite pure bluestone.

If the foregoing is correct, it will give an idea of the amount of solution theoretically necessary for application at each cycle. Assuming that a pile or an individual piece of rock is dry, all that can be accomplished in the way of extraction by a single application of solution or water is to wash off the soluble salts on the surfaces, and no greater quantity than is required for this purpose need be used; a larger amount means simply more dilute effluent liquors. Also, as the volume of the pores is comparatively very small, no large amount of solution is needed for filling them; the amount of solution remaining on the surfaces after washing is probably enough. It is certain that complete immersion of the ore or the use of an excessive amount of solution is unnecessary and undesirable.

In heap leaching, it is probable that carbonates, cupric oxide, cuprous oxide, and metallic copper will be attacked in the order named. Copper silicate is an indefinite compound, with widely varying character and solubility, so no definite statement can be made with regard to it. It seems probable, although not certain, that if an ore carries any considerable amount of its copper as oxide compounds, a certain amount of sulfuric acid will have to be used.

Chalcocite and bornite are readily attacked and dissolved, but chalcopyrite is more refractory. It is probable that an ore containing chalcopyrite as its principal copper mineral will be much more difficult to treat by the method.

The reactions are exothermic, and where, as at Rio Tinto, heaps consist entirely of heavy sulfides, careful attention must be paid to keep the resulting temperature below the ignition point.

A very important matter practically is the character of the gangue and its behavior in the heaps under the influence of the solutions.

As the method depends on alternate wetting and drying to provide for penetration of solutions into a lump of ore and their subsequent removal, the size of lump and proportion of fines in relation to depth of pile are important. In addition, some rocks, under the action of the solutions and alternate wetting and drying, may crumble more or less completely. If the resulting products are coarsely granular, this crumbling is not serious; it may even be advantageous in allowing quicker and more complete penetration of solutions. But when a rock breaks down into very fine or claylike particles, it will be more difficult to leach and the uneven distribution of the effects of such breaking down will result in uneven leaching. Data are not available to show how an area in a pile consisting entirely of such completely broken down rock will behave. The first experimental test of the method at Douglas, however, consisted of 25 tons of sand tailings. This small heap allowed complete penetration of solution which, when the heap was allowed to dry, showed marked tendency to return to the outside surface by capillary action, bringing dissolved salts to the surface. The factor of ease of drying comes in here, however, and it is possible that the depth of pile with a rock that breaks down easily will have to be reduced; but so many factors are involved that at present the depth of pile is purely experimental.

The practice at Rio Tinto is of little help in this or other matters, as the heavy pyrite and the siliceous ores leached there vary widely in composition and behavior from our southwestern porphyries.

These statements concerning some of the physical factors involved show that the method is not so simple as the operation would indicate, and the same is true of the chemistry. It will also prove to be true that both successful operation and any possible improvements can only be had by as complete a knowledge of the physical and chemical factors as possible. This sounds trite enough, but frequently expensive failures have been made in apparently simple proposals through neglect of this obvious consideration.

Chemistry of Process

Turning to the chemistry of the process, we can be quite sure of the occurrence of some reactions. So-called oxidized copper compounds will be present in appreciable amounts in the ores under consideration. Cupric oxide, carbonate, or silicate will be probably acted on as follows:

1. $CuO + H_2SO_4 = CuSO_4 + H_2O$
2. $2CuO + Fe_2(SO_4)_3 + H_2SO_4 = 2FeSO_4 + 2CuSO_4 + H_2O$
3. $3CuO + Fe_2(SO_4)_3 + 3H_2O = 3CuSO_4 + Fe_2(OH)_6$
4. $CuO + FeSO_4 + H_2O = CuSO_4 + Fe(OH)_2$

depending on the composition of the solutions and the amounts of ferric and ferrous iron, and free acid present.

Soluble basic constituents of the ore, other than copper, will be acted on similarly, resulting in reduction of acidity and precipitation of iron compounds. It is therefore evident that maintaining the balance of iron and acid in the solutions is important practically, and experiments have shown that some ores require the addition of extraneous acid to maintain this balance, while others do not.

Theoretically, so far as iron is concerned, the reaction

3. $3CuO + Fe_2(SO_4)_3 + 3H_2O = 3CuSO_4 + Fe_2(OH)_6$

will be exactly balanced by the precipitation reaction

5. $CuSO_4 + Fe = Cu + FeSO_4$

This may or may not be the case practically. Both free acid and soluble iron salts are produced in the piles by reaction of the sulfides present, and may be sufficient to make up the losses, caused mainly by the usual methods of precipitation, by which basic salts of iron are formed in considerable amounts.

In large-scale testing of the method at Tyrone, acid was added to the liquors, but at Bisbee this was not done.

Another important difference between Rio Tinto practice and the application of the method to our ores is that there will be a smaller quantity of waste liquors to be disposed of in normal operation. This is obvious, as iron is the leaching reagent and it is not produced in excess as at Rio Tinto.

We have assumed the delivery to the plant of 300,000 gal. of mine water per day, which has to be treated in any case, and the pumping capacity needed for the leaching heap has been based on using 1¼ gal. per ton per day. Treating 400,000 tons of ore for the first year would require 500,000 gal. of water for leaching per day. Soakage and evaporation is estimated at 50,000 gal. per day so that in order to equalize the water problem the following amounts will be needed per day (soakage is the amount of water taken up by the ore):

Year	Ore	Mine Water Gallons	Loss	Solution off Heap	Return to Heap	Waste
First	400,000	300,000	50,000	500,000	250,000	250,000
Second	800,000	300,000	75,000	1000,000	775,000	225,000
Third	1200,000	300,000	100,000	1500,000	1300,000	200,000

From these figures it appears that after the second year the amount of waste solution will not exceed about 13 per cent. of the total from the heap, and is less than the amount of mine water added to the system.

There will be an advantage in returning solutions from one part of the heap to other parts, in order to decrease the ferric-iron content of the effluent liquors before sending these to the cementation plant. This method should increase the extraction from the heap and decrease the quantity and increase the grade of the solutions. Furthermore, it should postpone the addition of further units of the cementation plant for the second and third year of operation for the quantity of solution to be depleted of its copper will be approximately the same. If this plan works out successfully, it will also delay the building of reducing heaps.

The reactions by which the sulfides of copper are converted into water-soluble sulfates in the heaps are not known with certainty. The equations given in the latest description[2] of operations at Rio Tinto are as follows:

6. $33FeS_2 + 28O + 4H_2O = 29FeS_2 + 4FeSO_4 + 4H_2SO_4$
7. $4FeSO_4 + 4H_2SO_4 + 2O = 2Fe_2(SO_4)_3 + 2H_2O + 2H_2SO_4$
8. $Cu_2S + Fe_2(SO_4)_3 = 2FeSO_4 + CuSO_4 + CuS$
9. $CuS + Fe_2(SO_4)_3 + H_2O + 3O = 2FeSO_4 + CuSO_4 + H_2SO_4$ or
10. $33FeS_2 + Cu_2S + 3H_2O + 33O = 29FeS_2 + 2CuSO_4 + 3H_2SO_4 + 4FeSO_4$

showing the oxidation of pyrite to form ferrous sulfate and free acid; the oxidation in solution of ferrous to ferric iron; and the solvent action of ferric sulfate on Cu_2S and CuS.

Similar equations have been published for years, but they do not really explain what takes place, and it is difficult to see why they should be written in the forms given.

It seems certain that an important reaction accounting for the solution of copper from a sulfide mineral is

$$CuS + Fe_2(SO_4)_3 = CuSO_4 + 2FeSO_4 + S$$

but unquestionably other reactions take place, and it cannot be said that the chemistry of the process is known with any degree of certainty or completeness.

In the absence of direct evidence, a definite explanation cannot be given, but some of the observed facts point toward what follows.

We have to explain that starting with sulfide of iron and sulfide of copper, in the presence of excess of air and moisture, there are produced in the heaps ferrous and ferric sulfate, free sulfuric acid, and copper sulfate. The theory of capillary action seems quite clearly to explain how copper can be removed from the interior of a piece of ore without showing any macroscopic changes, and if the progress of leaching is followed by a proper microscopic study, this will probably be verified and the actual way in which the solutions get in and out followed. This has been done to a certain extent. It is simple enough to explain the formation of ferric iron from ferrous iron in the presence of excess of air.

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[2] Courtney De Kalb: Metallurgical Methods at Rio Tinto. *Min. & Sci.* Pr. (Feb. 5, 1921).

The direct oxidation of pyrite by oxygen to produce SO_2 and iron oxides is a reaction that starts slowly at quite low temperatures, but the velocity increases rapidly with rising temperature. This reaction may, or may not, have a measurable velocity at ordinary temperatures, but at the lowest temperatures at which it will occur it is probable that moisture will, as it does for many other reactions, act catalytically to increase the reaction velocity. A determination of the lowest temperature at which this reaction occurs appreciably should be made, and the influence of moisture on it; but as there is no evidence to the contrary and this explanation fits the facts it may be the correct one. As the reaction is strongly exothermic and as there is very inefficient radiation in the heaps, the heat produced at the beginning will accumulate and act as an accelerator; we would expect in a pile of closely packed pyrite with small amounts of moisture present, a constantly rising temperature.

This is the "heating up" that has been so often described for Rio Tinto. This accumulation of heat may be sufficient to start the heaps burning, to prevent which, care must be exercised. This is exactly what we would expect if the above were the correct explanation of the reaction taking place.

The velocity of the reaction at low temperatures is probably very small, but we are dealing with reactions that take several years for completion.

The minute amounts of SO_2 produced at any one time are under favorable conditions for conversion into SO_3, as they are in the presence of relatively enormous surfaces capable of acting as contact surfaces. If oxidized to SO_3, any oxide of iron produced, being in intimate contact with the acid, will be readily dissolved in the presence of moisture.

The foregoing, if true, will be as valid for sulfides of copper as for sulfides of iron. It is well known that some copper sulfides, under the influence of air and moisture, oxidize readily to form sulfates and that other sulfides do so much less readily; this probably explains the particular amenability of the Rio Tinto ore to the process, although the experimental results at Bisbee would indicate that the reactions take place easily with the forms of copper sulfide present in these ores.

The relative amounts of copper dissolved by direct oxidation of copper to sulfate and by direct solvent action of ferric salts cannot be determined; the latter reaction is no doubt important, but careful records kept of the ferric and ferrous iron entering and leaving the experimental piles did not disclose any relation between the amount of ferric iron entering and the copper dissolved.

If the foregoing is correct, it is evident that, at Rio Tinto, the most favorable conditions for rapid oxidation cannot be maintained without danger of spontaneous ignition of the heavy sulfides; but this is not the case for a pile of siliceous ore, for which the best conditions can be selected. The investigation has not been carried on sufficiently to determine what these conditions may be. It may therefore be possible that the conditions under which the test heaps at Bisbee and Tyrone were run were not the most favorable, and that further work will result in an appreciable shortening of the time required for extraction.

The recorded history of the method at Rio Tinto goes back many years, as stated by Mr. DeKalb.[3] The precipitation of copper by immersion of iron in solutions of copper salts is one of the oldest known chemical facts, and as, generally speaking, iron has always been cheaper than copper, it is probable that the method was applied at Rio Tinto for many years before the recorded development of efforts at systematic use.

In this country precipitation of copper from mine waters has been carried on for many years, at various localities, but until recently no serious attempts have been made to apply heap leaching. The first work done on the method by the Phelps Dodge Corpn. was about 1900. After a visit to Rio Tinto, when Doctor Douglas was impressed with the possibilities of the process for Bisbee, some experiments were made to determine, first, whether heavy pyritic ore carrying copper from the Copper Queen mine could be treated; second, if at the same time the excess of acid and ferric salts produced could be used to leach the copper from low-grade oxidized ores in "reducing beds" similar to those at Rio Tinto. From the records of these tests, the main criterion seemed to have been as to whether or not the test pile would "heat up;" and as no "heating up" was observed when water was applied to the heap, the experiments were pronounced a failure. The addition of chlorine, as common salt, to the liquors appeared to give favorable results, but was too expensive to be practicable. The role of iron as a reagent and the necessity for its presence in the leach liquors was not recognized in these early tests.

It was shown later, by laboratory work, that the presence of iron salts in the water used for wetting the ore was an essential factor, and it was proved that, by attention to this and other necessary conditions, low-grade ores of various kinds could be leached in the laboratory by the method.

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[3] *Loc. cit*

During the large-scale leaching and electrolytic tests at Douglas, authorization was obtained for a trial of heap leaching. The material used for this test was sand tailings from the Tyrone mill, about 25 tons of which were placed on a platform and treated for several months. The results obtained were sufficiently encouraging to recommend a large scale test at Tyrone. A brief description of this experiment, which was carried on by A. W. Hudson, is as follows:

Tyrone Experimental Heap

The ore for the test had mostly been produced from development work and had been dumped adjacent to the various tunnels or shafts. Some of the ore had been in the dumps for over seven years, while some was from recent operations.

The leaching site was selected on account of its proximity to the ore. It consisted of a hillside with a natural slope of between 12 and 15 per cent. This was too steep to retain the slimes, so surface benches were graded with a gradual slope following the contours of the hill. The area was about 250 by 250 ft.

To assist in waterproofing the ground, mill slimes were spread to a depth of about 6 in.; culverts were made from the largest rocks.

About 20,000 tons of ore were elevated to the site and distributed with wagons, making a heap with an average depth of 6 ft. The following is an approximate analysis of the material: Cu, 2.71 per cent.; SiO_2, 66.0 per cent.; Fe, 6.0 per cent.; CaO, 0.3 per cent.; S, 5.0 per cent.; Al_2O_3, 14.5.

Scrap-iron precipitation launders were built ahead of the heap so that all solutions leaving the heap were pumped to the plant, where they were depleted of the copper, enriched in iron content, and flowed by gravity to the heap again for washing purposes.

Leaching operations were commenced during February, 1917, and continued intermittently for three years, at which time further work was discontinued because the concentrating mill required all the water from the mine.

During the three years, a total of 38.6 per cent. of the copper was extracted, figured from measurements and assays of solutions on and off the heap. The addition of acid was found necessary.

Following the Douglas leaching tests, considerable experimental work was done at Bisbee by various methods on low-grade ores, mostly carrying acid-soluble copper. After the discovery of the extensive disseminated deposits of Sacramento Hill, and their exploration, it became evident that these contained large amounts of ore too low in grade for concentration, so recommendations were made by the research department of the Phelps Dodge Corpn. that a systematic and complete test of heap leaching be made on this material. This work at Bisbee has been described by Joseph Irving, who was in charge of the operation of the experimental heap.

It is an interesting fact that the proceeds from this small heap paid all expenses of operation, and that the actual cost of the copper produced compared favorably with that of other much larger scale operations during the same period. Experimental work paying for itself is sufficiently rare to make this worth noting.

Bisbee Experimental Heap

A dump containing a quantity of low-grade sulfide ore that had been mined from an air shaft two or three years previous provided the ore for the test.

The leaching site selected for the heap was in a gully, part of the floors being covered with old lumber and part dressed with slimes. The bed of the creek was used as the main drain into which all other culverts drained. The surface area of the heap was about 12,000 sq. ft. About 10,000 tons of ore were moved to the prepared site with mine cars, making a heap with an average depth of approximately 20 ft. The following is an analysis of the ore: Cu, 1.3 per cent.; SiO_2, 60.7 per cent.; Fe, 10.5 per cent.; CaO, 1.2 per cent.; Al_2O_3, 12.1 per cent.; S, 9.9 per cent.

Leaching operations were commenced during April, 1917, with the use of mine water, which was later replaced by waste liquors from the scrap-iron precipitation plant. The solutions were measured daily going to and from the heap, the latter being passed through a series of scrap-iron launders for the removal of the copper. Solutions depleted of copper but enriched with iron were returned to the heap for washing purposes.

This heap was treated for three years with various resting and leaching periods, at the end of which time further work had to be discontinued on account of stripping operations by steam shovels on Sacramento Hill. During the three years, a total extraction of 45.2 per cent. of the copper was effected, figured from measurements and assays of solutions both on and off the heap.

The heap was systematically sampled with a series of drill holes and pits, as well as a steam shovel cut through one end. The general average assay was 0.36 per cent. Cu, showing the actual extraction of copper to have been 72.3 per cent.

This large difference between calculated and actual results was doubtless due to unaccounted for losses of liquor, which drained directly through into the ground, and emphasizes the necessity for proper preparation of the site for commercial work.

Large-scale Installation for Treating Low-grade Ores from Sacramento Hill

The material to be mined from Sacramento Hill is divided into four classes, as follows:

1. Waste.........	0.0 to 0.5 per cent. copper
2. Low grade.....	0.5 to 1.0 per cent. copper
3. Concentrating..	1.0 to 3.0 per cent. copper
4. Smelting......	3.0 per cent. copper and over

The second grade is classified as leaching ore. It is estimated that the deposit contains 8,500,000 tons of this material, averaging 0.72 per cent. copper.

During March, 1920, the first of the leaching ore from Sacramento Hill was placed on the leaching site, this ore being encountered during stripping operations with steam shovels. From that time until October, 1921, when operations at the Hill were suspended, approximately 380,000 tons have been mined and placed ready for leaching. The proposed general layout of the first unit of the plant and heaps is shown in Fig. 1.

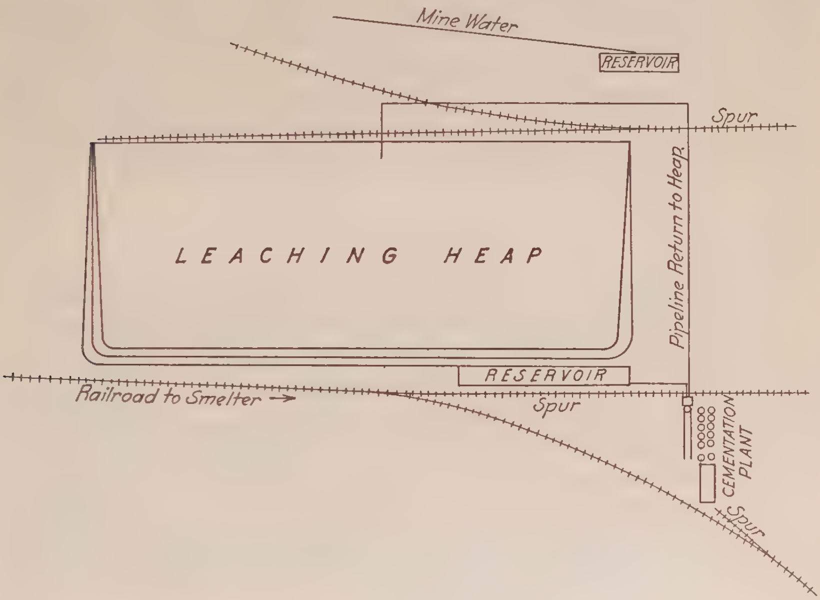

FIG. 1.

The place selected for the leaching heaps was chosen because of its proximity to the ore, the contour of the ground, and the nature of the floor. The site selected for the first 2,000,000 tons of ore is approximately 1800 by 750 ft. The average slope of the ground is between 3½ to 4 per cent. thus allowing easy drainage of solutions from the heap. The floor consists of caliche and conglomerates.

The ore mined by the steam shovels was loaded into steel dump cars, built by the Western Wheeled Scraper Co., having capacities of 20 and 25 cu. yd., which dump either side of the track. These cars are dumped by compressed air.

Before operations were commenced, a track was built along the north end of the leaching site, and dumping operations were started on the ground sloping from this side to the south. Each trainload, consisting of six cars, was sampled at the heap after dumping. As the dump increased, the track was moved to the edge, the coarser material rolling to the foot of the heap. The large lumps were broken, with powder, to from 8 to 12 in., and some of these were used for building the culverts or drains. Fine ore was kept on the surface of the heap as required for a covering and for building the basins necessary for irrigation purposes.

The culverts were built ahead of the ore being dumped, the hardest and most suitable rock being used for the purpose. These culverts are used both as ventilating flues and as drains for the solutions. The rock must be packed tightly in place to resist the force of the lumps rolling down the face of the heap, otherwise the culvert would collapse and its purpose be defeated. While the cross culverts are continuous across the site, those connecting at right angles are staggered, so that the system is an interlaced network of flues. The culverts are not always built in regular order, for advantage is taken of all depressions or drainage channels on the ground surface. The opening in the culvert is 12 in. in the clear.

Preparation of Site

Before the ore was piled, the site was cleared of cactus, brush, etc. Waterproofing the surface of the ground under the heaps is important, for it was clearly shown, by sampling the Bisbee test heap, that the actual percentage of extraction calculated from an average analysis of the washed ore was much greater than that calculated from the records of quantities and analyses of solutions during operation. Assuming that the latter were correct, the only explanation for any great difference would be in loss of solution by seepage into the ground under the piles.

The question of a suitable and sufficiently cheap waterproofing method for the site of the large plant has not been settled satisfactorily. As long as it is kept wet, a layer of clay or of slime tailings will probably be sufficiently waterproof. As soon as slime tailings from the new mill are available, they will probably be used for additional ore heaps. This will be the cheapest available waterproofing method, as it should cost but little to flume these slime tailings from the mill to the site; and the cost of waterproofing per ton of ore piled under these conditions should be nearly negligible. At present, no slime tailings or clay are available at a possible cost.

Another possible method of waterproofing is by spraying the surface with oil. If this is done, the surface should first be thoroughly dry, and it should be dried between successive coats of oil, and thoroughly dried before the ore is laid down. While an oil-coated surface will be fairly waterproof, the culverts, etc., must be built upon it, which would cause it to be badly broken, unless it was thoroughly dried beforehand.

A good part of the ground under the heaps will consist of caliche, which should have formed upon it a crust of calcium sulfate from the leaching solutions and thus arrest the percolation of solutions into the ground, but it does not seem probable that too much reliance can be placed on the water-repelling character of such a layer, and a positive method of waterproofing should be adopted if possible.

Plant

There will be two reservoirs—one above and the other below the leaching heap. The reservoir above will be used for settling out suspended matter from the mine water previous to irrigating the heaps; it will also act as a storage in case of need. The one below will be used as a storage for the liquors coming from the heaps before going to the cementation plant for the recovery of copper. These reservoirs are not yet constructed.

In Fig. 2 is shown the proposed general arrangement of the first unit of the cementation plant, which has yet to be erected. This plant is designed so that practically all of the operations will be performed automatically. There will be twelve redwood tanks, each 24 ft. in diameter by 10 ft. high, placed in two rows of six in series. They will contain false bottoms for supporting the scrap iron. Underneath the false bottom will be an acid-proof stirring arm worked from a shaft in the center of the tank and suspended from the top of the tank. This will be used to agitate the solutions when necessary and to move the precipitated copper to the central discharge on the bottom of the tank, which will be partly conical to assist in this operation.

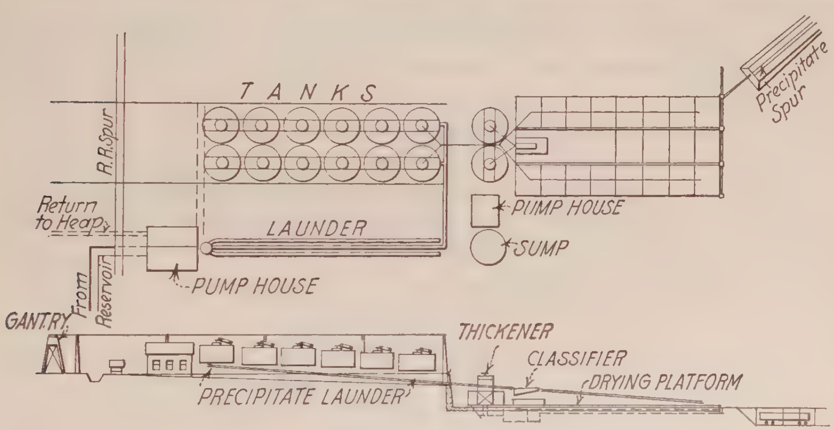

FIG. 2.

The liquor entering the tank will be introduced alongside the agitator shaft and delivered underneath the false bottom, flowing up through the iron, where it will discharge over a peripheral launder to the next tank in series. Each tank in the unit will be connected to the next tank in parallel by a launder so that any tank may be cut out for inspection without interrupting operations.

The iron will be distributed to the various tanks by a gantry crane, which will take the scrap from the railroad cars at the head of the plant or from storage.

The liquor, after being depleted of its copper, will flow to an equalizing tank, where the required amount will be returned to the heaps for washing purposes. Provision is made for the installation of a scrap-iron launder plant should it be found necessary to remove the last traces of copper from the solution.

The classifier will be of the Dorr type, built of acid-proof material, as will also the thickening tanks of which there will be two, 24 ft. in diameter by 10 ft. high. These will be equipped with acid-proof diaphragm pumps, which will remove the thickened cement copper to the drying floors, the clear overflow being pumped back into the cementation-tank circuit.

The drying floors will be built of concrete and so arranged that all surplus water can be drained to a sump and either returned to the system or go to waste.

Flow Sheet

Subject to variation from data gained from subsequent operation, the flow sheet and method of plant operation that will be followed are as follows; Fig. 3 shows the present plan.

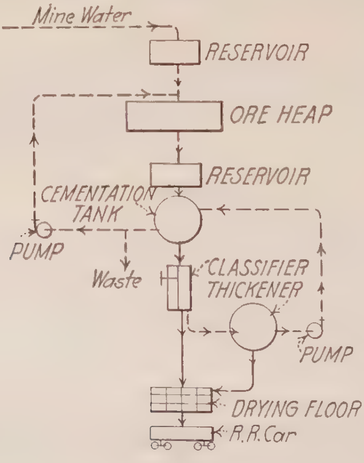

Fig. 3.

The mine water, after leaving the reservoir, will be measured, sampled, and passed on to the leaching heaps, where it will be enriched and flow to the reservoir at the foot of the heaps. From there it will be measured, sampled, and go to the head of the cementation plant, passing through two rows of six tanks in series containing scrap iron. The solution, now depleted of its copper, will flow to a sump, where the required quantity will be returned to the heap, the remainder being sampled and run to waste. The precipitated copper will be removed, at intervals, from the cementation tanks and will pass through a classifier where the coarse copper will be removed and deposited on to drying floors. The fine or suspended copper will go to thickening tanks, from which the clear solution will be returned to the cementation tanks, while the thickened product will be pumped to and deposited on the drying floors. When sufficiently dry, the product will be shoveled into mine cars and dumped into railroad cars for shipment to the smelter.

As the operations in the heaps at Bisbee and Tyrone were experimental, the procedure there may not be closely followed as experience is gained in the larger heaps.

The first of the ore was laid down before the cementation plant was available. If a part of this was installed, the logical method would be to start wetting the heaps as soon as possible after they were laid down, applying only enough water to moisten the ore thoroughly, and then allow them to wait until ready to start operation. If the first part of the ore was wet in this way, a considerable part of the copper would be made soluble by the time the heap was finished.

Without sufficient precipitating capacity first, however, most of this soluble copper would be lost when the heaps were wet during the rainy season.

The following assumptions have been made in the design and flow sheet:

In normal times, a delivery to the leaching heaps of about 400,000 tons yearly up to the total quantity of ore graded as leaching material. The cementation plant to be increased by units as required. Grade of ore 0.72 per cent. Cu. Period of extraction, 6 years to make a total recovery of 70 per cent. divided as follows:

	Per Cent.
First year	20
Second year	12
Third year	10
Fourth year	10
Fifth year	9
Sixth year	9

The period of extraction is an arbitrary assumption and the possible profit will be reduced in proportion to extension of time of treatment.

While all assumptions are believed to be conservative, the operation should be considered as a large scale experiment, the results of which will be of interest in view of the bearing it may have on the future treatment of stripping ore and other low-grade material, which in some cases may mean for some mines the addition of considerable tonnage to what is now estimated as ore.

The main experimental factor is the required time of treatment, and in operation the adoption of a method of treatment of site which will absolutely prevent solution losses.

DISCUSSION

J. Parke Channing, New York, N. Y.—When the Tennessee Copper Co. started smelting in the Ducktown district the ore was first heap-roasted. These heaps or piles were all protected by sheds which were kept in good condition and I feel quite sure that none of the copper was ever lost by leaching. When pyrite smelting was started and the roasting of ore abandoned, the roast yards were carefully cleaned up and B. B. Gottsberger made a final calculation of the total amount of copper that went into the roast yards and the total amount that was taken out. My recollection is that there was an unaccountable loss of 2 lb. of copper per ton of ore. Inasmuch as we were absolutely sure that this was not leached by rain water, the only explanation that I could give was that this copper was volatilized even at the low temperature of the roast heaps, but we have no scientific evidence to this effect.

R. C Canby, Wallingford, Conn.—The thought that impressed me while reading this paper is the statement that it is not the washing action of the solutions but the capillary action of the solutions to and from the inside of the ore that produces the results. So these periods of oxidation might perhaps more properly be called periods of capillary action.

C. S. Witherell, New York, N. Y.—I think capillary attraction also performs another role; not only does it serve to make the leaching solution penetrate the lumps of ore through numerous small cracks, but when the salts crystallize, an expansion takes place, thus disintegrating the lumps and opening other passages for the leaching solution.

Edward L. Blossom, New York, N. Y. (written discussion).—For successful heap leaching there are two outstanding requirements: (1) Every step of the operation must be carried on at low cost. (2) The copper, if not already in water- or acid-soluble form, must be capable of oxidizing to that form in a reasonable period of time. The authors of this paper seem to have fixed 6 years as the maximum but hope to make the time shorter.

Carbonates and oxides are the most readily leachable copper minerals. Of the sulfides, chalcocite and bornite are the easiest dissolved. Chalcopyrite is more refractory. The material should be broken (cost permitting) sufficiently to expose all copper mineral to contact with the solution, but as few fines as possible should be made.

Water, with or without sulfuric acid, and with or without mine waters carrying iron salts, is the solvent employed. Access of moisture and oxygen to interior of heaps is indispensable for producing the desired results. The authors stress the importance of alternately wetting and drying each piece of ore in the heap, by this means they depend on reversed capillarity to bring the dissolved copper salts from the interior of each piece to the surface whence it can be removed by the next wash. This is the same physical factor as that which brings desert alkalis to the surface of the soil, and its recognition as an important factor in leaching is highly creditable to the experimenters. So far as I know, this factor has not previously been mentioned in the literature on the subject.

The authors recommend waterproofing the ground on which the heaps are to be spread and present figures showing that losses up to 37 per cent. of the extracted copper have resulted from neglect of this precaution. The authors' propose to precipitate their copper from solution in tanks resembling Dorr thickeners, each tank having a false bottom above the stirring arm to carry the scrap iron. This also is a novelty. Scrap-iron launders of the old type will be used, if at all, only for removing the last traces of copper from the solution. A portion of the depleted liquor will be returned to the heaps for washing purposes.

The chemistry of the process—how insoluble copper compounds are converted into soluble salts—deserves careful attention. The authors give a number of reactions which probably contribute to the desired end, but of the relative importance of these reactions little is known with certainty. Some of the copper sulfide is probably converted into sulfate by direct oxidation. More of it is certainly dissolved by the action of ferric sulfate, which also attacks any metallics which may be present. Oxides, carbonates, and silicates of copper are more or less completely dissolved by free acid. Ferrous and ferric sulfates, together with free sulfuric acid, are continuously produced in the heaps by oxidation of iron sulfide minerals. These reagents, freshly generated in close proximity to the copper mineral, doubtless play a major role in the extraction. But the same reagents may be introduced with the wash liquors. The efficacy of sulfuric acid is presumably independent of its source, but for some reason the iron salts introduced when mine water or depleted liquor from the precipitation tanks are led onto the heaps do not produce any definitely ascertainable increase in copper extracted. This, I have been told, is also the experience at Rio Tinto. Yet the authors state that in laboratory work the presence of iron salts was shown to be an essential factor and anyone who has observed the powerful solvent effect of ferric sulfate on the minerals in question cannot but be surprised that addition of this reagent in heap leaching should not improve the results.

For this apparent conflict between theory and practice two explanations suggest themselves:

1. Mine water and tank liquors, having been in contact with reducing agents, contain a relatively small proportion of ferric, as compared with ferrous, sulfate. The latter does not become an active solvent unless it is oxidized in the heaps.

2. Mine water and tank liquors (in contrast with freshly made laboratory solutions) usually contain basic iron compounds ready to fall out of solution when the acidity and the concentration of ferric sulfate are reduced. Both are reduced by contact of the solution with reactive copper minerals, and in consequence a film of iron compounds is likely to be precipitated just where it will do most harm, viz., on the surface of partly dissolved particles of copper ore.

Once precipitated, this film is difficult to remove even if the subsequent washes contain free acid, and in consequence the needful contact between mineral and solvent is lessened or destroyed. Oftentimes mere dilution of a neutral iron-bearing solution will bring down the precipitate. Cases are on record where application of mine waters has so cemented a heap as to ruin it for leaching purposes.

The remedy, if there be a remedy, is not easy. A very considerable addition of acid to the wash waters would be required to insure maintenance of acidity in all portions of the heap. In presence of acid-soluble minerals other than copper the cost of this would be altogether prohibitive.

Were it possible to oxidize all the iron in the wash water before applying it to the heaps, interesting results might follow, but too much experimental work has been done on these lines to leave much hope that such oxidation can be economically accomplished.

We are thus brought back to the conclusion that improvements in heap leaching must lie in the increase of oxidation effects within the heap. Mention is made of the catalytic effect of the relatively enormous surface of the ore in converting SO_2 to SO_3. May not some cheap accelerator of catalytic action be found that can be mixed with the ore?

Relative Efficiency of Amalgamation and Cyaniding

By Allan J. Clark,* E. M., and W. J. Sharwood,† Ph. D., Lead, So. Dak.

(New York Meeting, February, 1923)

When the cyanide process came into general use, late in the nineteenth century, chlorination was quickly supplanted, but amalgamation yielded place more slowly, being still the major process at many plants, where it is followed by cyanidation of the tailings.

The research work of the Denny brothers in South Africa[1] was probably the most important contribution of its time to the metallurgy of gold. By grading analyses of mill products, they pointed to the necessity of fine grinding to accomplish maximum extraction by cyanide. The work here described is merely an extension of the Dennys' method.

Developing the fine-grinding principle, the steps of "crushing in solution" and of "all-sliming" were logical sequels, though both had been attempted early in the history of the cyanide process. Crushing in solution introduced certain complications in amalgamation; when "all-sliming" was adopted it was observed that the omission of amalgamation did not materially affect the total recovery. Thus the field of the older process has become much restricted, being at present confined to ores which, yielding freely to amalgamation, either are too low in grade to show an economic advantage in "all-sliming," or present chemical or other difficulties making the use of cyanide solution unadvisable during the crushing and preparation of the ore. South African ores are typical of the former class; Homestake ores are of both classes, but even in these instances the amalgamation work is being curtailed and greater burdens thrown on the cyanide plants.

It is obvious that the time required for the dissolution of gold depends on the size of its particles. In an effort to determine the critical size, the following laboratory method was evolved. A standard amalgamation test[2] had already been established, which involves the agitation of about 100 gm. of ore with 20 gm. of pure mercury, the latter then being separated and its gold contents determined. This method was applied directly to certain mill products with interesting results, a few of which may be noted here. The column "Free Gold" gives the value per ton thus amalgamated, as against the "Total Gold" determined by fire assay.

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* Metallurgist, Homestake Mining Co. † Chemist, same company.

[1] G. A. Denny and H. S. Denny: Rand Metallurgical Practice and Recent Innovations. *Jnl.* So. African Assoc. of Engineers (June, 1906) **11**, 239 to 351.

[2] W. J. Sharwood: *Trans.* (1915) **52**, 162.

TABLE 1.—*Comparative Tests of Amalgamation and Cyanidation on Three Lots of Amalgamation Tailings*

Screen sizes for original material	SERIES I.—Screen opening, 0.031 × 0.227 in. Water ratio, 8.67. Tons per stamp-day, 7.00.	SERIES II.—Screen 12-mesh, square opening. Life, 10 days. Water ratio, 8.42. Tons per stamp-day, 7.96.	SERIES III.—Screen 10-mesh, square opening. Life, 31 days. Water ratio, 9.00. Tons per stamp-day, 8.0.
Over 50-mesh	Weight, 28 %. Crushed to 80-mesh, yielding: +100, 10 %; +200, 31 %; −200, 59 %; together containing: Total gold, $3.11; free gold, $2.52. Comparative tests on the above material: By direct cyaniding: Solution, $2.30 Extr., 81 % Residue: Total gold, $0.53 Free gold, $0.27 By amalgamation on 12-ft. plate: Tails: Total gold, $2.32 Free gold, $1.20 Cyaniding the above tails: Solution, $2.11 Extr., 89 % Residue: Total gold, $0.25 Free gold, $0.06	Weight, 29 %. Crushed to 80-mesh, yielding: +200, 65 % Total gold, $4.06 Free gold, $3.04 Amalgamating plate gave tails ctg.: Total gold, $2.99 Free gold, $1.72 Cyaniding the above tails gave: Solution, $2.87 Extr., 86 % Residue: Total gold, $0.44 Free gold, trace −200, 35 % Total gold, $1.36 Free gold, $0.69 Amalgamating plate gave tails ctg.: Total gold, $1.29 Free gold, $0.42 Cyaniding the above tails gave: Solution, $1.24 Extr., 92.5 % Residue: Total gold, $0.10 Free gold, trace	Weight, 20 %. Crushed to 80-mesh, yielding: +100, 27 %; +200, 23 %; −200, 50 %; together containing: Total gold, $3.24; free gold, $1.83. Amalgamating plate gave tails ctg.: Total gold, $2.00 Free gold, $0.70
50- to 80-mesh	Weight, 10.3 %. Not tested.	Weight, 9.2 %. Crushed to 80-mesh, yielding: +100, 53 %; +200, 33 %; −200, 14 %; together containing: Total gold, $3.29; free gold, not det. Amalgamating plate gave tails ctg.: Total gold, $2.48 Free gold, $1.32 Cyaniding the above tails gave: Solution, $2.13; extr., 76 % Residue: Total gold, $0.67 Free gold, $0.18	Weight, 13 %. Crushed to 80-mesh, yielding: +100, 30 %; +200, 30 %; −200, 40 %; together containing: Total gold, $3.88; free gold, $2.53. Amalgamating plate gave tails ctg.: Total gold, $2.87; free gold, $1.75. These tails, subdivided, gave: +200, 60 % Cyaniding gave: Solution, $3.31 Extr., 86 % Residue: Total gold, $0.53 Free gold, $0.03 −200, 40 % Cyaniding gave: Solution, $1.14 Extr., 98 % Residue: Total gold, $0.10 Free gold, trace
80- to 100-mesh	Weight, 4.4 %. Not tested.	Weight, 7.8 %. Total gold, $3.00; free gold, $0.63. Amalgamating plate gave tails ctg.: Total gold, $2.42; free gold, $0.06. Cyaniding the above tails gave: Solution, $1.83; extr., 75.5 %. Residue: Total gold, $0.60 Free gold, $0.03	Weight, 10.5 %. Total gold, $3.06; free gold, $0.36. Amalgamating plate gave tails ctg.: Total gold, $2.83; free gold, $0.15. Cyaniding the above tails gave: Solution, $2.52; extr., 86 %. Residue: Total gold, $0.48 Free gold, trace

TABLE 1.—*Comparative Tests of Amalgamation and Cyanidation on Three Lots of Amalgamation Tailings—(Continued)*

Screen sizes for original material	SERIES I.—Screen opening, 0.031 × 0.227 in. Water ratio, 8.67. Tons per stamp-day, 7.00.	SERIES II.—Screen 12-mesh, square opening. Life, 10 days Water ratio, 8.42. Tons per stamp-day, 7.96.	SERIES III.—Screen 10-mesh, square opening. Life, 31 days. Water ratio, 9.00. Tons per stamp-day, 8.0.
100- to 200-mesh	Weight, 12.8 % } Total gold, $2.10; free gold, $0.72	Weight, 8%. Not tested.	Weight, 12%. Not tested.
Below 200-mesh: Sand	Weight, 6.8 % Cyanidation gave: Solution, $1.79; extr., 88 %. Residue: Total gold, $0.25 Free gold, trace	Weight, 9%. Not tested.	Weight, 44.5 %. Not tested.
Below 200-mesh: Slime	Weight, 37.7 %. Total gold, $1.74; free gold, $0.30. Cyanidation gave: Solution, $1.44; extr., 86 %. Residue: Total gold, $0.24 Free gold, trace	Weight, 37%. Not tested.	

TABLE 2.—*Amalgamable Gold in Certain Mill Products*

	TOTAL GOLD PER TON	FREE GOLD PER TON
Homestake (220-stamp) mill. Tailing (after passing 3 rows of plates, which caught about 70 per cent. of the original gold)	$1.91	$0.35
Golden Star (200-stamp) mill. Tailing (after 4 rows of plates):		
Sandy portion	1.77	0.24
Slime	1.36	0.30
Regrinding plant:		
Coarse sand to tube-mills	3.98	0.50
Discharge of tube-mills, before going to plates	3.20	1.41
Tailing after passing plates	2.60	0.53

It remained to establish a standard laboratory method for cyanide testing, and to check the amalgamation test for possible variations due to the drying of samples before testing, presence of lime, traces of cyanide, etc. In former laboratory tests we had sought to follow fairly closely the schedules in actual use at the cyanide plants. It had, however, been observed that with Homestake ore the laboratory results were rarely in close accord with those obtained in the large way, often showing extractions as much as 10 per cent. lower than the yield in actual practice. The cause was probably the difficulty of reproducing, in a small test, the thorough aeration secured in the leaching vats.

The routine adopted for the cyanide test of the sandy material from this ore involves two or three distinct periods of leaching in glass percolators, intermediate drying of the charge with exposure to air, and finally washing with a small proportion of water. Slime is agitated in bottles and the residue filtered and washed. The complete method would then consist in examining any ore or tailing for free gold, either as a whole or by size-fractions, then subjecting portions to cyanide treatment, re-grading the cyanide residue, if desired, and examining the residue or its fractions for free gold. The coarser portions may be ground to any desired fineness, and the product passed over an amalgamated silver-plated copper plate if it is desired to remove the easily amalgamable gold, before treating with cyanide solution.

The ore selected for a series of tests, some of which are summarized below, was a dense, unoxidized ore from the Homestake mine. It was crushed in 5-stamp mortars of the regular pattern, but through coarser screens (of steel wire) than the former Homestake standard; mercury was added at the mortar. The customary inside copper amalgamating plate was omitted from this mortar, and the crushed ore discharged from the screen, which was raised by a 3½-in. block, passed over an amalgamated plate of silver-plated copper, 4½ ft. wide by 12 ft. long, and set at a slope of 8 per cent. At the foot of this plate the samples were taken. The treatment and results are sufficiently indicated in Table 1, in which all the gold values are stated in terms of the ton of material treated.

Soluble Silica in the Preparation of Zinc-sulfate Solution for Electrolysis

By Jesse O. Betterton, Omaha, Neb.

(New York Meeting, February, 1923)

Recently some experimental work was conducted by the author in connection with the direct leaching of certain zinc ores with sulfuric acid with the object of subsequently recovering the zinc by electrolysis. The ores contained relatively large quantities of soluble silica, making it necessary to evolve special methods for handling the silicate solutions resulting from a leach of the ore, and also for the final removal of the silica before electrolysis. The problem was complicated by the presence of vanadium in the ore, the commercial recovery of which was very desirable. This paper, however, deals only with the silica phase of the problem.

The zinc in the ore occurs in the form of a silicate (hemimorphite) and, to a very small extent, as the phosphate (hopeite) and carbonate (calamine); the vanadium is found as descloizite and vanadinite but mainly as descloizite.

The experimental work was done in a plant producing 50 lb. of zinc daily, in which the complete cycle of operations, including the final electrodeposition of the zinc, was performed continuously for one year. Some of the conclusions given here are the result of the operations in this plant; the others are from laboratory tests inaugurated both as a guide to the plant operations and, in certain cases especially, to determine the limits in question.

The practice in the 50-lb. zinc plant was to direct leach the ore crushed to pass 40 mesh, with spent electrolyte from the cells containing approximately 3 per cent. zinc and 10 per cent. free acid. This resulted in an acid leach solution and an ore residue. The acid leach solution was subjected to a special treatment for subsequent removal of vanadium; then the silica and vanadium in solution were precipitated together and removed as a "silica residue" by pressing through an ordinary plate and frame press. The next step was to treat the filtrate from the silica residue with zinc dust as a final purification step before it went to the cells for electrolysis. There were thus the following products: Ore, ore residue, acid leach solution, silica residue, silica filtrate, zinc dust purification residue, and final solution. The analyses of such products as concern the silica phase of the operations are given in Table 1.

TABLE 1.—*Representative Ore Leach*

	Ore	Ore Residue	Acid Leach Solution	Silica Residue	Zinc Dust Purification Residue
SiO_2	15.43	7.08	1.30	49.84	10.60
PbO	4.00	10.00		3.61	1.20
Fe_2O_3	28.24	63.19		1.39	11.10
FeO	0.22	2.84	Sp. gr.		1.43
Al_2O_3	2.64	2.86	1.22	2.84	5.74
Mn_2O_3	1.76	Tr.		6.16	10.10
CaO	0.66	0.44		0.45	3.04
MgO	0.76	0.23		0.23	4.82
ZnO	42.31	4.76	8.47	12.96	28.95
V_2O_5	1.25	0.32	0.14	4.19	1.04
P_2O_5	1.14	0.05		4.12	0.22
SO_3	0.22	5.19		1.86	18.26
Organic matter			H_2SO_4		
$CO_2.H_2O$	1.43	3.78	0.50	12.09	3.75
Total	100.06	100.72		99.84	100.25

DIFFICULTIES CAUSED BY GELLING OF ORE LEACH

In the early stages of the work, considerable trouble was experienced in the ore leach; this was later found to be caused by the more or less gelled condition of the slightly acid silicate solution. This part of the process was a decantation practice and sometimes the iron slimes in the ore residue would settle perfectly and at others they would settle hardly at all, with various degrees of settling between these extremes. Occasionally, a solution would visibly gel before it had sufficient time to be decanted off the ore residue, which was usually 18 hr. from the time of starting a leach.

It was early noted that if all new acid were used for a leach, these troubles did not appear or appeared only in a mild form. On the other hand, it was found to be almost impossible to conduct a leach entirely by the use of spent electrolyte, although the free acid was the same in both cases. Another significant point was that the foregoing difficulties were roughly in inverse proportion to the rate of filtration of the solution while separating the ore residue on an ordinary filter paper. Whenever a leach sample refused to filter, or started to slow up badly, toward the end of a leach, it was a sign that the iron slimes would not settle and perhaps that the solution would completely gel.

In studying the causes of the preceding difficulties it was found that but a small part of the non-settlement of the iron slimes was caused by a colloidal condition; sometimes the iron slimes would settle enough that the upper part of the solution was almost entirely free from solids, yet the clear solution would refuse to filter. It was therefore evident that a change in the viscosity of the solution had occurred; it was afterwards found that such was the case. Perfectly clear solutions will, in time, lose their filtering qualities and finally refuse to filter before there is any visible change in the appearance. A more scientific way of explaining this is to say that in the transition from the hydrosol to the hydrogel condition, filtering and settling cease with the incipient formation of the hydrogel.

In addition to the necessity of finding the causes of the phenomenon, there was the question as to whether any relief from such troubles was to be expected. If the gel started to form at once in very minute amounts and uniformly increased as time elasped, it was likely that such solutions would refuse to filter and to allow reasonable settling at so early a period that all attempts to get a workable method would be useless. Fortunately, it has been definitely proved that the change is not gradual, but occurs rather abruptly after a lapse of time, which can be quite accurately predicted, provided the factors governing the phenomenon are understood quantitatively. Also, these factors have been rather extensively determined, as far as practical work demands; the time element is more than sufficient for ordinary purification operations and solutions of ample concentration in zinc can be readily handled.

The factors thought to cause gelling that were investigated were:

1. *Temperature.*—Below 45° C., the maximum temperature resulting from the action of the acid on the ore during a leach, there was no effect; temperatures as high as 90° C. resulted in no action, within several hours, at least.

2. *Carbon Dioxide.*—Both high-grade zinc-carbonate ore, containing some soluble silica, and mixtures composed half and half of carbonate and silicate of zinc were tried and no gelling tendencies were noted.

3. *Air.*—In order to determine whether the Pachuca tank could be used for the leach, a small test apparatus was rigged up. Several leaches were duplicated in respect to all conditions on the small scale and compared to simultaneous leaches with the mechanical agitator ordinarily used. No difference could be detected in the behavior of the resulting solutions, hence the conclusion that the use of compressed air under ordinary operating conditions is without effect.

4. *Oxidizing Compounds in the Spent Electrolyte (Permanganates, Manganates and Chlorates).*—No differences due to this cause could be detected. In making the comparisons, samples of ore were leached

directly with spent electrolyte, and again with the same spent electrolyte after its oxidizing power had been balanced with ferrous sulfate.

5. *Zinc Concentration or the Effect of a Strong Electrolyte.*—The effect of this was most marked. By keeping the acidity and silica constant in a series of tests and varying the zinc, it can definitely be demonstrated that an extremely regular relation between the zinc concentration and time of gelling exists—the higher the zinc the sooner the gelling. An inspection of Figs. 1 and 2 will show that, given the acidity, zinc, and silica contents of a solution, its time of incipient gelling can be quite accurately predicted.

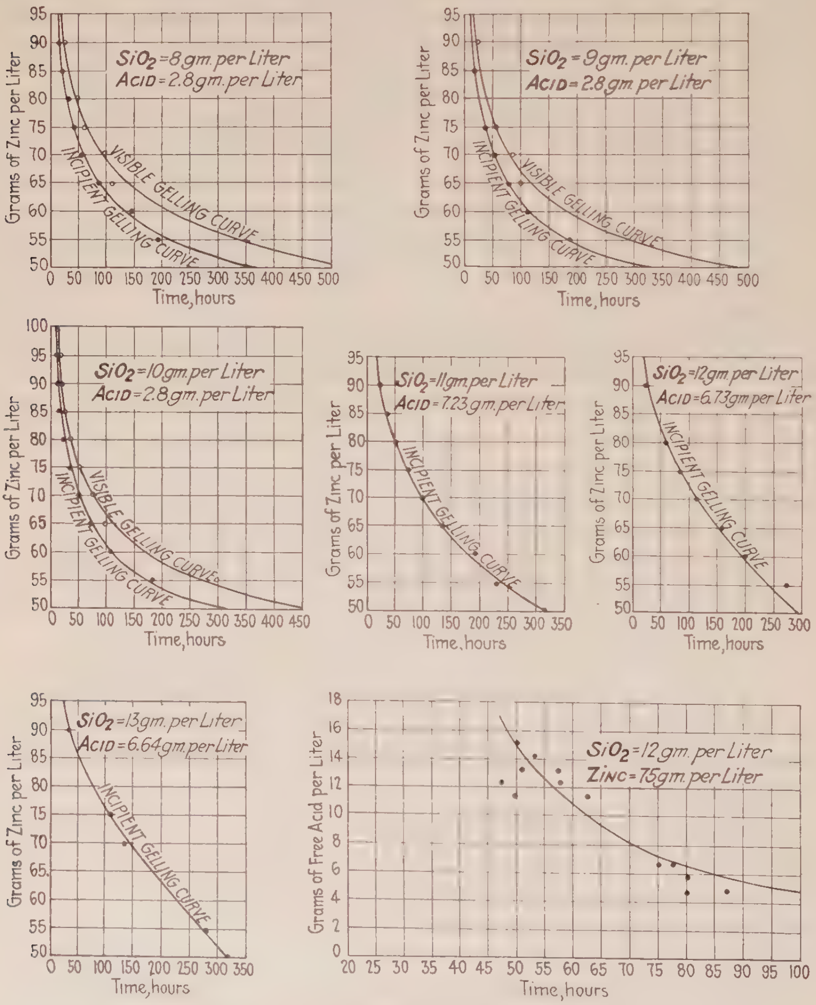

FIG. 1.

6. *Acidity.*—By exactly similar methods, it was shown that the higher the free acid the quicker the gelling.

7. *Silica Concentration.*—Following the same procedure of varying but one of the three variables at a time, it was shown that the time interval preceding incipient gelling is shortened as the silica is increased from 8 to 10 gm. silica per liter. The curve then reverses and the time interval rapidly increases as the silica increases from 10 to 13 gm. per liter. This is an important consideration in connection with practical operations and has been confirmed in a large measure by subsequent operations of the 50-lb. plant. Here solutions containing as high as 100 gm. zinc, 5 gm. free acid, and 20 gm. silica per liter have been successfully handled.

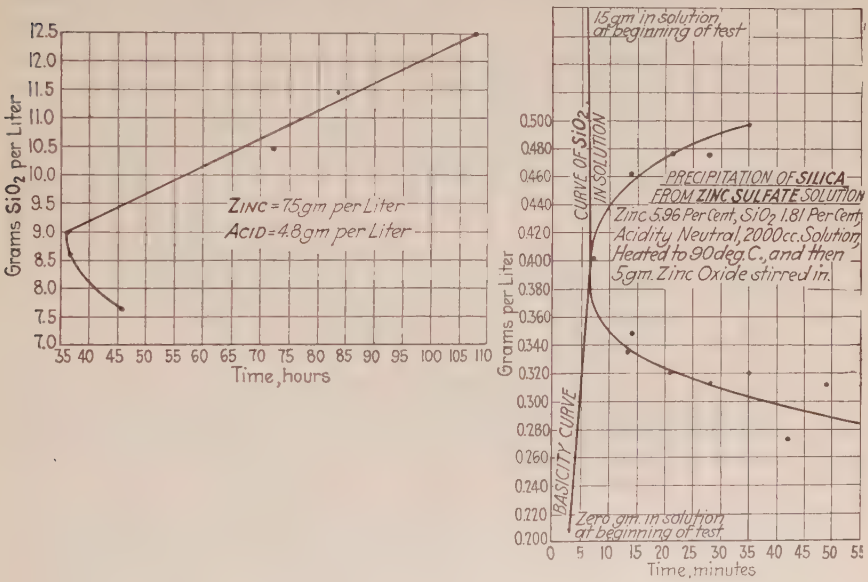

FIG. 2.

In some of the accompanying curves, two sets of points were determined. First, the times at which the clear solutions refused to filter, termed the "incipient gelling curve;" second, the times at which well-defined visible gelling occurred, termed the "visible gelling curve."

The visible gelling curve has no real significance and is given to illustrate the abruptness of the change ordinarily taking place in a solution. For instance, solutions will filter readily for many hours, even days in some cases, and then abruptly lose this characteristic; very soon thereafter, depending on their zinc and silica concentrations, gelling becomes visible and progresses until the whole mass is practically a solid. The visible gelling curve does not depict the times of first visible gelling; rather it shows a further stage in the process about which there can be no doubt whatever. Even this, in most cases, is near enough to the time the solution stopped filtering to illustrate the abruptness of the change that occurred in the solution when it ceased to filter.

The tests for making the curves shown were conducted at ordinary room temperatures, 15° to 20° C., and were done with exactly 200 c.c. of solution in conical-shaped beakers covered with watch glasses. At regular intervals, chosen to determine the time accurately and yet make as few tests as possible, 10 c.c. of solution were removed by pipette and placed upon a filter discharging back into the solution being treated. As long as this portion went through the paper positively and entirely, the solution was considered filterable. At the time incipient gelling started, there could be no doubt about an abrupt change having taken place in the viscosity of the solution. The filtering time, previously fairly uniform, was greatly increased, and soon thereafter a 10-c.c. sample would refuse to filter, usually less than half of it going slowly through the paper and the rest remaining on the paper as a visible gel. The solution was considered non-filterable at the time such a 10-c.c. sample was withdrawn from the test. Time intervals thus obtained are accurate to ½ to 3 hr., the error increasing as the zinc concentration lowers.

Removal of Silica from Solution

A method for removing the silica from solution in filterable and washable form, or the so-called granular condition, was developed much in advance of a complete understanding of the factors governing gelling in the acid solutions.

In the early work, 15- to 30-lb. ore batches were leached and the resulting faintly acid solutions were treated with milk of lime and heated to boiling for the removal of the silica. Large quantities of lime, in one case 6 lb. of lime to 17 lb. of ore, were required and it was suggested that a lime silicate was the product formed. However, as improved methods of adding the lime, such as adding very finely crushed air-slacked lime very gradually over the entire surface of the bath greatly reduced the quantity required, it became evident that the sole result obtained by this lime was to render the solution alkaline. Because of the rapid sulfating, the lime formed lumps with a protective coating of calcium sulfate; this necessitated the use of excessive quantities of lime to produce the required alkalinity.

As lime water does not precipitate zinc hydroxide from a zinc-sulfate solution, unless a large excess is added, and also as it does not precipitate calcium sulfate, because that salt is more soluble than the oxide of lime, it was decided to treat a batch of solution with it and thus determine if alkalinity is the true cause of the precipitation of the silica. The results of the tests were completely successful and were it not for the increase of volume, in these tests 100 to 200 per cent. increase, the method would be suitable for practical operations.

With the principle once established, zinc-oxide fume was used to bring about the desired alkalinity. The use of such agents as commercial whiting, finely ground limestone, and lime, for the removal of small amounts of silica from solution, depend on their secondary reactions with hydrolyzed zinc sulfate. These reactions produce zinc hydroxide, whereby the hydroxyl concentration is increased by virtue of the considerable solubility of zinc oxide in hot zinc-sulfate solution. The lime enters into the reaction to the extent of the volume of solution that can be considered saturated with calcium oxide at a boiling temperature.

Using zinc oxide, the silica is precipitated in less than 1 hr. when heated at a temperature of 90° to 100° C., 5 lb. of zinc oxide in excess being added to 2 tons of solution containing 1 per cent. silica and 5 to 6 per cent. zinc. The silica is not entirely removed from solution under these conditions, 1 or 2 per cent. of the total silica comes down later in the zinc-dust purification.

During the earlier stages of the reaction, the precipitated silica separates rapidly, but until precipitation is practically complete the clear supernatant solution refuses to pass through a filter paper. When the precipitation of the silica is practically complete, the solution filters readily; in fact, it was not the precipitated silica that previously prevented filtration but the silica in solution in a transition stage. A clear recognition of this fact at the time greatly assisted in solving the problems later connected with the gelling of acid solutions.

On the acid side, in the case of weak solutions, days may elapse between the time the solution stops filtering and the time a fairly solid gel has established itself; while on the alkaline side, precipitation of silica can be brought about in a few minutes. Yet the gelling of an acid solution is fairly abrupt, considering the length of time the solution has previously been filterable. Whether these transitions differ in time only and result in the same end products in both cases, is open to question.

An acid solution in the gelled state is apparently one product; it will not filter and cannot be washed. It consists of the fixation, or rather slowing down to practical fixation, of an incomplete transition, and really consists of solid particles of silica (probably hydrated) uniformly distributed throughout a mass of ungelled solution. On the other hand, the precipitate from alkaline solution settles, filters, and washes readily. Whether the acid condition of gel can be considered as an incomplete transition while the alkaline condition is a practically complete transition, and the actually precipitated silica is the same in both cases; or, whether the acid product can be considered as a hydrated acid of silicon and the alkaline product as an hydroxide of silicon, hence two essentially different products, is another interesting question. From the position of silicon in the periodic table, it could conceivably act in this way.

The silica press cake, when formed at 64 lb. pressure, has a specific gravity of 1.54 and a moisture content of 70 per cent. When repulped with weak wash water and again pressed, at 53 lb. pressure, its specific gravity averages 1.43, and its moisture content 78 per cent. The analysis of a representative residue has already been given.

Table 2 gives additional data regarding time, temperatures, alkalinity, and rate of silica removal for solutions ordinarily met in commercial zinc work.

TABLE 2.—*Precipitation of Silica from Zinc-sulfate Solutions*
(Zinc 5.96 per cent., SiO_2 1.81 per cent., acidity neutral, 2000 c.c. solution heated to 90° C., and then 5 gm. zinc oxide stirred in)

Time Interval, Minutes	Basicity, Gram ZnO per Liter	Gram SiO_2 per Liter in Solution
0	0.000	1.81 per cent. approx. 15.000
7	0.402	0.370
14	0.462	0.336
21	0.476	0.320
28	0.476	0.312
35	0.496	0.320
42		0.272
49		0.312
55		0.284

CONCLUSION

In these conclusions, all statements are to be accepted in the sense of their application to practical zinc metallurgy.

1. The conditions governing the gelling of acid silicate solutions are simple, and essentially consist of three variables, zinc, silica, and acid concentrations of the solutions.

2. The time of gelling can be predicted, conditions as to acid, zinc, and silica concentrations being known.

3. The limits for safe work are ample, both as regards concentration of zinc in solution and time necessary for operations ordinarily to be employed.

4. The silica can be readily removed in a filterable, washable form, without excessive acid losses or the introduction of elements other than zinc.

ACKNOWLEDGMENTS

Among my assistants concerned with completing these tests, I wish to acknowledge and thank Mr. G. Chad. Norris, for useful suggestions and help throughout the whole course of these experiments; Mr. W. Watson, for analytical work and obtaining data for the curves; Mr. C. L. Ross, for first pointing out the relation that existed between the specific gravity of solutions and the interval before gelling, also for data obtained from the operations of the 50-lb. plant.

DISCUSSION

William N. Watson, Broken Hill, N. W. Rhodesia, Africa (written discussion).—Being acquainted with the process described and with its operation on an experimental scale, the writer wishes to contribute some additional information on the subject. The following notes were prepared in conjunction with G. Chad. Norris, who carried on the experimental work after the departure of Mr. Betterton, and to whom I wish to acknowledge my indebtedness.

In Table 1, giving the analysis of a typical ore sample, it might have been added that the percentage of acid-soluble silica ranged from about 10 to 13 per cent. The silica content of the zinc-dust purification residue is shown as 10.6 per cent. It should be noted that this residue was always small in bulk and weight, and, from the manner of handling it must be contaminated with dust; hence all the SiO_2 shown does not enter the circuit as zinc silicate. Very frequently the SiO_2 content of these residues was as low as 0.2 per cent. and seldom higher than 2 per cent., 1.25 per cent. being about an average.

In discussing the factors thought to cause gelling of the solutions, the author introduces the question of carbon dioxide. He states that "Both high-grade zinc carbonate ores . . . and mixtures of half-and-half carbonate and silicate of zinc were tried and no gelling tendencies were noted." It is to be presumed that "no *additional* gelling tendencies were noted" is meant; that is, over straight silicate ore. This is directly opposed to Watt's Dictionary of Chemistry, on the subject of Silica, Vol. 4, p. 448, where it is stated that "The solution of SiO_2, x H_2O is coagulated by . . . also by CO_2. The acid conditions of the tests probably upset the coagulating action of the CO_2.

In testing the effect of oxidizing compounds, the author omits to note the effect, if he ascertained it, of a compound having high oxidation potential present in the ore, namely, vanadic acid. The effect of varying concentrations of vanadic acid in the solutions under test might have shown points of interest, though probably not in the region affecting practical operations.

The curves showing the effects of varying concentrations of zinc, acid, and silica are of great practical importance and value; with them it is possible to predict the time that will elapse before any particular solution will gell. They show that it is easily within the reach of works practice to handle successfully solutions of such high concentrations of dissolved silica.

Admitting as a fact the author's statment that the visible gelling curves as shown in the first three diagrams of Fig. 1 have no real significance, one point is out of alignment in each curve and that point at a concentration of 65 gm. Zn per liter in each case; was this because of an experimental error or has it some special significance?

The points in the curve dealing with the effect of increasing acid are rather irregular, but omitting three, two between 45 and 50 hr. and the lower one at 80 hr., the remainder are about equidistant on opposite sides of the curve. The results shown by this curve appear to be in contradiction to a statement in Mellor's Modern Inorganic Chemistry to the effect that "The passage of silicic acid from the sol to the gel condition is retarded by the presence of a little acid . . . ". The results shown were undoubtedly obtained, but the effect of the additional acid is in need of some explanation. In this connection, it might be noted that the hydrogen ion concentration in the curves is not wholly due to sulfuric acid, as shown, but also to phosphoric and vanadic acids, and, perhaps, partially to silicic acid itself, the mixture being estimated and reported as sulfuric. The additional acid in the curve of acid effect is, of course, all added sulfuric.

In the curve with SiO_2 10 gm. and acid 2.8 gm. per liter, the gelling point with a Zn concentration of 75 gm. per liter is shown as approximately 35 hr. Turning to curve 1 of Fig. 2, which has the same Zn concentration, we find that, with 4.8 gm. acid, the solution of 10 gm. SiO_2 is shown not to gel until about 57 hr. As this latter curve is at higher acid, it should, by the acid variation curve, gel sooner than the first solution, that is, in less than 35 hr. The following table of comparisons may make this more clear:

Comparison of Fig. 1 and Fig. 2, Curve 1

Fig.	SiO_2, Gm.	Zn, Gm.	Acid, Gm.	Time of Gel, Hr.
1	8	75	2.8	40
2	8	75	4.8	41
1	9	75	2.8	37
2	9	75	4.8	36
1	10	75	2.8	35
2	10	75	4.8	57
1	11	75	7.23	75
2	11	75	4.8	87
1	12	75	6.73	80
2	12	75	4.8	97

The effect of variation in acid concentration is borne out except in the case of the curves relating to 10 gm. SiO_2 per liter. In Fig. 2, it might be expected, as the acid is higher than in Fig. 1, that gelling would take place in a shorter time than with the corresponding solution in Fig. 1; but it is shown as having taken about 20 hr. longer. From this, it can be suggested that the "break" in the curve, where the reversal action takes place, should occur at the 10-gm. point and not at 9 gm. as shown.

In connection with the concentration of silica in the solutions, it might be noted that solutions of the following concentrations have been successfully carried:

1. Zn, 99.99 gm. per liter, SiO_2, 23.74 gm. per liter.
2. Zn, 89.92 gm. per liter, SiO_2, 25.11 gm. per liter.

Acid, approximately 5 gm. per liter in each case. Many solutions of 20, 21, and up to 23 gm. SiO_2 per liter have been handled without difficulty and successfully treated.

While appreciating that the author is insistent on his conclusions being accepted in the light of their application to practical zinc metallurgy, the requirements of which they satisfy, it must, however, be noted that, in discussing the reactions which lead to the gelling of the zinc sulfate-silicic acid solution, he states "It has been definitely proved that the change (from hydrosol to hydrogel) is not gradual but occurs rather abruptly after a lapse of time" (page 161, lines 20–21). It appears to be unnatural that the various systems, zinc sulfate, silicic acid, sulfuric acid, cited by the author should remain stable for a period, extending into days in some cases considered, and, thereafter, without any apparent modification of the system, that the equilibrium should be abruptly upset, and a transition from the hydrosol to the hydrogel condition of the silicic acid should suddenly set in.

It appears more conceivable to look on the transition as progressing continuously from the moment the system is created, the rate of change accelerating as the amount of transposed silica increases, and filterability only ceasing when the amount of hydrogel that has been formed is sufficiently great to be detected by the rather crude means available. It might be suggested that the means employed were not of sufficiently delicate a nature to detect the change until it had become extremely positive, and until, in fact, it was almost apparent to the eye. The means of detection used was the ability or inability of the solutions to filter through Whatman's No. 30 filter papers, 11 cm. diameter.

I would like to suggest that the change takes place gradually throughout. When the ore is dissolved in acid, the reaction

$$2ZnO, SiO_2, H_2O + 2H_2SO_4 = 2ZnSO_4 + H_4SiO_4 + H_2O$$

takes place, zinc orthosilicate reacting with the acid to give zinc sulfate and orthosilicic acid. The formation of the zinc sulfate is known to take place to completion in 10 min., and so it must be accepted that the free silicic is also present in this time. Most textbooks state that solutions up to 5 per cent. colloidal silica can readily be obtained by dialysis, which solutions can be concentrated by boiling to about 14 per cent., and will then stand for a few days without change (see Mellor). If these solutions can so be treated, the idea of abrupt change in the solutions under test does not appear feasible. It has been suggested that the gradual change accelerates throughout the reaction. It can be noted that the time interval between stopping of filtration of the solutions during the tests and visibility to the eye of the gel was always much less than that between the start of the tests and the stopping of filtration, or, in other words, the rate of transition was greatly accelerated, visibly accelerated, from the time of stoppage of filtration onward. It might conceivably be deduced from this that the rate of the transition accelerates in some mathematical progression throughout the test. It is conceivable that such acceleration might be due to osmosis within the solution, minute diaphragms of silicic acid gel being set up and osmosis through these leading to local concentrations of the crystalloid zinc sulfate, which, as the author shows, would hasten the transposition of the hydrosol silica in the zones of crystalloid concentration.

The author also shows by means of the curves in Fig. 1 that for any definite silica concentration, an increase of either zinc or sulfuric acid will shorten the time period within which filtration of the solution will cease, or, in the light of the hypothesis advanced above, will speed up the rate of transposition of hydrosol silicic acid to hydrogel. In order to increase the zinc concentration, the SO_4 ion concentration must simultaneously be increased. Increase of the sulfuric acid concentration also is equivalent to increase of the SO_4 ion concentration. The possibility, therefore, is suggested that the rate of transposition of the hydrosol to hydrogel in the system under consideration may, after all, be dependent on the SO_4 ion concentration.

It might have increased the available data if the two portions of a test solution which, as the author states, always resulted in a filtration of some and a remaining behind on the paper of the rest of the solution, had been subjected to analysis. It is possible that some difference might have been detected between the portion going through the paper and that remaining behind.

If the above hypothesis be accepted, it appears to complicate matters still further. In place of the straight solution of silicic acid abruptly transposing itself to the silicic acid gel, we have solutions containing silicic acid sol, silicic acid gel, and also silicic acid in the state of transition, as well as the zinc sulfate and sulfuric acid. In view of this possible complication of the whole system, I would like to appeal to the author, or to other members versed in the science of colloidal chemistry, to point out the way to a clear understanding of the whole matter and to advance some theory that will fit in with the collected data.

The excess of zinc oxide required over that which can possibly be taken into solution by the hot zinc sulfate solution is probably due to the fact that each particle of silica, as it gels, automatically encloses particles of zinc oxide, thus insulating the enclosed oxide from further action. The proportion of silica escaping precipitation, the author quotes as 1 to 2 per cent. of the total. This is probably a fair average, though as high as 5 per cent. going through has been noted.

Recently some tests were performed to obtain data of the time-temperature factors of the silica precipitation. Two portions of a leach solution were separately treated with similar amounts of zinc oxide and raised to boiling point, when one of the tests was removed from the heating source and insulated by being placed in a container packed with hot lime. Temperature was noted at definite time intervals and the condition of filterability was determined for each interval. Both solutions showed considerable filterability at as low as 60° C., which was reached in about 10 min. Solution A took 20 min. to boil, filtered well at that time, but did not run completely dry; at 30 min. it was considered completely filterable. Solution B took 25 min. to boil and was then insulated from the heat as above described. At 30 min., it likewise was completely filterable. Hence it is evident that continued boiling is not essential to ensure complete precipitation and that the time element enters here to a greater degree than temperature. It was thought, however, that the silica was completely precipitated at boiling and the additional time was necessary to granulate it completely; it was noted that the silica from B was less granular than that from A.

In discussing the transition of gels, etc., the author uses the term "alkaline side." It might have been preferable to retain the term "basic," for the solution at no time is alkaline to any ordinary indicator.

Regarding Table 2, it has to be stated that the tendency that will be noticed for the silica in solution to rise after the minimum point had been reached, is probably due to the effect of concentration by evaporation. Tests recently carried out to show the concentration of zinc oxide obtainable in straight zinc sulfate solution gave interesting results. The solution employed in Table 2 was duplicated, except for the silica content, and the conditions of the new test were identical with those of the one in the paper, namely, 2000 cc. solution heated to 90° C., 5 gm. pure zinc oxide stirred in, and readings of basicity (as determined by titration of measured and filtered portions of solution with standard acid, using methyl orange as indicator) were made at intervals of 7 min.

TIME INTERVAL, MINUTES	BASICITY, GM. ZnO PER LITER
0	0.000
7	0.478
14	0.478
21	0.478
28	0.478

This table shows that the solution became saturated with zinc oxide very shortly after its addition and then remained constant throughout. The above figures agree closely with those given in the table for the 21- and 28-min. periods, the value for 35 min., in the table, being probably increased by evaporation; 28 min. is also the time corresponding to lowest silica (disregarding the 55-min. test) and the full basicity possible in that concentration of zinc sulfate was then attained, the precipitation of the silica having had no effect on the basicity of the solution, which was, no doubt, maintained by the excess of zinc oxide.

The real significance of this contribution is the fact that a means is now offered for the treatment of zinc ores carrying large amounts of acid-soluble silicic acid, without resorting to the procedure deemed necessary by O. C. Ralston, who, in his Hydrometallurgy of Zinc, states that such ores require to be heated with strong sulfuric acid to a sufficiently high temperature to desiccate the silica before anything can be done with a leaching process.

JESSE O. BETTERTON (Author's reply to discussion).—The figure for acid-soluble silica in the ore given in the discussion is correct. The author, though, would like to know the brand and composition of the dust that "inevitably" contaminated this residue, as it was the custom to weigh and sample carefully all products resulting from an ore leach and then take the analyses corresponding and account for the distribution and total of the ore contents for each leach. There were many changes in laboratory procedure and new analytical methods as a result of the accuracy of such data from the plant. The accompanying table gives the results of over 30 ore leaches; it will be noted that the average silica content of the zinc-dust purification residue was 8.6 per cent.

SiO_2 Distribution.—ORE LEACHES 52–85 INC.

PRODUCT	WEIGHT, LB.	SiO_2, PER CENT.	SiO_2, LB.
Ore	14,487	13.7	1,985
Ore residue	5,315	7.3	388
SiO_2 residue	3,900	36.0	1,404
Zinc-dust purification residue	1,529	8.6	132
Unaccounted			61

A large part of the zinc-ore reserve of the property consisted of zinc carbonate ore and mixtures of carbonate and silicate ores. The object of investigating the gelling tendencies of CO_2 was to determine whether these ores would introduce additional treatment problems. There were no gelling tendencies that could be ascribed to the presence of CO_2. The statement in Watt's Dictionary of Chemistry referred to a condition too far afield from the purposes and scope of this investigation, so the author made no attempt to prove or disprove it.

As regards the probable gelling effects of vanadic and phosphoric acids: both were present in the tests carried out for plotting the curves, as ore, spent electrolyte, and head-tank solution were used in making up the tests. Approximately 2.0 gm. per liter of the free acid shown results from the presence of combined vanadic and phosphoric acid. Their presence in these amounts was a condition of the treatment problem that had to be met and, as usual, experimental effort did not go beyond such limits, as there was not the time, equipment, or trained personnel available for purely scientific investigations.

In Fig. 1, in the visible gelling curves the points for 65 gm. zinc per liter are out of alignment; likewise, in two cases, for 55 gm. zinc per liter. It is unlikely that any special significance can be attached to these irregularities. There was no positive test for placing these points, and in any 24 hr. three investigators were employed. It is probable that any one investigator would have produced a more regular curve but the time position of the whole curve would probably vary somewhat for each investigator. The curve of acid effect and the statement in Mellor that the presence of a little acid retards gelling, would appear in accordance with, rather than contradiction to, the statements and curves given in the paper when viewed as a whole.

In the case of any of the zinc-sulfate solutions containing silica under discussion, in a range starting with moderately high acidity, through low acidity, the neutral point, and ending with the maximum alkalinity that can be obtained by dissolution of zinc oxide, the time interval for gelling will be longest with the solution slightly acid, unless a reversal occurs in the curve of acid effect as the neutral point is closely approached, or the maximum time occurs at exact neutrality. In Fig. 2, the curve of silica concentration shows a reversal at 9 gm. per liter instead of 10 gm., as would be expected from a consideration of the curves in Fig. 1.

The curves in Fig. 1 were plotted from results obtained as follows: Ore and solutions used were made up in bulk and analyzed in duplicate. The calculated quantity of ore for each determination was weighed and the required solutions measured from a burette. After dissolution of the ore was completed, usually three samples were analyzed to check the work. While good agreement was obtained between calculated results and results obtained by analysis, they were not exact checks. The calculated results were used in plotting the curves. The same procedure was followed for a silica concentration curve and showed the gelling time of 9 and 10 gm. of SiO_2 per liter to be practically the same, a difference of but 30 min. being noted. Because of the importance of this curve the work was repeated and each test analyzed. The curve given was plotted from the results of these analyses. The analyzed results for SiO_2 were somewhat lower in every case than the calculated results, the greatest difference being at 9 gm. SiO_2 per liter. A consideration of all these facts makes it probable that the point of reversal is between 9 and 10 gm. SiO_2 per liter. It may also be an error to plot the upper part of the curve as a straight line. More points would be required to determine this.

The author stated that it was fortunate for the success of practical work that the formation of a gel did not start at once and uniformly increase as time elapsed, but that it occurred rather abruptly after a lapse of time. Solutions that will filter readily and positively through as coarse a sieve as a Whatman's No. 30 filter paper, 11 cm. diameter for upwards of 300 hr. and then "rather abruptly" form so much gel that they can only partly pass these large openings, appear to have undergone some abrupt change. The idea of an accelerated rate of change from the start is, of course, not without its virtues from a purely speculative viewpoint, but for practical purposes the first stretch of the mathematical progression is nil and the transition annoyingly abrupt. However, the author believes the transition to be abrupt and to be due to supersaturation of the solutions with respect to silica. Following this view the dissolved silica probably exists in a colloidal condition until the first formation of crystalline nuclei acts similarly to the introduction of a crystal of sodium sulfate in a supersaturated solution of that salt. Or, it may be that the silica when in a colloidal condition is changing to another modification because of the influence of strong electrolytes present, and that when the solution becomes saturated with respect to the modified form that it begins to deposit the gel.

Whether the gelling is abrupt or progressive, whether it is the result of supersaturation or osmosis within the solution, the gelling time of any given silica solution is inversely in relation to the amount of strong electrolyte present. In the paper these relations were expressed in terms of zinc, the concentration of free H_2SO_4 and SiO_2, factors under the control of the metallurgist, as a matter of convenience. Summarized, they can be embodied in the simple relation stated.

Attention has been called to the fact that solutions of colloidal silica as high as 14 per cent. SiO_2 can be made, but as these gel the author fails to see what bearing this fact may have on the rate of gelling. As no strong electrolyte is present in such cases, explanations founded on the influence of strong electrolytes or osmosis evidently could not hold; it might, however, be a case of supersaturation of the solution.

The writer was much interested in the new material submitted bearing on the precipitation of silica. In the paper attention was called to the fact that hot zinc sulfate dissolved zinc oxide, thereby increasing the hydroxyl concentration, and that this condition was necessary for the precipitation of practically all the silica in filterable form. To speak of the solution as being alkaline would appear proper; the degree of this alkalinity is determined by means of titration with standard acid using methyl orange as indicator. The change to acidity is clearly indicated when using methyl orange of proper dilution, although methyl orange is not an entirely satisfactory indicator for $ZnSO_4$ solutions; as far as the writer's experience goes there is no fully satisfactory indicator. Regarding Table 2 and the corresponding table in the discussion, the later work supplements the former and clears up such minor irregularities as existed.

Proposed Plan for Crushing, Grinding, and Concentrating Low-grade Sulfide Ore

By Arthur Crowfoot, Morenci, Ariz.

(Canadian Meeting, August, 1923)

As a result of work done, on an operating scale, in its experimental mill during 1920 and 1921, the plan here given was proposed for crushing, grinding, and concentrating low-grade sulfide ores at the Morenci Branch of the Phelps Dodge Corpn. The flow sheet is designed to recover 85 per cent. of the total copper and 95 per cent. of the sulfide copper from ore assaying 2.0 per cent. total copper and 0.3 per cent. acid soluble copper. At present, 1400 tons are treated daily under the conditions of this flow sheet, two 700-ton units having been installed in the 4000-ton concentrator.

The run-of-mine ore will be reduced, in the dry-crushing plant, to −6 mesh and then delivered to the concentrator feed bins. The crushing will be done in three stages: (1) Run-of-mine ore is reduced to −2½-in. in a gyratory crusher; (2) the 3-in. ore is reduced to ½-in. in 72 by 24 in. rolls; (3) the ½-in. ore is reduced to −6-mesh size in 54 by 24 in. rolls. The 54-in. rolls will operate in closed circuit with electrically vibrated screens, "Hum-mer" type, a conveyor circuit being used to carry the roll product to the screens. Sufficient screens will be installed to pass 1.5 tons undersize per hour per square foot.

Fine-ore bin chutes will be piped for compressed air, which will be used intermittently to keep the ore feed in motion if it tends to arch or pipe in the bins.

The Concentrator

The concentration of the copper and iron sulfides in the ore will be effected in three stages:

1. By direct treatment of the −6-mesh crushing-plant product on primary tables, which will recover about 35 per cent. of the original copper content of the ore in a clean granular concentrate. The table middling will be dewatered in Dorr rake classifiers and ground to −48-mesh in ball mills operating in closed circuit with circulating classifiers.

2. The −48-mesh overflow of the circulating classifiers will be treated by flotation at the rate of 3 tons per sq. ft. of cell area in service per 24 hr.; a froth concentrate is produced, about one-third of which will be finished concentrate and the balance rough concentrate, which will require further treatment in cleaner cells. The −48-mesh middling from the primary flotation cells will be classified in bowl-type Dorr classifiers designed to overflow −65-mesh material; the sand raked out will be ground in ball-tube mills working in closed circuit with these classifiers.

3. The −65-mesh overflow of the bowl-type classifiers will be treated by flotation at the rate of 3 tons per sq. ft. of cell area in service per 24 hr., a rough froth concentrate being produced, which will require further treatment in cleaner cells to bring it to shipping grade.

The novel features of the flow sheet are the two-stage grinding and two-stage flotation treatment of the −6-mesh middling material from the primary tables; and the making of the waste tailing product directly from the secondary flotation rougher cells, thus eliminating the finishing installation of tables or vanners generally used in the final treatment of flotation tailing. It developed during the test work that grinding to −65 mesh size was profitable on Morenci ore and that, because of this fine grinding, the elimination of the vanner and table treatment of flotation tailing was desirable.

The problem was to determine how the work of the final tables and vanners could be thrown on the flotation system without having to enlarge this system and without violating one of the fundamental principles of concentrator flow-sheet construction, which calls for a guard unit at the foot of the mill, from which the final waste tailing is produced.

The guard-recovery unit should not normally be entrusted with the treatment of more than one-third of the original copper fed to the concentrator; that is, a two-thirds recovery or better should be made preceding the final clean up. But the unit must have sufficient capacity to take care of the additional copper sent down to it should trouble occur in the preceding recovery units. It was decided that the grinding to −6-mesh size should be done in two stages—first to 48 mesh and then to 65 mesh. Also that the flotation operation should be carried out in two stages—at 48 mesh and at 65 mesh.

It had been developed that the best flotation-feed rate for Morenci ore was about 1.5 tons per sq. ft. of cell area per 24 hr.; it was therefore decided to provide enough flotation area for the required rate of feed, or 0.67 sq. ft. per ton treated per 24 hr. The total flotation cell area was, however, divided into two units of equal areas, termed the primary and the secondary flotation units.

The −6-mesh primary table middling is first ground to −48-mesh size and passed through the primary flotation unit at a rate of 3 tons of ore per square foot of cell area per 24 hr., or double the normal rate decided on. The tailing from the primary flotation unit is then classified in bowl-type classifiers designed to overflow all −65-mesh material to the secondary flotation unit and to return to the secondary mills for regrinding all material too coarse for final treatment. The ore is then passed through the secondary flotation unit, again at the rate of 3 tons per sq. ft. of cell area per 24 hr. The ore is, therefore, floated twice at double the normal rate of flow instead of once at the normal rate. This system has the advantage that by the time the feed reaches the secondary flotation unit there is little more than 14 per cent. of the original copper left in it in a recoverable form.

Under the conditions of the flow sheet, for every hundred units of copper fed to the concentrator, there are recovered 37 units on the primary tables, 34 units in the primary flotation, and 14 units in the secondary flotation; 60 per cent. of the copper lost in the tailing is acid-soluble copper.

DEWATERING AREA REQUIREMENT FOR CONCENTRATE AND TAILING

The primary-table concentrate is dewatered in circular concrete tanks equipped with filter bottoms; the tanks are 18 ft. in diameter and 8 ft. deep. The dewatered concentrate is discharged through the bottom of the tanks, falling on to a conveyor belt, which delivers it to the loading bins. For dewatering the flotation concentrate, 80 sq. ft. of settling tank area is allowed for each ton of concentrate dewatered to 50 per cent. water per 24 hr. The dewatering is finished to 17 per cent. moisture in the filter cake on Oliver continuous filters. Because of the clayey nature of some of the gangue constituents, it is necessary to allow 13 sq. ft. of settling area for each 24-hr. ton of −65-mesh tailing produced and dewatered to 60 per cent. water and 40 per cent. solids, in which condition the tailing is flumed 3 miles to the tailing disposal plant.

LARGE DUPLEX, SHALLOW, FLOTATION CELL AT MORENCI CONCENTRATOR

When considering the treatment of large tonnages of pulp by the flotation process, it is necessary to consider the adoption of large unit flotation cells in order to conserve operating labor and space and to reduce the number of air piping fixtures, etc. A duplex unit that will treat 500 dry tons of flotation feed per 24 hr. has generally been considered an advantageous size to use in large capacity concentrating plants; this unit is probably the best for practical commercial operation, although larger units can be constructed.

The special feature of the large unit cell developed at this plant is its small depth, whereby power consumption, and consequently flotation cost, is reduced partly because of the decreased static head and partly because the blankets are available for periodical cleaning.

This cell is the outgrowth of a shallow cell developed for use in the experimental mill. In August, 1919, there were installed in the experimental mill four cells of the same width as the standard Callow cells in use, but twice as long—two standard 8-ft. bottoms being placed end to end. The depth at the feed end was 15 in. and at the discharge end 17 in. The cells were practically wide flat launders equipped with porous bottoms. There were no cross-section baffles except at the discharge end for the overflow of waste tailing material. The cells worked satisfactorily in the treatment of −65-mesh material (the second stage of flotation). An important advantage gained is a saving of about 25 per cent. on blower horsepower consumption, the pressure requirement at the cell being reduced from an average of about 4.75 lb. per sq. in. to about 3.5 pounds.

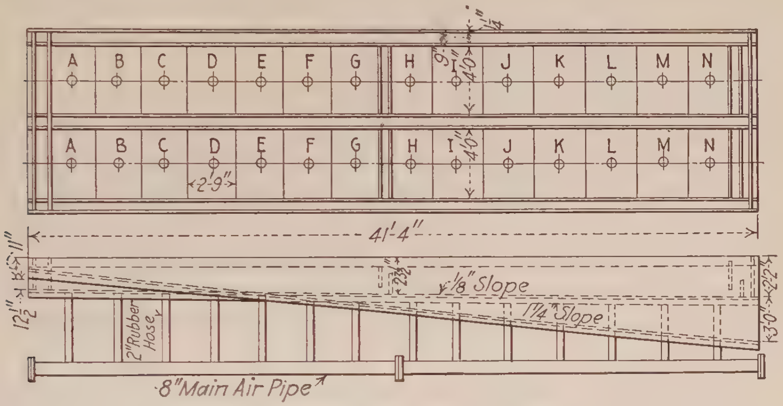

FIG. 1.—40-FT. FLOTATION MACHINE.

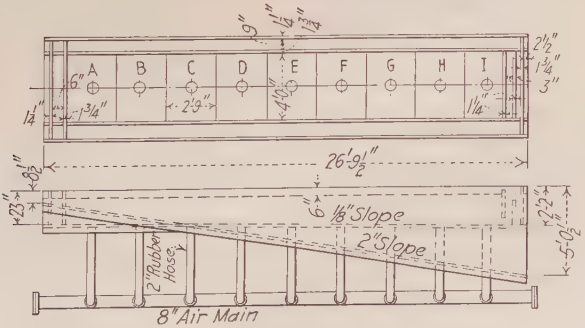

FIG. 2.—CLEANER FLOTATION CELL.

The large unit cell for use in the concentrator was constructed as a result of the experience gained with these cells. This large cell is a duplex unit, each unit having a net inside length of 40 ft. and a net inside width of 4 ft. As first constructed, it had one cross-sectional baffle in the center of each compartment, as shown in Fig. 1. This was removed later.

The unit contains twenty-eight air compartments, fourteen in each cell unit, each compartment being 47 in. wide by 35½ in. long. Starting with a depth of 15 in. at the feed entry the first seven air compartments are laid on a slope of ⅛ in. per ft. The eighth air compartment is raised so that it is 15 in. below the froth overflow and the remaining six compartments are laid on a slope of ⅛ in. per foot.

Air is supplied through the bottoms of the air compartments from two 8-in. air mains laid on the floor, one on each side of the cell. Connection from the main to the compartments is through 2-in. four-ply, rubber hose, 5 ft. long. The top of the cells is therefore entirely free from air piping and there are no obstructions when it becomes necessary to change blankets.

The cleaner cell is constructed along the same lines (see Fig. 2).

Under the conditions of the Morenci flow sheet, the waste tailing product of the concentrator is produced directly from rougher cells of this description.

At the time this flotation unit was installed (1920), it was developed that, in the treatment of Morenci ore, it would be better to reduce the tonnage floated per square foot of cell area from 2.5 or 3 tons to 1.5 tons, exclusive of return middling.

A comparative test run was made between the new cell unit and the standard 8-ft. Callow cell units to determine what improvement in extraction might be expected. The flotation feed, at the time, was carrying from 33 to 40 per cent. of its total copper value in the acid-soluble form; therefore, the end results obtained are not especially good though they are comparable. The following results were obtained:

Data	40-ft. Duplex Cell	Standard Cell System	Difference
Assay per cent. total copper in feed	1.42	1.32	+ 0.10
Assay per cent. acid-soluble copper in feed	0.48	0.53	− 0.05
Assay per cent. sulfide copper in feed	0.94	0.79	+ 0.15
Assay per cent. total copper in tailing	0.64	0.74	− 0.10
Assay per cent. acid-soluble copper in tailing	0.39	0.42	− 0.03
Assay per cent. sulfide copper in tailing	0.25	0.32	− 0.07
Assay per cent. total copper in concentrate	21.27	22.58	− 1.31
Assay per cent. acid-soluble copper in concentrate	1.17	1.25	− 0.08
Per cent. of total copper recovered	56.63	45.43	+11.20
Per cent. of sulfide copper recovered	74.33	60.40	+13.93
Tons floated per square foot rougher porous medium	1.42	2.48	−1.06
Air pressure, pounds per square inch	3.5	4.75	−1.25

The + and − indicate that results from 40-ft. cell operation are greater or less than results from standard cell operation.

From these figures, the 40-ft. duplex cell floating 1 ton less per square foot of porous medium recovers 11.2 per cent. more of the total copper and 13.93 per cent. more of the sulfide copper than the standard flotation cells. The feed to the 40-ft. flotation unit was a little richer in both total and sulfide copper than the feed to the standard cells, as shown by the following:

Data	40-ft. Duplex Cell	Standard Cell System	Difference
In the feed			
Pounds total copper per ton	28.4	26.4	+ 2.0
Pounds acid-soluble copper per ton	9.6	10.6	− 1.0
Pounds sulfide copper per ton	18.8	15.8	+ 3.0
In the tailing			
Pounds total copper per ton	12.8	14.8	− 2.0
Pounds acid-soluble copper per ton	7.8	8.4	− 0.6
Pounds sulfide copper per ton	5.0	6.4	− 1.4
In the concentrate			
Pounds total copper per ton	425.4	451.6	−26.2
Pounds acid-soluble copper per ton	23.4	25.0	− 1.6
Pounds sulfide copper per ton	402.0	426.6	−24.6
Extraction			
Pounds total copper per ton	16.1	12.0	+ 4.1
Pounds sulfide copper per ton	14.0	9.5	+ 4.5

The standard cells produced a concentrate 1 per cent. higher in total copper value than the concentrate produced in the 40-ft. unit; this, however, is a question of manipulation and one which, within a range of 3 or 4 per cent. of copper in the concentrate, does not affect the tailing to any appreciable extent.

DISCUSSION

R. C. Canby, Wallingford, Conn.—Eleven years ago, Doctor Ricketts said that flotation would be eventually simply a launder with air admitted at the bottom through a porous medium; this process is perhaps a step in that direction.

It is unfortunate that we do not use the proper name for this class of flotation, we should distinguish between "bubble column flotation" and "pulp-body flotation" (Taggart's term). In the bubble column flotation, there is no separation of the concentrate in the pulp; the separation takes place in the bubbles above the pulp. A large series of tests, with sorting assays, have shown conclusively that this is the case.

Mr. Towne, the discoverer of this process, originated the term "bubble column." As these bubbles rise from the surface of the pulp, if watched through a glass-side cell, the little grains of sand flowing around the bubbles and the little grains of sulfide flowing downwards between and around the bubbles can be seen; but the sand goes down more rapidly than the bubbles rise, whereas the sulfide particles, due to the contaminating agent, move downwards slightly slower than the bubbles rise and are, therefore, carried up and over, and constitute the concentrate. So that there would undoubtedly be a limit to length and shallowness of a cell.

To show Mr. Towne's appreciation of the descriptive character of this term, it should be noted that at the time he discovered this action of the rising bubble and the descending particles, he changed the name of his company that was to introduce the flotation process, to "The Bubble Column Corporation," reincorporating his company for that purpose. I wish that we would get the fact into our minds that there is this distinction in flotation. There must be room for the bubble column above the pulp. The necessity of a deep body of pulp may not exist because the separation takes place above the pulp.

We have always talked about the Callow process because the Callow cell was in use, but it is the bubble-column process that should be recognized as the means of separation. You do not necessarily need depth of pulp, it is above the pulp that you need sufficient depth for the bubble column. Mr. Towne insisted upon multiple opportunity. He may not have described the action of the bubble column as clearly as has Taggart, but the name he chose for it was little short of an inspiration. The long cell gives multiple opportunity. The Callow cell also allows multiple opportunity, with the rising column of bubbles giving opportunity for the particles to separate.

A Contribution to the Kick versus Rittinger Dispute

By H. E. T. Haultain, Toronto, Ont.

(Canadian Meeting, August, 1923)

The study of rock crushing or grinding in tube-mills is difficult on account of the large size of the units employed in the field and the large number of variables entering into the problem. Three or four years ago, in the laboratory of the Department of Mining Engineering of the University of Toronto, a serious effort was commenced to study the matter under laboratory conditions. Experiments were carried on in a cylindrical tube-mill 18 in. long and 12 in. in diameter, inside measurements,

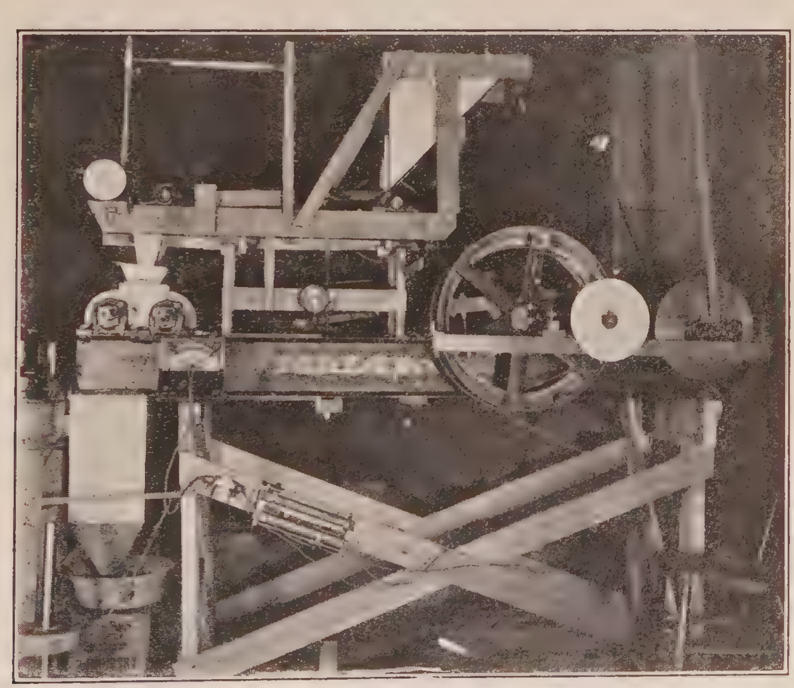

Fig. 1.

supported on ball bearings and connected with a dynamometer. Several hundred experiments were made, but it was early realized that, for proper interpretation of the results, more definite information was required in regard to the laws of crushing as interpreted by Kick and by Rittinger.[1] For the study of this phase of the work it was thought that a more simple apparatus, a more simple process of crushing was required, and that this could best be found in small carefully constructed rolls. This apparatus, shown in Fig. 1, was obtained by the financial help of the Honorary Advisory Council for Scientific and Industrial Research of Ottawa. Rolls 7 in. in diameter with 2¼-in. face were mounted on shafts supported by the highest grade of S. K. F. ball bearings, and were driven from a countershaft, also supported by S. K. F. bearings, which was connected with an integrating dynamometer. This apparatus proved to be more sensitive and reliable than had been expected and gave concordant results when working with quantities of rock as small as from 2 to 5 lb. The following figures show the range of error.

Feed, Pounds 4–6-mesh Hollinger Quartz	Rate of Feed, 5 lb. in Seconds	Integrator Readings, Foot–pounds
5	116	3874
5	63	3952
5	40	3919
5	31	3945
	Average.	3922

With a different setting of the rolls, the rate of feed was kept practically the same.

Feed, Pounds 4–6-mesh Hollinger Quartz	Rate of Feed, 1 lb. in Seconds	Integrator Readings, Per 1 lb. of Feed
1	11.0	754
2	10.0	754
5	9.4	744
10	10.3	733
	Average.	746

The process of crushing rock produces new rock surface and heat; also, to a minor extent, noise and electrical phenomena. The physicists have connected up the energy of new surface formation with the phenomena of surface tension and of latent heat. (The formation of steam from water is a surface-producing process pushed to the extreme.) From their calculations, it would appear that the energy actually absorbed in producing the new surface formed in the process of crushing is small compared with the total energy required in practice. Doubtless, the energy absorbed only in new surface formation is proportional to the new surface, as indicated in Rittinger's philosophy. Apparently most of the energy required in the practice of rock crushing appears as heat, but surface formation *per se* does not generate heat. The heat must be due to friction, friction occurring at the surface of the particles and friction occurring within the particles, the latter being the internal friction of distortion (molecular friction? hysteresis?). The surface friction will vary with the type of apparatus used and, doubtless, with the type of rock being crushed, and with the degree of crushing.[2]

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[1] *Kick's Theorem.*—"The energy required for producing analagous changes of configuration of geometrically similar bodies of equal technological state varies as the volumes or weights of those bodies."

Rittinger.—"The energy required for crushing is proportional to the new surface formed;" *i.e.*, for any given unit weight of rock, is proportional to the reciprocal of the diameter of the average size.

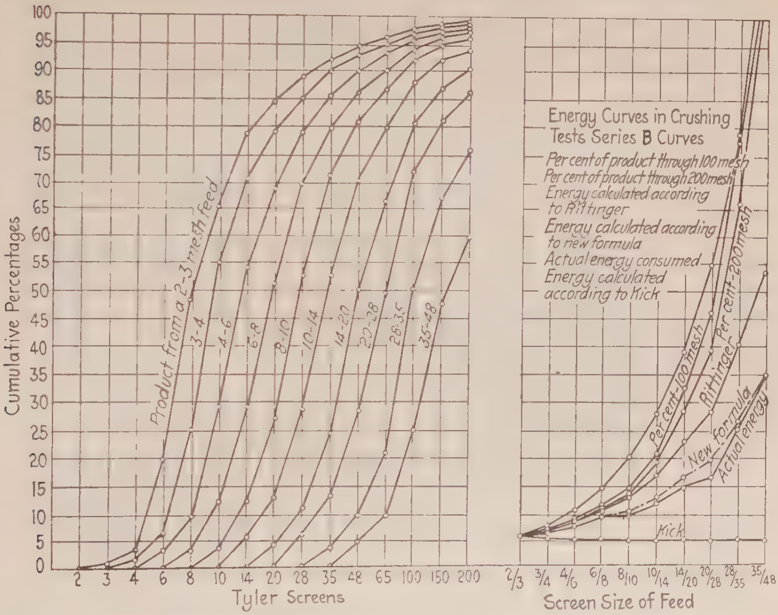

FIG. 2.—CUMULATIVE PERCENTAGE DIAGRAMS OF SCREEN ANALYSIS OF PRODUCTS FROM TEN CRUSHING TESTS ON HOLLINGER QUARTZ; IN EACH CASE, RATIO OF AVERAGE MESH OF FEED TO SPACING OF ROLLS WAS 2.4 to 1.

The heat resulting from the internal friction of distortion (would that it might be safely referred to as molecular friction) that occurs prior to rupture may probably vary according to Kick's philosophy, but it would seem difficult to reasonably connect with either the Kick or the Rittinger theories the energy consumed in surface friction. The writer ventures to think that those responsible for lengthy arguments in favor of Kick or of Rittinger have not given due consideration to this phase of the problem. Apparently a very large number of experiments, rather than argument, will be necessary for a satisfactory conclusion.

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[2] There is no doubt that rocks behave very differently during crushing. Some rocks are tough, some are brittle. Professor Dyer and the writer have studied and visualized their behavior by a series of photographs taken at the rate of 120 photographs per second, which are shown on the screen at the normal movie picture rate of 15 per second.

Two series of experiments with the new apparatus are illustrated in Figs. 2 and 3. The rock was crushed in a small Dodge crusher, then carefully sized by Tyler screens in a Ferraris type of screening apparatus. In the Tyler series, the ratio of the diameters of the openings in adjacent screens is the square root of two, 1.414, and each feed product was screened between adjacent screens. For each size of feed product, the rolls were set with an opening equal to one-half the size of the smaller screen opening. For example, the product that passed through 8 mesh and stayed on 10 mesh was crushed by rolls set with an opening equivalent to 20 mesh; for a product between 28 and 35 mesh, the rolls' opening was 65 mesh. Thus for each product there was attempted an equal degree of crushing; "analogous changes in configuration." The crushed products were screen-sized with Tyler screens and are plotted as cumulative percentages. The similarity of the curves shows the degree of

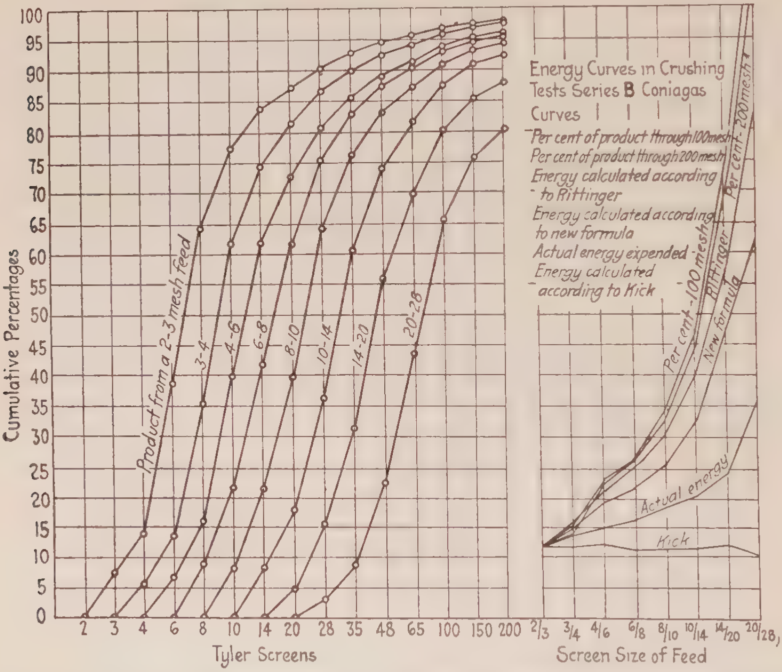

FIG. 3.—CUMULATIVE PERCENTAGE DIAGRAMS OF SCREEN ANALYSIS OF PRODUCTS FROM NINE CRUSHING TESTS ON CONIAGAS CONGLOMERATE; IN EACH CASE, RATIO OF AVERAGE MESH OF FEED TO SPACING OF ROLLS WAS 2.4 to 1.

uniformity. Departure from complete uniformity may have been due to the difficulty of setting the rolls exactly the right distance apart.

The energy actually required per pound of rock crushed, as shown by the dynamometer, is plotted in the graph to the right. For comparison are shown the curves as calculated from the screen analyses, according to Kick and to Rittinger, all calculated as starting from a common point for the first crushing in the series. Also for comparison are shown curves for the percentage of 100 mesh and 200 mesh produced in each crushing. These are also calculated to start from a common point; that is, different scales are used, the object being simply to show the nature of the curve for comparison with the others.*

A graduate student, J. G. McNiven, carrying out these experiments, developed a formula on a rational basis, which fitted very well with the results for Hollinger quartz, but not so well for Coniagas conglomerate, a compact, very fine-grained tough rock.

In each set of experiments both the Rittinger and the Kick figures are far from the actual results. With the Hollinger rock, the Rittinger figures are the nearer, while the Kick figures are somewhat nearer with the Coniagas rock. Coniagas rock took about twice as much energy as Hollinger quartz.

The sizing screens take no cognizance of shape. They are at best an imperfect measuring stick and as yet there is no measuring stick available for shape. In many of these experiments, particles were counted and weighed in the different sizes, showing a similarity between coarse and fine. A study of the shapes produced by various crushing methods has been commenced and promises to show marked differences.

Attention should be drawn to the possible, or probable, lack of "equal technological state" in different sized particles of even such a uniform rock as the Hollinger quartz, which was used in these experiments. In all rocks, there must be planes or zones of weakness due to strains of cooling or of pressure which may be classified as geological, or due to crystal structure or mineral aggregate which may be classified as mineralogical. Apparently a substance without any of these weak-

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*NOTE.—The figures used as multipliers in the Kick and Rittinger calculations are as follows:

MESH	KICK	RITTINGER	MESH	KICK	RITTINGER
2	3	2.23	28	11	35.8
3	4	3.15	35	12	50.5
4	5	4.46	48	13	71.5
6	6	6.34	65	14	101.0
8	7	8.92	100	15	142 8
10	8	12.65	150	16	204.2
14	9	17.85	200	17	286.0
20	10	25.1	−200	19	715.0

nesses breaks under compression with a conchoidal fracture, and in crushing with rolls tends to be very flaky. It may be that the reduction in the number of these weaknesses as the particles become smaller is the cause of the rise of the crushing energy curve. It may be that well-annealed glass would give a curve much nearer to Kick's.

The writer submits the following tentative conclusions:

Neither Kick nor Rittinger accurately connects up size of product with energy required to crush rock.

The mean of the two is probably more nearly correct than either in the simplest form of crushing; *viz.*, cracking with rolls.

The nature of the process and of the material prevents any simple formula applicable to all rocks, or all degrees of crushing.

The energy required for the actual process of crushing (disregarding losses in the machinery itself) is absorbed mainly in three ways: (1) Actual formation of new surface, that is, the final rupture; this is a small part of the total energy. (2) Internal friction accompanying the distortion prior to rupture. (3) Surface friction of particle on particle and of particle on crushing surface.

This surface friction probably accounts for the greater part of the energy, and is the most important variable in practice. It varies with the rock, the type of crusher, and with the degree of crushing. It is probably least in rolls just cracking particles of a brittle material and greatest in an overloaded tube-mill. It is high in a disk grinder and in the arrastre.

It is in variation in this surface friction that the main variation in efficiency occurs. This being a surface action it is probable that it follows closer to Rittinger than to Kick. If this be the case then the less efficient the crushing method the less will Kick's law apply.

This problem can best be studied with small laboratory apparatus on account of the large number of experiments required. With suitable apparatus, it will be possible to construct from laboratory experiments a characteristic curve for any given rock, which may be advantageously applied to the interpretation of field results.

It is hoped that with further development of methods and technique with small apparatus of the type indicated, it will be possible to predict with useful accuracy the power and size of plant required for any given rock crushing.

DISCUSSION

Robert S. Lewis,* Salt Lake City, Utah (written discussion).—This paper brings out some interesting points in regard to the interpretation of the results of crushing tests. Much power is consumed in making the finer sizes, but, as yet, there is no standard method of measuring the material that is − 200 mesh. The accepted method of reporting crushing tests is to assign a lump figure to this material. It is not known how correct this figure is. In Fig. 2 the product from the 2–3 mesh feed contains less than 5 per cent. of minus material, but the product from the 35–48 mesh feed gives 40 per cent. of – 200 mesh material. Due to the wide variation that would result if the – 200 mesh material were not correctly evaluated, it would seem that confining tests to the coarser material and thus producing a minimum of – 200 mesh product would give results that could be more accurately interpreted.

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* Professor of Mining, University of Utah.

Again, the size of particles on any one screen size do not bear a fixed relation to the size of opening through which they have last passed. Different rocks break into different shaped particles, and long lath-shaped grains give different screening results from flat tabular grains. Some tests made by H. C. Sherman, a graduate student at the University of Utah, illustrate this point. Portions of galena, quartz, and sphalerite were screened on Tyler screens in a Ro-Tap machine for 30 min. Then from 20 to 50 grains of each size were examined under the microscope. The length and breadth were measured and averaged; the vertical dimension was not determined. The results are tabulated below.

RATED APERTURES OF TYLER SCREENS	
48 mesh	0.295 mm.
65 mesh	0.208 mm.
80 mesh	0.175 mm.
100 mesh	0.147 mm.
150 mesh	0.104 mm.
200 mesh	0.074 mm.

MEAN DIAMETER OF LIMITING SCREENS ON THESE SIZES	
For – 48 + 65 mesh	0.2515 mm.
For – 65 + 80 mesh	0.1915 mm.
For – 80 + 100 mesh	0.161 mm.
For –100 + 150 mesh	0.1205 mm.
For –150 + 200 mesh	0.089 mm.

MEASURED PARTICLES, MILLIMETERS

	QUARTZ		SPHALERITE		GALENA	
SCREEN SIZE	LENGTH	BREADTH,	LENGTH	BREADTH.	LENGTH	BREADTH
– 48 + 65	0.465	0.319	0.448	0.299	0.376	0.270
– 65 + 80	0.353	0.247	0.349	0.241	0.315	0.222
– 80 + 100	0.281	0.203	0.405	0.204	0.254	0.175
– 100 + 150	0.242	0.160	0.293	0.210	0.224	0.145
– 150 + 200	0.168	0.133	0.190	0.111	0.153	0.106

These points emphasize difficulties in the way of determining the exact energy consumed in crushing but in no way do they detract from the possibility of making useful predictions in regard to large-scale crushing from carefully made laboratory tests, once a definite method of conducting such tests is established.

J. W. BELL,* Montreal Quebec.—The author's results are of great importance if he has devised a practically perfect machine for measuring the power used in crushing. In order to understand his results clearly, I would like to take his data and work it over into the form that I am accustomed to use.

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* Assistant Professor of Mining Engineering, McGill University.

Looking at these results in a very casual way, it strikes me that the actual facts are that the quartz results show that Kick's law is totally inaccurate. In the 35- to 48-mesh feed tests, if the power required by Kick's law should be, say for argument, 1 hp., the author's tests, assuming that his machine was perfectly efficient, called for 7 hp., and the Rittinger law for 10½ hp. I do not see how the author can look at that result and consider Kick's law at all reliable. Assuming the absolute efficiency of this machine (in other words, that the power measurements are perfect) Kick's law says that crushing requires only one-seventh the power that this theoretically perfect machine calls for. The Rittinger curve shows that about 1.5 times the actual power is required; in other words, the Rittinger result is too high, but it is not nearly as much higher as the Kick result is lower. The Kick result is perfectly absurd; and all of my investigations have convinced me that Kick's results are absurd when applied to the crushing of rock.

One of the things I have noticed when the theorists discuss Kick's law is that they talk about anything except rock, which is the only thing in which we are interested. The thing in which we are interested is the development of some practical way of measuring the efficiency of crushing machines. I do not want to be dragged into a discussion of the theory; in the first place, I find it difficult to understand. I cannot see just how the surface tension of solids and all those things come into the question. I do not wish to discourage the theoretical considerations, but I would wish to see work done that would result ultimately in a practical result. The author has made a good start toward doing a great deal of that work; there is no doubt, however, that he will have to do a great deal more.

In his crushing tests on the Coniagas conglomerate, the Rittinger theory does not make a very good showing, but it does not make quite so bad a showing as the Kick. If we look at that part of the curve having the greatest variations, the Kick theory calls for around 2 hp., the actual energy we will say is 7 hp., and the Rittinger theory calls for around 16½ hp. In my experimental work, we had some results that seemed to indicate the possibility of Rittinger's theory not being quite right on fine sizes. Apparently, especially in crushing the conglomerate, the amount of surface produced per horsepower, calculated by the Rittinger theory, is too great. Our own tests indicated that possibility.

Rocks are so variable in composition and structure that I never have believed that Rittinger's theory could be used as a law in the sense of the law of gravitation, for instance. We know that the theoretically practicable results obtained with the falling ball machine and the theoretical results are in close agreement, and I have not the belief at all that Rittinger's law is an absolute law; I think that it is the nearest approach to the practical method of measuring the efficiency of a machine that we have. Of course, after the author has made a great number of experiments, I may have to modify that belief.

There is a factor, which I have never seriously considered, that may come into these crushing tests. There are ores that contain considerable kaolin. When this kaolinized rock is crushed, a lot of extremely fine material is produced which we would include with what has been produced by the crushing operation, but which as a matter of fact is merely produced by being liberated; and it seems to me that in the case of rocks of that kind we should make corrections for a fine material that is merely liberated by crushing but which takes practically no power to produce. It seems unlikely that that factor would come into this particular series of tests on conglomerate; however, there is probably a higher proportion of what Messler calls natural slime in this Cobalt conglomerate, than there is in pure quartz. Messler, I think, has the idea that when a purely crystalline rock like quartz is crushed, there is practically no production of what he calls the "natural slime;" that is, material which he arbitrarily defines as taking a certain length of time to settle.

There is a much higher proportion of natural slime in these Cobalt porphyritic rocks than there is in Hollinger quartz. It takes a far greater settling area to settle slimes in Cobalt than it does those from the north; of course, they crush somewhat finer. It may be accounted for in that way, that more fine crushing is done on the Cobalt rocks.

I agree heartily with the author's idea that for the present we had better try to get out these curves; in other words, study a great many rocks and find out how much power it takes to produce the different kinds of units. After he has made a great many tests of that sort, I think we will be able to get a clearer idea of whether the Rittinger law is the better guide than the Kick. The difficulty I have with accepting Kick's law today results first from my own experiments.

Professor Taggart made a series of ball-mill tests at Yale, working them all out by Kick's law; I worked them all out by the Rittinger law and I do not think it is an exaggeration to say that the conclusions that one would be obliged to draw by the Kick law on that series of practical ball-mill tests were absurd. The only way that you could really get a sensible interpretation of what happened in the mill was to use either Rittinger's law or the practical man's method of measuring efficiency—the tons of a certain product produced per horsepower. The comparison of Rittinger with tons produced per horsepower was very interesting because of the striking agreement of results. The Kick's results were striking in their total lack of agreement with either of those two methods.

A result like that, of course, as far as I personally am concerned, convinced me that Kick's law was hopelessly inadequate to tell anything about the efficiency of a crushing machine—and I still think so, but I am looking forward now with a great deal of interest to the extension of the tests.

A. O. GATES, Salt Lake City, Utah (written discussion).—What is of most interest to me are the two diagrams of screen analysis (Figs. 2 and 3) from which one can compute (within the 200-mesh limitation) the surface or other effect produced by a measured amount of energy, and thus obtain a better understanding of how, where, and when this energy is converted in crushing. I wish that the "actual energy" instead of relative amounts had been given; it should be given so that it may be studied in connection with other phases of the crushing problem.

It is apparent that the Kick theorem was the basis in the planning of this study, the intent being to produce the same number of units of work according to this theorem in each crushing operation, as shown in the two figures referred to by the same amount of "energy calculated according to Kick;" this is probably as good a point to start from as any.

I would like to call attention to the relative shape of the curves "actual energy" and "Rittinger;" they have the same characteristics and, without the actual energy figures, I would say that applying a multiplier (constant) to one or the other they would be found proportional; hence a proof of the Rittinger theory. At any rate these data seem to connect up the energy with the surface produced in some form, and it does not seem to connect at all with the initial volume of the particles. The mere fact that the curve found for actual energy lies between the Rittinger and the Kick curves does not mean that the basis of measurement has anything to do with Kick.

In the little that I have contributed to the literature of the subject, I have called attention to the – 200-mesh surface which is not easily measured and which can be charged with much of the uncertainty in measuring surface. It is reasonable to calculate the surface of a definite weight of particles lying between two screen openings, as various investigators following Mr. Stadler's ideas have done. I prefer the method of the crushing-surface diagram; but there is no justification scientifically for the assignment of such figures as DelMar, Stadler, or Haultain specify to the – 200-mesh.

I suppose that the multipliers used for Rittinger calculations given by the author are the average reciprocal of diameter in inches for the particles, plus the screen size to which assigned and minus the next smaller size given; at least they are close to such a value and no doubt proportional; but the – 200-mesh has no such average value so far as I have been able to discover. Perhaps it is too far fetched to consider that some of the finest particles produced are about the size of the molecule, which may be around 0.00000001 in. in diameter, but I have a right to assume that they do approach this size as a limit. Then the average size of the – 200-mesh lies between 345 and 100,000,000, which value may possibly be determined from the position of the mean ordinate of an hyperbola between these limits, say somewhere between 1,000,000 and 50,000,000.

Coniagas rock took about twice as much energy as the Hollinger; for some reason, probably in the rock structure, about one-third more − 200-mesh was produced in crushing the Coniagas, and right there is a clue as to why; at least this point is worth looking into.

I do not agree that the energy actually absorbed in the formation of new surface is small compared with the total energy required in practice; we will find crushing efficiencies around 50 per cent. in practice, maybe much higher. Also, I do not know whether surface formation *per se* does or does not generate heat; maybe it does just the opposite, like the expansion of a gas.

But surface friction is a questionable matter; I believe that surface friction is simply the formation of infinitely fine, or even visibly coarse, particles by the rubbing of surfaces together, and therefore is no different in the end than any other portion of the crushing operation. Probably surface friction is really − 200-mesh.

As for most of the energy required in practice appearing as heat, the temperatures of a tube-mill, in Arizona, were once taken at the feed end and the discharge end; allowing for tonnage, dilution, etc., but not for radiation, between 2 and 10 per cent. of the motor power appeared as this heat. From my own experiments at Purdue, I know that some of the energy taken up by the rock in compression is returned in some way upon release of the pressure. Probably, when compressed, the rock is hotter than the uncompressed particles or pulp surrounding it and some of its heat is transmitted from it while in this state; but that is only for an instant; the conductivity of the rock is low, so that not a great amount of the crushing energy appears as heat.

As to the apparatus: it seems to me as though a belt drive is interposed between the rolls and the dynamometer; were the probable belt losses deducted in making the energy determinations? I know that belt losses in transmission are usually of greater moment than bearing losses, in spite of the amount of propaganda telling of the enormous power savings possible with antifriction bearings. The power here is apparently between 0.05 and 0.20 hp. transmitted by a relatively large belt. Of course what I am seeking is a deduction, more or less constant, that may bring the actual energy and the Rittinger curves closer together, and perhaps out of these very valuable tests prove the accuracy of the Rittinger law.

I hope that some one with the time and the apparatus will make a similar series of tests in a hydraulic testing machine, such as was used in my Purdue experiments; I believe that is far superior to rolls, ball mills, or any method where allowance in some way must be made for friction.

David Cole, El Paso, Tex. (written discussion).—I agree with the author as to the merits of the Kick and Rittinger theorems. There are, and probably will be, too many variables and too much opinion entering into the problem to permit the early establishment of the "law." We do not need the law so much as we need an acceptable method of measuring and expressing comminution resistance that can be adapted as a standard.

Lennox[3] carried out some interesting experiments along that line. His comminutor was too small for the purpose, he employed various grinding periods, and used Rittingers' yard stick to measure his results. He experimented with rocks and ores from widely separated mines and districts comparing them with "Portland mill feed" making a list that is the only index we have; it is frequently referred to and is very interesting. I would modify his procedure by using as the standard for comparison a high-resistance rock with which most managers, engineers, and millmen are acquainted. Sioux Falls jasper affords the necessary characteristics. Samples are found in every mining district of this continent; nearly all managers, mining engineers, and millmen have seen samples or have used it, so I would make this the standard rock, representing 100 per cent., in the proposed resistance scale or comminution index.

In 1914, I made a machine for making these measurements and for experimenting with different-shaped grinding media to see if an improvement on the "ball" could be demonstrated. This machine, shown in Fig. 4, consisted of a 24-in. piece of 12-in. pipe to one end of which a ½-in. plate was welded; a flanged spindle fastened to this head served as its axial support. The spindle was carried on ball bearings, to minimize the friction, so that when revolved by hand the resistance to turning would be recognized as coming from the load inside of the mill instead of from bearing friction -in this way the important matter of proper speed for the best work was felt out and determined. The cylinder had longitudinal rows of rivets with large heads inside to serve as lifters, or means to prevent slippage, of the load and the outer end was closed by a circular plate secured with hinge bolts having wing nuts. A 1¼-in. hole in the center of the cover served for introducing the water used with the charge and for visual examination of what was going on inside but was ordinarily closed with a wooden plug. A light pulley on the spindle provided means of applying power from a small motor. A revolution counter recorded the turns and, in the absence of an automatic stop switch, was used to standardize the duration of the test; a hinged arrangement permitted the tilting of the machine for emptying the contents quickly.

With locally made dies we forged steel shapes and tested the following forms of grinding media: Balls, double cones, slightly convex-faced cubes, convex-faced tetrahedrons. Each form was made in a quantity sufficient for a full charge for the comminutor. The "shapes" were approximately 1 in. in size and the same weight of each was used in the charges experimented with. A charge of rods for the machine, consisting of ⅝, ½, and ⅜ in. diameter cold-rolled steel was also provided.

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[3] "Grinding Resistance of Various Ores." *Trans.* (1919) **61,** 237.

A single large sample, cut from a uniform band of ⅛ in. quartz material coming off from a Wilfley table, was thoroughly dried, mixed, and weighed into equal standard charges having uniform characteristics and was stored in paper bags for use in the experiments; thus the same amount (and in every way the same quality) of rock, the same amount of water, the same amount of the media to be tested were charged into the machine and revolved at a standard speed the same total number of turns. The resulting pulp was then subjected to the Tyler standard screening analysis and the results plotted to express the relation of the work performed by the various shaped media.

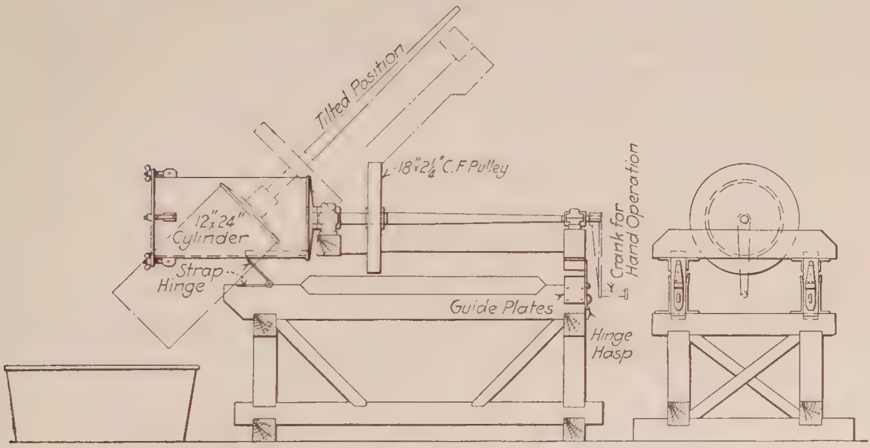
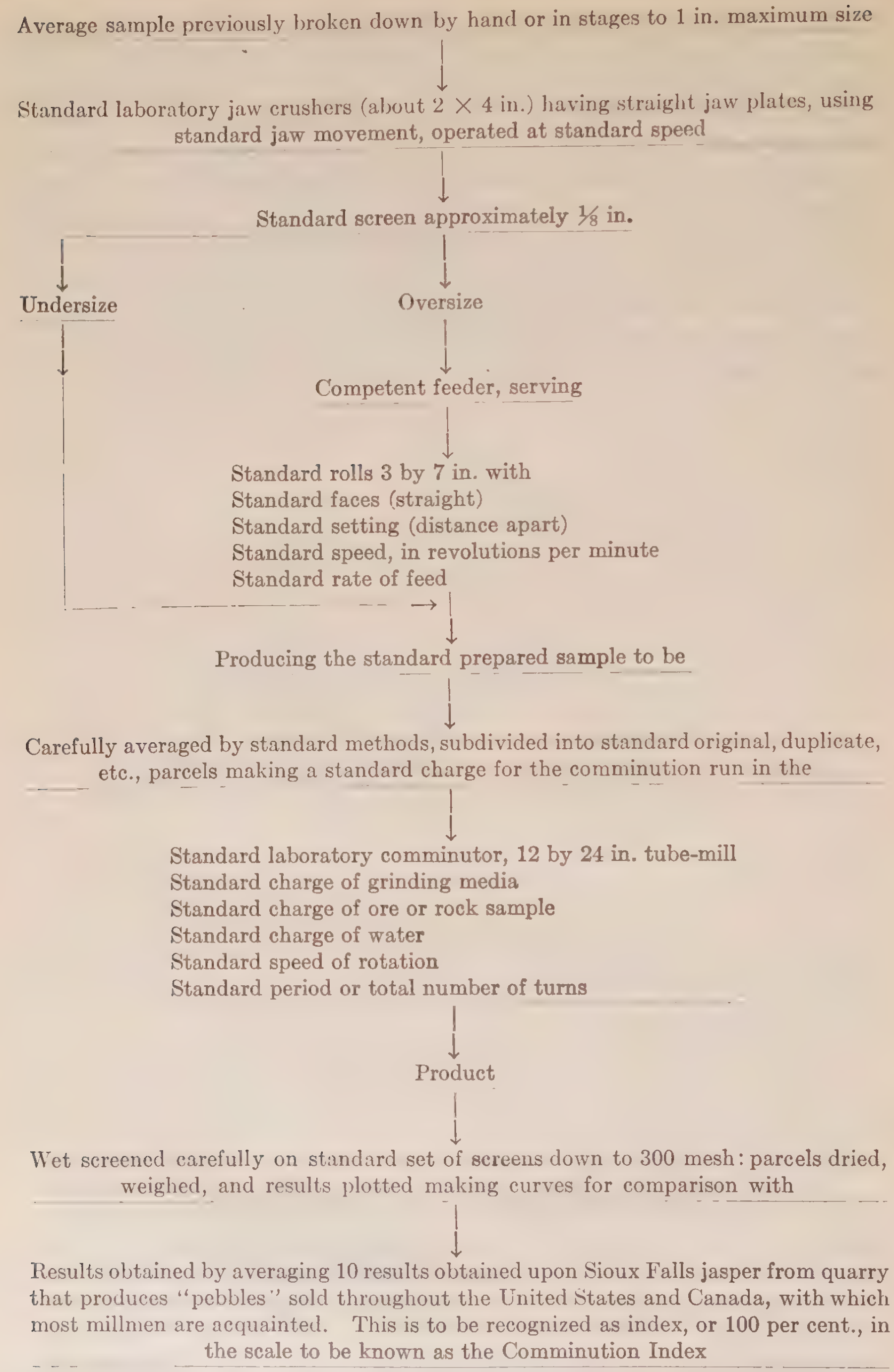


FIG. 4.

Many tests were made. The data sheets are lost but were not particularly interesting because there was little, if any, difference between the results obtained with the various "shapes," until the rods were used, which were by far the best.

Experiments to establish a comparative resistance scale were planned, but were not carried out because getting samples from various mines and districts (to examine their relative comminution resistance) required time, and because experiments with flotation (then new) that were being carried on simultaneously, quickly eclipsed the comminution index part of the project.

In the present state of the art, when we have designing to do that involves fine grinding, we are much handicapped. We obtain as much



Average sample previously broken down by hand or in stages to 1 in. maximum size

Standard laboratory jaw crushers (about 2 × 4 in.) having straight jaw plates, using standard jaw movement, operated at standard speed

Standard screen approximately ⅛ in.

Undersize

Oversize

Competent feeder, serving

Standard rolls 3 by 7 in. with
Standard faces (straight)
Standard setting (distance apart)
Standard speed, in revolutions per minute
Standard rate of feed

Producing the standard prepared sample to be

Carefully averaged by standard methods, subdivided into standard original, duplicate, etc., parcels making a standard charge for the comminution run in the

Standard laboratory comminutor, 12 by 24 in. tube-mill
Standard charge of grinding media
Standard charge of ore or rock sample
Standard charge of water
Standard speed of rotation
Standard period or total number of turns

Product

Wet screened carefully on standard set of screens down to 300 mesh: parcels dried, weighed, and results plotted making curves for comparison with

Results obtained by averaging 10 results obtained upon Sioux Falls jasper from quarry that produces "pebbles" sold throughout the United States and Canada, with which most millmen are acquainted. This is to be recognized as index, or 100 per cent., in the scale to be known as the Comminution Index

FIG. 5.—PROPOSED FLOW SHEET FOR STANDARD TESTS ON WHICH TO BASE A COMMINUTION INDEX.

information as we can by visiting the mine or quarry and then depend on our experience to make a more or less satisfactory guess as to the proper machines and their arrangement. This method does well enough for breaking and crushing, as in preparing railroad ballast, concrete aggregate, or coarse crushing stages in milling, but it is extremely poor for problems involving fine grinding, because the difficulties multiply in proportion to the degree of fineness to which the ore must be pulverized and we have at present little on which to found judgment. If we could send a representative sample of the material to be handled to a laboratory and receive an assay of its comminutability, so to speak; get a report showing the results obtained when using an accepted standard method of reduction to fine size, expressed in convenient curves that would give an index of comminution resistance; and add to this information the visual impression, above referred to, we could with greater confidence proceed with the design of the fine crushing and grinding departments of our plants.

The flow sheet shown in Fig. 5 illustrates a scheme of treatment that, I believe, could be relied on.

Ball Paths in Tube-mills and Rock Crushing in Rolls*

By H. E. T. Haultain† and F. C. Dyer,‡ Toronto, Ontario

(Canadian Meeting, August, 1923)

There has been much written on ball-mills, but no small amount of the literature is simply the expression of individual opinion without sufficient data. This is no doubt due to the complexity and obscurity of the phenomena involved. The outstanding paper is by E. W. Davis,[1] superintendent, School of Mines Experiment Station, University of Minnesota, which is based on a large amount of data from careful experiments with full-size apparatus, together with some mathematical philosophy. The entire philosophy is summed up in the equations given; but the present writers question the foundation upon which this philosophy is based.

Encouraged by the work of Davis, the authors of this paper continued work they had been doing along this line. At first, photographs were taken of the ends of rods in a small laboratory rod mill. The mill was tilted slightly toward the feed end and the discharge end was removed. The rods remained in the mill long enough to permit good observation and good photography.

Later, a mill 24 in. inside the stepped lining was constructed. It consisted of two sheets of plate glass, ½ in. apart, set in a cast-iron ring. It was driven by a variable speed motor and was connected to an accurate speedometer (magneto and voltmeter type). In place of balls or rods, cast brass disks were used. These were ⅜ in. thick and varied in diameter from 2 in. down to ¾ in. Four small projections on each side prevented the disks sticking to the glass when water was used; sometimes steel balls ⅜ in. in diameter were added to the load of disks. There was also constructed a 6-in. mill with corrugated linoleum lining and containing a dry load of various sizes of steel balls. The mills were so connected to the motor that the small mill made two revolutions to one of the large mill.

Many runs were made with a great variety of materials, including crushed marble, seeds of many kinds, and many mixtures (some wet and some dry). In a test with small, hard, white peas, two of the photographs show the Davis effect at the Davis speed, but the authors fail to get any such effects with disks or balls at these speeds.

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* This paper was illustrated by slow-motion pictures.

† Professor of Mining Engineering, University of Toronto.

‡ Assistant Professor of Mining Engineering, University of Toronto.

[1] Fine Crushing in Ball-mills. *Trans.* (1919) **61,** 250.

Experiments were then conducted with mills 6 in. and 3 in. in diameter. The mills were of wood, with the surface roughened by coarse sandpaper; the material was crushed marble, about 30 mesh, with the fines screened out. Many experiments were made, but the Davis effect was not obtained with the Davis speed. When sand, galena, and shot were used, the sand and galena gave identical curves; the speed (revolutions per minute) varies inversely as the square root of the diameter of the mill.

When the 6-in. mills contained 0.4, 0.3, and 0.2 load, the Davis effect was obtained at speeds considerably higher than the Davis. When the mill was run at Davis speed with marble crushed to 60, 120, and 200 mesh, the Davis effect was obtained with the 120-mesh.

When the 6-in. mill lined with corrugated linoleum is run at the Davis speed with 60-mesh marble, the grains carried upward by the corrugations travel the Davis path for that speed but the others do not. Evidently, the path of the grain depends on the character of the material and the character of the path depends on the amount of slip.

Through the courtesy of Clifford Sifton and Filmcraft Ltd., a movie film was taken of the 2-ft. mill in action, using an ultra rapid camera at the rate of 120 exposures per sec. These are shown on the screen at the standard rate of 16 per sec., thus slowing up the speed to about one-seventh. This film shows plainly the slippage, or adjustment of position, as between the disks and the lining, and as between different layers of disks as they rise in the circular path. The disks were painted white with a black radial line. This line enables one to see clearly the rotation of the disks relatively to each other in this adjusting of position. The film also shows, to some extent, the effect of segregation, and how, at some speeds, the hollows of the steps or waves in the lining may be filled with particles of ore. It shows the effect of eddies in the pulp. It shows surges in the movement of the disks, these surges being apparently due to irregular slippage. It shows the great activity at the toe of the cascade, where reversal of direction takes place. It shows much more of the free-fall parabolic paths with the balls in the small mill than with the disks in the larger mill.

These pictures are a beautiful aid to the visualization of what may take place in a ball-mill, but they are not a perfect or complete representation of what happens inside either a ball-mill or a rod-mill. The disks are neither rods nor balls. They do not have so many points of contact on each other as do balls, and consequently may be more ready to slip. They do not carry up the ore and water as do rods, because the ore and water slip down between the disks and the glass sides.

Despite this high-speed movie, and the hundreds of photographs and thousands of visual observations, actual milling conditions have not been truly seen. We can only reason by analogy and by comparison. The movie shows conclusively the large amount of slip with disks and more slip with larger disks than with small. The sand photographs show the effect of slip between the grains or layers of sand, apart from lining slip. In none of the experiments do we get away from slip, except with certain selected fine-grained material, so fine that the particles adhere rather than slide. When shot was used with the roughened, but not stepped or corrugated, lining, the slip was evidently very great, but when a corrugated lining was used, a lining that would definitely lift a layer of shot, the Davis effect was approached at the Davis speed.

If there were no slip between layers, the other balls would follow close behind and there would be a hole in the center as in the Davis diagram. Davis explains the absence of the hole in his sand pictures as due to the effect of *interference*. The shot shows that it is clearly due to slip. Considering layer by layer of balls, the outer layer cannot slip on account of the corrugations, but the next layer slips somewhat on the first layer. The slip between each succeeding pair of layers increases still more until we get to a neutral layer that is not sure whether it is rising or falling or stationary. No matter what may be the nature of the steps or corrugations, or whether there is only a light load in a 6-in. mill or the heavy load in a 6-ft. mill, there must be slip or adjustment between balls in the circular rising paths. The line joining the centers of adjacent balls passes through an angle of more than 90° in reference to the line of action of gravity. The corrugations *may* hold the outer layer of balls without slip, but there will be increasing slip with the other layers. Any slip or adjustment in the outer layer will be augmented in the others.

Of all the varying conditions existing in ball-mills, slip is probably the least determinable, the most varying. The only sure thing about it is that it is always there.

In addition to the effect of slip on ball paths, segregation seems to be of importance. Where there is much free parabolic fall (Davis effect), the large balls go to the center, where they get very little chance for work. The ore particles would tend to go to the periphery. (How important may be the thickness of pulp in this connection?)

The writers would urge a more definite understanding as to the use of terms applied to these ball paths, especially in regard to the word "cascade," which is sometimes applied to the free parabolic path and sometimes to the balls rolling down over each other. There seem to be five or perhaps six definite zones in the ball paths. Zone No. 1 is the zone of the circular path. No. 2 is the zone of the parabolic path, or free fall, on which Davis lays so much emphasis; No. 3 is the zone of cascade; No. 4 is the zone of impact; No. 5 is the zone of reversal of direction. Between zones 1 and 3 there may be a neutral zone No. 6. Each one of these zones will have a larger or smaller part in the actual crushing, but this paper is not concerned with crushing effects.

In conclusion, this paper is a further illustration of the complexity of the subject. Their procedure and technique have features of novelty, but the writers do not claim to have produced any philosophy that has not been mentioned before. There have been so many suggestions and conjectures as to what goes on inside the mill that probably everything that can be imagined has been brought forward. The writers hope, however, that their work may help to clear up the ideas in the minds of some men, and thus prepare the way to further advance.

This paper represents a large amount of work done by the writers and their students. About 500 photographs were taken in addition to the movie films. On the strength of this, the writers venture to express certain conclusions that they draw from their work:

1. Next to speed, the most important factor in determining the paths of balls or rods in tube-mills is slip.

2. The amount of slip depends on several variables (the size of ball, the nature of the lining, the character of the pulp, and other factors) but slip is always present in practice.

3. In some mills (in many mills), the effect of slip may be so great as to eliminate all free or parabolic fall at ordinary mill speeds.

4. There is always a continuous change occurring in the relative positions of the balls in the upward or circular path.

5. At the speeds adopted in practice, there is more of the cascade effect than of the free-fall or parabolic-path effect.

6. In tube-mills, the paths of the balls and of the ore particles are affected by segregation. This seems to be due entirely to size and is but little affected by differences in specific gravity. The action is practically the same as that taking place in a talus or rock dump where the smaller pieces settle through the spaces between the larger, and the large pieces roll.

7. At low speeds, the small sizes are the middle; at high speeds they tend to go to the periphery.

8. The best mixing seems to be at the highest speed attainable without the parabolic path. If there is much of the parabolic path zone present there may be distinct segregation of the larger balls toward the center and of the smaller balls and of the ore toward the periphery.

Note the difference between rod-mills and ball-mills in this respect. Segregation takes place between different sized rods just as between balls, but the ore particles do not get the same chance to settle down through the rods as they do through the balls.

DISCUSSION

E. W. Davis, Minneapolis, Minn. (written discussion).—While I think that the authors' idea in making these motion pictures was to show the great difference between theory and practice in the operation of ball-mills, after a careful study of the pictures, I am convinced that they really show how closely theory and practice coincide. In the first three films, which the authors were kind enough to send to me for study, the theoretical and actual ball paths are very far apart. Both the authors and I thought that this discrepancy could be accounted for, to some extent at least, by the fact that the ball-mill contained no ore, and therefore the slippage between the balls and the lining was excessive. They therefore obtained a picture showing the effect of the addition of quartz to an operating mill, with the very satisfactory results here shown. To demonstrate exactly the relation between the theoretical and the actual ball paths, an apparatus was arranged at the Mines Experiment Station of the University of Minnesota so that the individual pictures on the film could be thrown upon a white-paper screen for sufficient time for the operator to mark on the screen the location of some of the balls. The next picture was then shown and the location of the same balls again noted. By continuing this process, it was possible to trace the paths of travel of the balls and to have definite curves on the paper chart, when finished, that could be compared with the theoretical curves mathematically derived and shown in the paper[2] referred to by the authors.

When making these charts, it was found possible, after a little practice, to follow many of the balls through a large part of their journey in the mill; but below the pulp line they were often lost. These charts therefore show the actual paths through which a number of the balls passed. In some cases the position of the ball in each photograph is shown, while in other cases curved lines are drawn showing the paths. On each chart, an area was mapped showing the location in the mill in which the balls were in contact with and moving relative to the lining of the mill or other balls; this is, of course, the only portion of the mill in which crushing occurs. These mapped areas were shaded on the charts, and the reproductions of these charts therefore indicate not only the paths through which the balls actually traveled, but also where the grinding took place. In order that the theoretical and actual ball paths may be quickly compared, the theoretical paths, as mathematically derived, were superimposed on the charts; they are shown as dash lines.

The first chart, Fig. 1, was made at the beginning of the film and shows the small balls at a period when the mill contained practically no quartz. All the balls are very near the inner paths of the theoretical curves. At the top of their paths, the balls are far below the theoretical points from which they should start to fall freely. As a matter of fact, they are not falling freely, but are rolling and bouncing over one another most of the distance toward the bottom of the mill. The shaded area indicates the zone of marked agitation, where the balls change their directions of travel.

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[2] *Loc. cit.*

In this zone, wholly below the pulp level, most of the crushing is done. The distortion in the pulp line in the mill indicates the position where most of the balls are falling. In this chart the pulp line shows very little depression, except near the center of the mass.

The second chart, Fig. 2, was made after part of the quartz had been added to the mill. The theoretical and the actual paths coincide much more closely than in the preceding chart. The balls rise nearly to the theoretical point at the top of their paths from which they should fall freely. Their paths are, however, too nearly vertical to coincide exactly with the theoretical paths. The pulp line is considerably depressed at a point outside the center of the mass, indicating that a greater number of balls are falling freely through the pulp than when the mill contained less quartz.

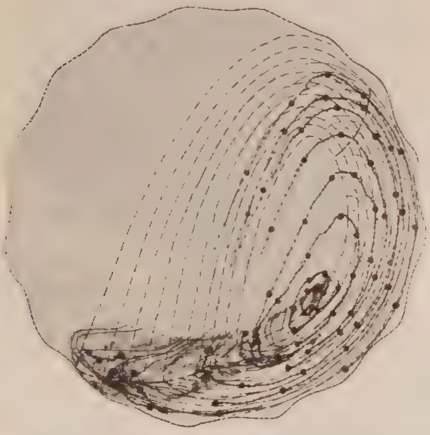

Fig. 1.—Chart made at beginning of film, showing small balls when mill contained no quartz.

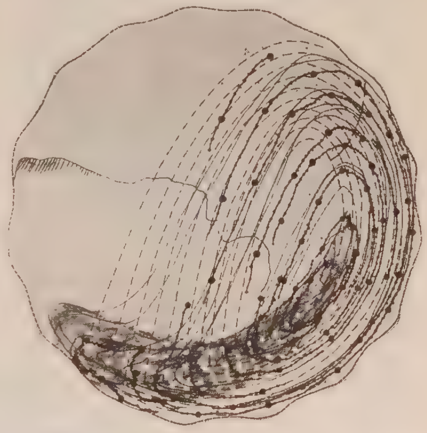

Fig. 2.—Chart showing small balls after some quartz had been added to charge.

The third chart, Fig. 3, was made, near the end of the film, after all of the quartz had been added. There is still considerable difference between the actual and the theoretical paths but the balls are falling freely and their paths more nearly coincide with the theoretical. They still show a tendency to crowd toward the center of the mass, caused by slippage between the balls near the top of their circular paths, which causes interference and prevents them from being projected out beyond the mass in a free-falling curve. This agitation near the upper end of the circular paths, being above the pulp line, is probably largely wasted work as little grinding can take place at this point. The pulp line is depressed still farther from the center of the mass, indicating that a larger number of the balls are falling freely through the pulp.

The fourth chart, Fig. 4, was made at the beginning of the film, showing the mill operating with large balls and containing no quartz. As in the first chart, the balls did not rise to a point in the mill from which they should theoretically begin to fall freely. Most of the balls are not thrown out sufficiently far to clear the mass. Interference, caused by the slippage at the top of the circular paths, is partly responsible for the discrepancy between the theoretical and actual curves.

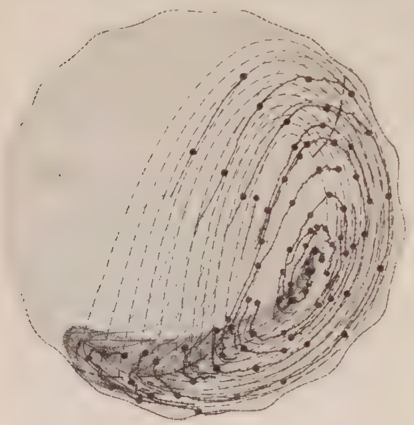

FIG. 3.—CHART MADE NEAR END OF FILM, SHOWING SMALL BALLS AFTER ALL THE QUARTZ HAD BEEN ADDED TO CHARGE.

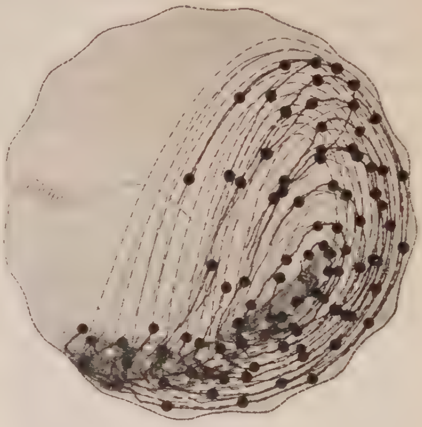

FIG. 4.—CHART MADE AT BEGINNING OF FILM SHOWING LARGE BALLS WHEN MILL CONTAINED NO QUARTZ.

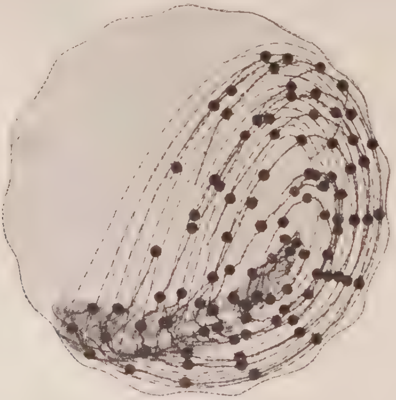

FIG. 5.—CHART SHOWING LARGE BALLS AFTER QUARTZ HAS BEEN ADDED TO CHARGE.

The fifth chart, Fig. 5, shows the condition in the mill after quartz has been added to the charge of large balls. The theoretical and actual curves check much more perfectly. The balls do not rise quite as high as they should, but their paths coincide very well with the theoretical paths. Crowding toward the center is still apparent, but a very much larger proportion of the ball charge is falling freely than with less quartz in the mill.

An attempt was made to measure the amount of slippage between the balls and the lining of the mill. A picture from the film was projected upon a blank paper, as before, and the location of a ball and a point on the lining of the mill were marked. This process was continued through the circular path of travel of the ball; it is, therefore, possible to see exactly how far the ball slipped behind the original point on the lining of the mill. A number of these slippage charts were made; three are shown in Fig. 6. In the first case, the mill contained no quartz; in the second case, some of the quartz had been added; and in the third case, all of the quartz had been added. With no quartz in the mill, the slippage amounted to 8.9 per cent.; after adding part of the quartz, the slip had decreased to 1.5 per cent.; and after all of the quartz had been added, the slip was practically zero.

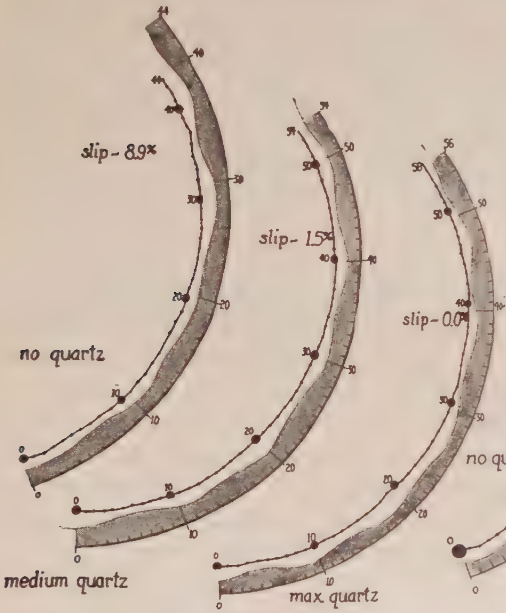

Fig. 6.—Slippage of small balls with varying amounts of quartz in charge.

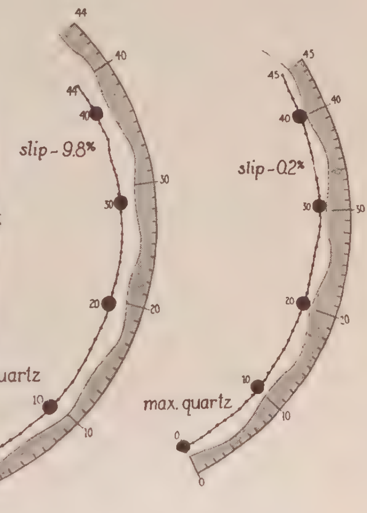

Fig. 7.—Slippage of large balls with no quartz and with maximum amounts of quartz in charge.

This same condition is evident in Fig. 7, which shows the conditions with large balls. With no quartz in the mill, the slip was 9.8 per cent.; and with all of the quartz in the mill it was again nearly zero. It is therefore apparent that the discrepancy between the theoretical and actual curves, when using little or no quartz in the mill, is caused largely by the fact that the balls slip on the lining. As in practice the ball-mill is never operated with no ore, the slippage on the lining of an operating mill is very small.

Conclusions.—The theoretical speed at which the material in a ball-mill should begin to cling to the lining of the mill and revolve with it is computed to be 54.19 r.p.m. for a 2-ft. mill. In the early films, taken with only water in the mill, the speed meter indicated that the cling speed was above 60 r.p.m. However, in the later film in which quartz was added to the mill, the cling speed is 54 r.p.m., as nearly as can be observed on the meter. In this respect, at least, theory and practice coincide.

By adding quartz to the mill, the pictures show that the authors have eliminated practically all slippage between lining and the balls or between the balls themselves during their travel through the circular path. However, when the balls approach the top of the circular paths some slippage occurs, which appears to account entirely for the difference between the theoretical and the actual ball paths. This slippage also appears to decrease with the addition of quartz to the mill; possibly, if sufficient quartz had been added, this slippage could have been eliminated and the check between the theoretical and actual curves made more perfect.

Personally, I am very much indebted to the authors for the demonstration they have made. I have been criticized for publishing the paper referred to for the reason that it was too theoretical and had no relation to the actual operation of a mill. The theoretical analysis of the operation of a ball-mill, like the theoretical efficiency of a steam engine, is 100 per cent. We never expect to attain this degree of refinement and the calculation serves only to show how closely we are approaching ideal conditions. The authors' work has indicated that the theoretical and actual operation of ball-mills are very close together. In some cases they coincide much more closely than would naturally be expected.

I wish to acknowledge the kind and able assistance of Prof. Peter Christianson of the School of Mines of the University of Minnesota in preparing these charts and photographs.

H. E. T. Haultain (author's reply to discussion).—We are delighted that Mr. Davis has been willing and able to make such a close analytical study of these films; we feel greatly indebted to him. That they permit of it increases the importance of the method and we hope that the films may be placed at the disposal of others who may be willing to study them seriously

We have not yet succeeded in photographing conditions as they actually exist within the operating tube-mill, nor is this likely to be accomplished. We make various approximations. There is still room for individual interpretation of results. Mr. Davis concludes that these films "indicate that the theoretical and actual operation of ball-mills are very close together." As there are many theories, presumably Mr. Davis refers to his own.

These films show that, with quartz present, the ball paths are closer to Mr. Davis' calculated paths than when no quartz is present, which simply means that there is less slip. Without doubt, Mr. Davis' beautiful mathematical parabolas are an approximation of the actual paths of balls that have been near the lining, particularly if the lining is new, but I believe most mill men will find considerable dissimilarity between the quartz films and Mr. Davis' theoretical picture. Two points are definitely different—the absence of the central hole and the presence of the very turbulent toe. How much parabolic free fall may exist depends on the amount of slip. Mr. Davis' theory took no cognizance of slip. More or less slip, however, is always present in all of our films, either between the balls and the lining or between different layers of balls. For a part of their upward path there may be no slip where the ball is in actual contact with the corrugated lining, but even this is true only for a part of the upward path. It is more noticeable in the films with the larger balls, because these balls, being 1 in. in diameter, become more or less wedged between the end plates, which are only 2½ in. apart. This condition does not obtain in regular practice. Hard gritty quartz in high dilution will tend to diminish slip. What will be the effect on slip of a very schistose ore? What will be the effect of a very thick pulp? Theory would probably say, slip.

Mining Methods at Bawdwin Mine

By A. B. Calhoun, Bawdwin, Upper Burma

(Canadian Meeting, August, 1923)

These mines, which belong to the Burma Corporation, Ltd., formerly a London company now incorporated in Rangoon, Burma, are situated in the semi-independent state of Tawng-Peng, one of the small divisions comprising the northern Shan states and erroneously known as part of Upper Burma. Bawdwin is approximately 23° 6′ N. latitude and 97° 20′ E. longitude, 450 miles north of Rangoon, 169 miles northeast of Mandalay and 50 miles south and west of the Province of Yunnan, China (see Fig. 1).

Although the mines were worked by the Chinese during the Ming Dynasty, 1412 A. D., the most extensive operations took place between 1796 and 1851, when the mines contributed largely to the silver market of China. During the reign of Tung Chik, 1868 A. D., they were abandoned partly because of the Mohammedan rebellion in Yunnan, which made life and property insecure, but largely because of the difficulty of operating the mines on account of water and poor ventilation. Since that time practically nothing was done, although the Burmese kings are said to have made several sporadic attempts to work the mines, until 1891 when Europeans were attracted by the great slag dumps (assaying about 40 per cent. lead) which the Chinese had left after extracting the silver.

After a railroad and smelter had been built and 200,000 to 300,000 tons of slag smelted for lead, exploration work was started and old workings cleaned out in order to locate the remnants of the orebody that it was supposed the Chinese had left. After two years of most discouraging work, the remains of a large orebody was discovered by the Dead Chinaman Tunnel or what is the 171-ft., or No. 2 level adit. From then on the development was rapid and today the Chinaman is considered one of the largest high-grade silver-lead-zinc orebodies in the world. The total ore reserves on Jan. 1, 1922, were:

	Tons	Silver		Lead		Zinc		Copper	
		Oz. per Ton	Ounces	Per Cent.	Tons	Per Cent.	Tons	Per Cent.	Tons
Lead-zinc.....	3,954,677	22.9	90,736,179	26.1	1,032,764	18.4	726,841	0.47	18,850
Copper ore...	335,681	23.2	7,755,845	12.8	42,983	7.7	25,819	11.0	37,082
Total......	4,290,358	23.0	98,492,024	25.1	1,075,747	17.5	752,660	1.3	55,932

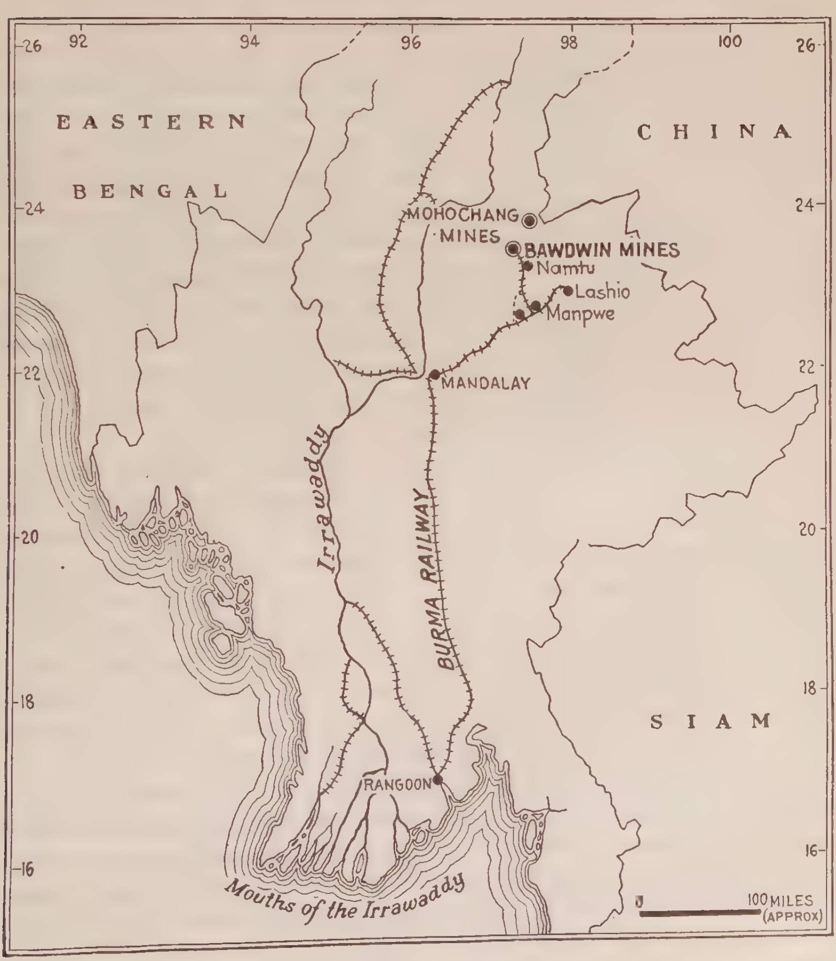

FIG. 1.—BURMA.

Of this, only a little over one-tenth is designated as probable ore. No mineral with a value of less than 20 per cent. combined lead and zinc is considered commercial ore unless it is part of the 335,681 tons of copper ore that averages silver 23.2 ounces, lead 12.8 per cent., zinc 7.7 per cent., copper 11.0 per cent.

The mineral lease comprises an area 5 miles long by 2 wide and covers a period of 30 years from Jan. 1, 1920. The royalty payable to the government is 2½ per cent. of 30 per cent. of the gross value of the metal contents of the ore. The company, as reorganized in Rangoon, has a total authorized capitalization of 20,000,000 shares at Rs.* 10 each, of which 13,541,682 shares have been issued. There is also a first mortgage of 8 per cent. convertible debenture stock of £1,000,000 at Rs. 10 to the pound.

GEOLOGY

The rocks at Bawdwin are of two classes: volcanics (comprising rhyolite tuffs, breccias and flows) and unfossiliferous sediments comprising quartzite, sandstone and shales, which are probably Ordovician. Rhyolite tuff (the ore-bearing formation) forms a wide band running northwest and southeast that has been exposed by the erosion of the overlying sediments. Through this band the main ore fissure passes longitudinally and forms the lode by metasomatic replacement of the rhyolite tuff by ascending ore-bearing solutions. Replacement has taken place parallel to the strike of the fissure, as shown by the laminated structure of the ore and its dimunition in value and density as it passes into low-grade ore, mineralized material, and finally barren tuff on approaching the east, or foot-wall, side.

In the Chinaman lode, there is a well-defined hanging wall, which makes a sharp demarcation between ore and waste; but there is no such foot-wall, the limit of the orebody in that direction is considered to be the limit of what is defined as commercial ore. On some levels of the Chinaman section, the solid sulfide is 50 ft. wide for over 1000 ft. along the strike and in places has reached a width of 140 ft., see Fig. 2. The faulted portion of the Chinaman lode is known as the Shan and is much narrower. This narrowing is partly accounted for by the character of the rock. In the Chinaman the fissure cuts through coarse rhyolite tuff with large feldspar crystals, which were easily dissolved by the ascending solutions making the rock more porous and favorable for the deposition of a large orebody, while in the Shan the tuff is more compact, siliceous, and fine-grained and consequently gives a clean-cut narrow fissure vein. The orebody has a tendency to finger out and become much smaller at the outcrop, as compared to the body below.

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* One rupee = 29 cents.

The ore is an intimate mixture of galena and sphalerite and in many places also of chalcopyrite, although the latter is often found in parallel bands alongside the former as pure unmixed chalcopyrite. The mixture of galena and sphalerite contains approximately 1 oz. of silver for every per cent. lead; it is generally considered that the silver accompanies the galena, but the ore is so complex and the crystals so intimately intergrown that it is most difficult to make a first-class separation of the metal by mechanical means. Taken as a whole, the southern end of the Chinaman section predominates in zinc-lead ore; the middle in more equal quantities of both; and in the northern end the zinc is partly replaced by copper. In practically all sections, the ore along the hanging wall is the highest grade, with the lead predominating over the zinc, but toward the center or the foot wall the zinc contents increase until, in many sections, the zinc predominates. Still farther toward the foot wall the ore becomes lower grade and below what is classified as ore until it is only mineralized; the lead however predominates and often is found as pure crystals of galena. These conditions, together with the following, have been instrumental in determining the method of mining:

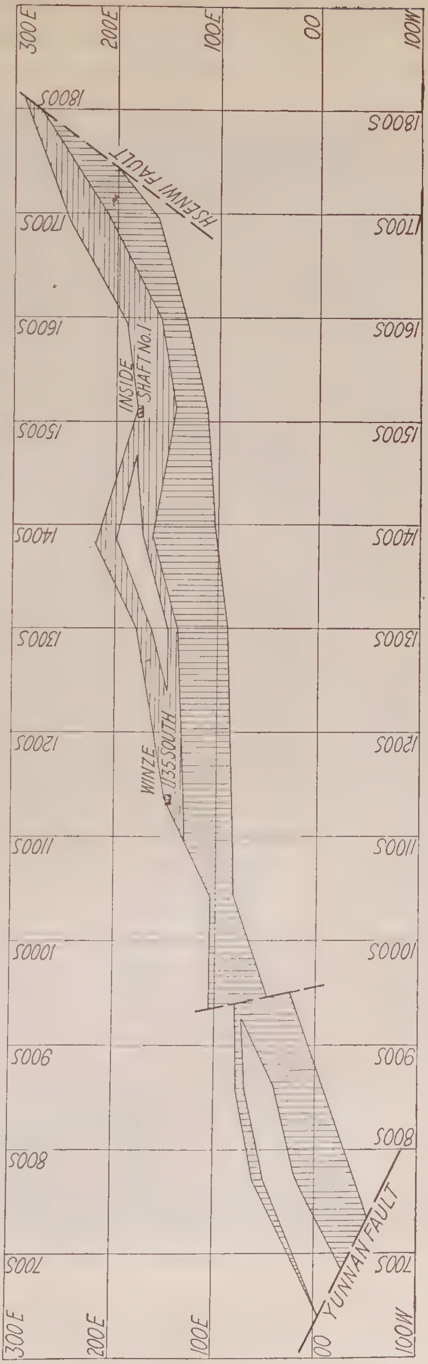

FIG. 2.—PLAN OF CHINAMAN OREBODY ON 430-FT. LEVEL.

(*a*) The Chinese worked the mine from the surface to 50 ft. below the 171-ft. level for the silver alone and consequently did not want lead ore, high in zinc or low-grade ore; so that there is a large remnant of

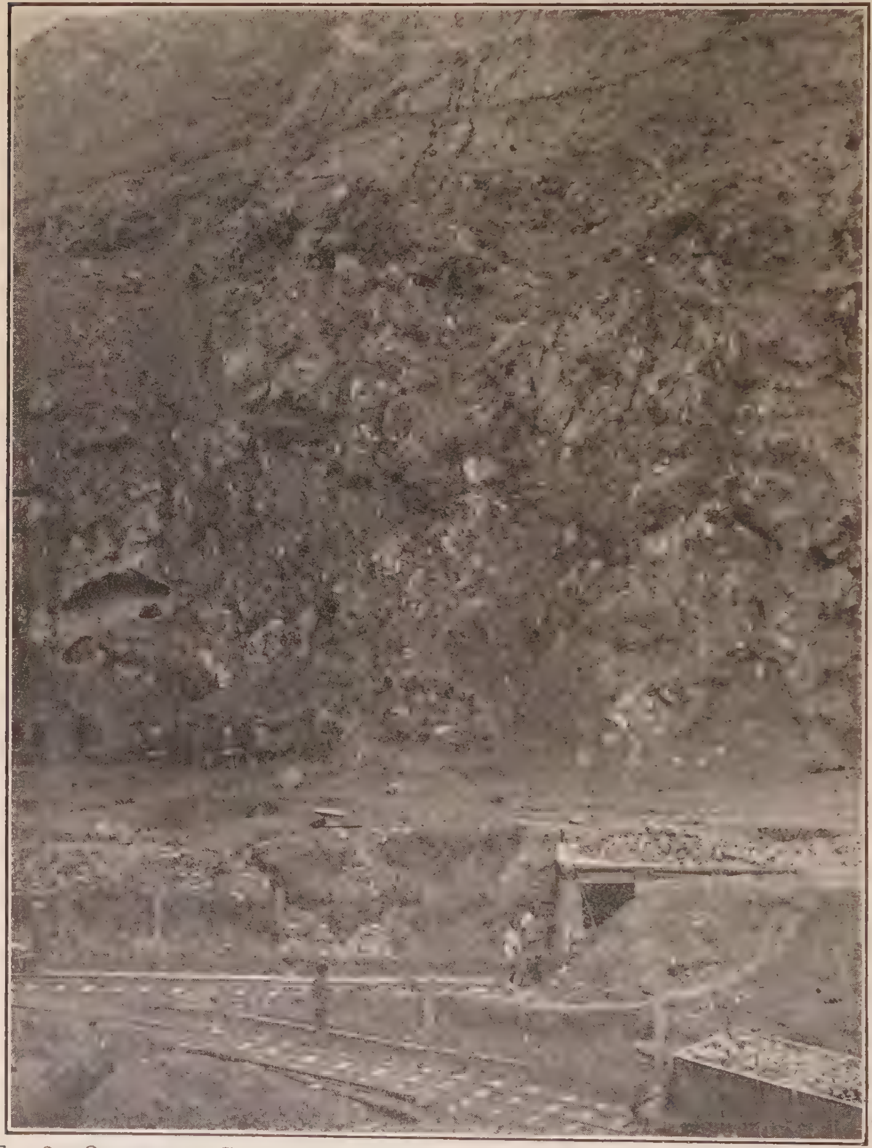

Fig. 3.—Outcrop of Chinaman lode showing old Chinese adits and workings.

the orebody on the upper levels mixed with Chinese filled stopes and workings. This remnant must be preserved for future requirements, as a considerable part of it is not commercial ore today.

(*b*) The banded structure of the ore; for instance, there are bands of chalcopyrite running parallel to the lead-zinc ore. These bands must be mined separately in order to save the copper.

(*c*) There is only one pronounced wall, the hanging wall. The ore starts from it as a solid mass high in lead and gradually passes through zinc and chalcopyrite bands into low-grade ore and, finally, into mineralized ground. That is, there is no definite stoping limit toward the foot-wall side except what is arbitrarily fixed as the limit of commercial ore (20 per cent. combined lead-zinc with whatever silver it contains). Later, this arbitrary value may be lowered and another 2,000,000 tons of low-grade ore added to the reserve. Any stoping method must take this low-grade material into consideration and not leave it in such condition that it cannot be economically mined.

General Conditions That Influence Methods of Mining

The country is very rough. There are no roads; the only means of travel are a 2-ft. gage railway with steep gradients and sharp curves, and trails along the ridges of the hills.

From November until May (the dry season), this region is healthy and enjoyable, but during the wet, or the remaining, months the climate is depressing. Many employees suffer from malaria and kindred ailments. The coolies, especially those from the high cold regions of China, suffer considerably from malaria as they do not take the necessary precautions.

The upper levels of the mines, on account of the many workings, including all the old Chinese stopes, drives, etc., in which a large surface of ore is exposed to oxidation, are quite hot. During the wet season, water percolates through and increases the oxidation and also the humidity of the air.

The rain falls in heavy showers; the fall amounting to about 70 in. in six months, making the mine quite wet and the ground heavy. It is also difficult to keep the railway free from landslides and washouts.

The mine is situated 3100 ft. above sea level, which is very favorable to the health. The air is clear and the dense fogs found 1000 to 2000 ft. lower do not prevail.

Water, Fuel, Timber, and Other Supplies

Good water, in abundance, is piped to all bungalows and to all the levels in the mine. This is a treat to one who has lived in the tropics as, in most cases, water must be boiled to insure no contamination. Partly furnished bungalows, electric lighting, and water are supplied to all Europeans and fuel to all employees. This fuel is brought in by pack mules a distance of 8 miles and costs Rs. 14 per stack (108 cu. ft.). Local timber, which is hardwood and heavier than water, is shipped in by rail from the company's timber reserves, about 20 miles away, at the following rates:

Mine logs	Rs. 41 per ton (50 cu. ft.)
Local sawn timber	Rs. 90 per ton (50 cu. ft.)
8 in. by 2 in. by 6 ft. lagging	Rs. 74 per 100 pieces
8 in. by 3 in. by 7 ft. 4 in. lining boards	Rs. 120 per 100 pieces

The laggings and lining boards are hand cut in the jungle by Chinese. Bamboo, 4-in. and 5-in., used for lining the sides of stopes preparatory to filling, cost Rs. 5 per 100 laid down in Bawdwin. High-grade ingyin timber is shipped from Mandalay at the rate of Rs. 140 per ton (all charges). This dense, hard, local timber, which a person might think would last for years in the mine, has a short grain and breaks without any warning under no great load. Some of this timber breaks sharply across the grain and looks as though it had been sawn through. It lasts, on the average, in this humid atmosphere about three years. In all permanent workings, the timber is now creosoted, which we believe will double its life. It is difficult to frame, and a nail cannot be driven into it unless a hole is first bored.

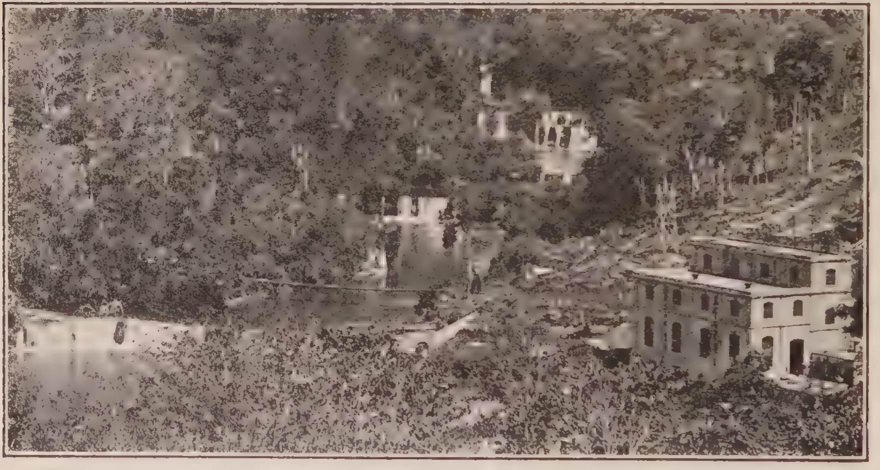

FIG. 4.—MANSAM HYDRO-ELECTRIC POWER PLANT.

Coke and coal are brought from India across the Bay of Bengal and then carried by rail 450 miles; coal costs at the mine, approximately, Rs. 56; coke, Rs. 79.

Fuel oil, from the Yenangyoung oil fields in Burma, is shipped from Rangoon and costs Rs. 0.3 per gallon.

Electric Power

The corporation has installed a 3000-hp. hydro-electric plant at Mansam Falls, which supplies the mine, smelter, and mill. A Diesel oil-engine plant capable of delivering 1800-hp. is available for emergencies. From the hydro-electric plant, the power is transmitted 38 miles over high-tension wires at 33,000 volts to the mine substation where it is stepped down to 550 volts for mine use. The mine uses 270,000 kw.-hr. per month and is charged Rs. 0.0106 per kw.-hr. An additional expense on the low-tension side, of motor operators, repairs, and maintenance, increases the cost to Rs. 0.0142 per kw.-hr., which is the charge applied to the mine operation. No depreciation charges are included in this cost of power, this being cared for by the head office.

FIG. 5.—GOTEIK BRIDGE, ON ROAD TO MANDALAY. THE STEEL STRUCTURE IS BUILT ON A NATURAL ROCK BRIDGE OVER A DEEP GORGE, MAKING IT, PROBABLY, THE HIGHEST BRIDGE OVER WATER IN THE WORLD.

Labor

Chinese from Yunnan, China, form the nucleus of the underground labor with a number of Gurkhas (from the independent state of Nepal, Upper India) and a small number of other Indians. Practically no natives of Burma work in the mine. On the surface, there are men from nearly every neighboring country. Chinamen from Shanghai, Canton, Indo-China, and neighboring Chinese provinces, many of whom cannot converse with one another; Chinese-Shans from the northern Shan states; men from all parts of India even Nepal and Afghanistan. Men of all castes, creeds, and languages. The laboring class comprises men from nearly all neighboring countries; the Burman makes a good clerk and a fair carpenter, but will not do hard labor. The other natives of the country will not work in the mine nor do manual labor on the surface under any conditions; they prefer to cultivate their small farms though they eke out a bare existence.

All technical supervision must be supplied by men from England, Australia, or the United States. It will be many years before the Anglo-Indians or Indians can be trained for this technical work and take the place of a European in the more responsible positions.

The Chinese and the Chinese-Shan coolies are the best laborers and when one makes his home in Burma, taking his family there, he makes an excellent miner. However, a large part of the labor is seasonal. The Chinaman prefers to come over at the beginning of the dry season and goes home before the wet season in order to plant his rice. It often takes 20 days for him to walk in from remote places in Yunnan. He is honest, compared to the Indian coolies, and is easy to handle if treated with reasonable justice. On an average, two Chinamen will do the work of one European, and one Chinaman that of two Indians. As a large part of the labor is seasonal, it is not efficient. The turnover is very large, averaging 11 per cent. for six months. During March, as many as 20 per cent. of the men leave for their homes. It is most discouraging to teach a coolie to run a drill or timber a stope, just to have him leave when he becomes proficient.

Pumping

The pumping problem is simple, as practically all the water is handled by a gravity system through three adits—No. 1 level, No. 2 level, and No. 6 level—except the development below the latter which is opened by two winzes. These require two sinking pumps (a No. 7 and a No. 9 B Cameron) to pump the water to the No. 6 level. This No. 6 level adit, known as Tiger tunnel, takes care of 40,000 gal. per hour.

Accessibility

The property is accessible from Rangoon, the main port in Burma, by the Burma Railways (a meter-gage line) that connects with the company's private road at Nam-Yao. From this point, a 2-ft. gage line leads over the mountains to Bawdwin, a distance of 45 miles.

Exploration, Sampling, and Estimating Methods

Exploration is today chiefly done by drives, crosscuts, and winzes. In the upper part of the orebody, down to 50 ft. below the 171-ft. level, the orebody is riddled with Chinese adits and workings; but as they are small, tortuous, and filled with mud and water, new adits, drives, and crosscuts were made to explore the orebody. A small amount of diamond drilling was done; but on account of the character of the ore, the drillers were continually losing the water or the bit so that it is more satisfactory to develop by actual workings. For development purposes, drives were originally run in the ore parallel to the strike and crosscuts put out to the extremities of the ore every 100 ft. with occasional rises in ore from these crosscuts to the upper level. During the later development of the mine, the drives were made in the country rock foot wall and crosscuts run to and through the ore. These drives require no timber and being outside and parallel to the orebody become the main extraction drives for the present stoping method. Two winzes, or inside shafts, were sunk from the No. 2 adit level to open up and develop the levels below and also facilitate the driving of No. 6 adit from each end.

Sampling

Each crosscut is sampled every 5 ft. on both sides and the average of the two taken for calculations. Drives and raises are also sampled every 5 ft. but these samples are not included in the calculations of the orebody but are used to prove the continuity of the ore and grade only. As most of the ore is friable and soft, it is not difficult to cut a small trench of even width with hammer and moil. However, there are occasional patches of low-grade siliceous ore, which require the aid of a Jackhammer. Samples, when assayed, are entered in an assay ledger, one page for each crosscut, drive, etc. Using a width factor of 1 for every 5 ft. of sample and carrying a running total of width factor times ounces or per cent. facilitates the averaging of the values of any drive, crosscut, etc., for its entire length, or between any two points (see Fig. 6).

Estimating

After several levels had been opened up by crosscuts and raises every 100 ft., it was apparent from the character of the orebody—that the size and grade were fairly uniform from level to level and crosscut to crosscut and the continuity and transition of values from higher to lower grade were so gradual—that it was permissible to estimate the orebody on the crosscuts alone; that is, the width and value of the ore in the various crosscuts on any particular level (with the aid of geological evidence in faulted country) would give the area and grade of that level. For the purpose of estimating, complete assay plans are made of each level and by referring to the assay ledger the average of all 5-ft. samples (both sides) is plotted and the values marked on the map, showing location, width, and assay values. Values of samples from all crosscuts, drives, etc., are shown on this assay plan for every 5 ft., or less, notwithstanding the material is mineralized only or waste. For estimating ore and tonnage, a graphic system is employed and another plan, called the estimating plan, is made of each level (see Fig. 7).

Burma Mines, Ltd.—

Location Refers to Distance from *Center Line of North Drive 168 East*

Date of Sampling	Sample No.	Width	Location	Side	Width Factor	Total W. F.	Silver			Lead		
							Ounces per Ton	Oz. × W. F.	Total	Per Cent.	Per Cent. × W. F.	Total
			0 - 2½′				1.0	0.5				
			0 - 2½′				0.5	0.2				
23 Oct. 1917	6775		2½ -5′	N.	0.5	0.5	0.8	0.4	0.4	*nil*		
	6776		2½- 5′	S.	0.5	0.5	1.1	0.5	0.5	*nil*		
	6777		5 -10′	N.	1	1.5	0.9	0.9	1.3	1.4	1.4	1.4
	6778		5 -10′	S.	1	1.5	1.0	1.0	1.5	Tr.		
	6779		10 -15′	N.	1	2.5	0.9	0.9	2.2	1.8	1.8	3.2
	6780		10 -15′	S.	1	2.5	0.4	0.4	1.9	0.2	0.2	0.2
26 Oct.	6827		15 -20′	N.	1	3.5	1.2	1.2	3.4	0.7	0.7	3.9
	6828		15 -20′	S.	1	3.5	1.3	1.3	3.2	2.8	2.8	3.0
9 Nov.	6959		20 -25′	N.	1	4.5	4.0	4.0	7.4	7.7	7.7	11.6
	6960		20 -25′	S.	1	4.5	3.3	3.3	6.5	6.6	6.6	9.6
11 Nov.	6979		25 -30′	N.	1	5.5	4.6	4.6	12.0	7.8	7.8	19.4
	6980		25 -30′	S.	1	5.5	1.9	1.9	8.4	5.5	5.5	15.1
12 Nov.	6999		30 -35′	N.	1	6.5	2.5	2.5	14.5	4.5	4.5	23.9
	7000		30 -35′	S.	1	6.5	2.0	2.0	10.4	5.2	5.2	20.3
18 Nov.	7052		35 -37′	N.	0.4	6.9	2.8	1.12	15.6	5.0	2.0	25.9
	7053		35 -37′	S.	0.4	6.9	34.1	13.64	24.0	45.8	18.32	38.6
19 Nov.	7075		37 -40′	N.	0.6	7.5	20.6	12.36	28.0	30.2	18.12	44.0
	7076		37 -40′	S.	0.6	7.5	40.9	24.54	48.6	43.8	26.28	64.9
	7077		40 -45′	N.	1	8.5	38.3	38.3	66.3	40.2	40.2	84.2
	7078		40 -45′	S.	1	8.5	31.1	31.1	79.7	33.8	33.8	98.7
21 Nov.	7098		45 -50′	N.	1	9.5	31.4	31.4	97.7	39.9	39.9	124.1
	7099		45 -50′	S.	1	9.5	23.5	23.5	103.2	36.0	36.0	134.7
25 Nov.	7203		50 -55′	N.	1	10.5	18.5	18.5	116.2	25.5	25.5	149.6
	7204		50 -55′	S.	1	10.5	19.7	19.7	122.9	30.6	30.6	165.3
27 Nov.	7219		55 -60′	N.	1	11.5	25.9	25.9	142.1	34.8	34.8	184.4
	7220		55 -60′	S.	1	11.5	20.8	20.8	143.7	29.6	29.6	194.9
	7221		60 -65′	N.	1	12.5	27.8	27.8	169.9	38.8	38.8	223.2
	7222		60 -65′	S.	1	12.5	35.1	35.1	178.8	45.8	45.8	240.7
29 Nov.	7235		65 -70′	N.	1	13.5	40.7	40.7	210.6	49.0	49.0	272.2
	7236		65 -70′	S.	1	13.5	21.9	21.9	200.7	32.8	32.8	273.5
1 Dec.	7244		70 -75′	N.	1	14.5	37.0	37.0	247.6	41.8	41.8	314.0
	7245		70 -75′	S.	1	14.5	23.2	23.2	223.9	47.4	47.4	320.9
4 Dec.	7248		75 -80′	N.	1	15.5	15.5	15.5	263.1	34.7	34.7	348.7
	7249		75 -80′	S.	1	15.5	2.0	2.0	225.9	3.8	3.8	324.7
18 Dec.	7294		80 -83′	N.	0.6	16.1	2.6	1.6	264.7	8.3	5.0	353.7
	7295		80 -83′	S.	0.6	16.1	0.5	0.3	226.2	Tr.		324.7
31 Dec. 1921	10675	60″	80 -85′	N.			2.0			3.0		
31 Dec.	10676	60″	80 -85′	S.			1.4			2.6		
31 Dec.	10677	60″	85 -90′	N.			2.2			3.2		
31 Dec.	10678	60″	85 -90′	S.			1.0			2.0		
31 Dec.		60″	90 -95′	N.								
31 Dec.		60″	90 -95′	S.								

Fig. 6.—Page from sampling ledger.

Mining Department.

Level *653′*

Working *West Cross-cut 941′ South*

Zinc			Copper			Average Values					Description	Miscellaneous
Per Cent.	Per Cent. × W. F.	Total	Per Cent.	Per Cent. × W. F.	Total	Ag	Pb	Zn	Cu			
1.5	0.7											Drive at 565. S. S. of X-cut.
0.7	0.4											Drive at 570. N. S of X-cut.
0.9	0.4	0.4	*nil*								Country	
0.7	0.3	0.3	*nil*								Country	
1.5	1.5	1.9	*nil*								Country	
1.0	1.0	1.3	*nil*								Country	
0.8	0.8	2.7	*nil*								Country	
0.8	0.8	2.1	*nil*								Country	
1.0	1.0	3.7	*nil*								Country	
2.3	2.3	4.4	*nil*								Country	
3.5	3.5	7.2	*nil*								Mineralized	
1.5	1.5	5.9	*nil*			3.6	7.1	2.5	*nil*		Mineralized	
1.5	1.5	8.7	*nil*								Mineralized	
1.0	1.0	6.9	*nil*			3.2	6.6	1.2	*nil*		Mineralized	
0.8	0.8	9.5	*nil*								Mineralized	
1.9	1.9	8.8	*nil*			2.2	4.8	1.3	*nil*		Mineralized	
5.3	2.12	11.6	*nil*			2.8	5.0	5.3	*nil*	35′ –36′	Mineralized	
14.0	5.6	14.4	4.4	1.76	1.76						Mineralized	
29.0	17.4	29.0	0.2	0.12	0.12						Ore	
20.0	12.0	26.4	0.8	0.48	2.24	31.6	39.2	21.9	0.8	36′ –40′	Ore	
31.9	31.9	60.9	Tr.		0.12						Ore	
29.9	29.9	56.3	Tr.		2.24	34.7	37.0	30.9	Tr.		Ore	
6.9	6.9	67.8	Tr.		0.12						Ore	
10.1	10.1	66.4	Tr.		2.24	27.4	38.0	8.5	Tr.		Ore	
11.3	11.3	79.1	Tr		0.12						Ore	Ore 36′–77½′. 27.8, 37.6 14.7, 0.5.
11.9	11.9	78.3	0.5	0.5	2.74	19.1	28 0	11.6	0.2		Ore	
13.2	13.2	92.3	0.2	0.2	0.32						Ore	
11.6	11.6	89.9	*nil*		2.74	23.3	32.2	12.4	0.1		Ore	
14.0	14.0	106.3	0.6	0.6	0.92						Ore	
11.8	11.8	101.7	0.9	0.9	3.64	31.4	42.3	12.9	0.7		Ore	
15.2	15.2	121.5	1.3	1.3	2.22						Ore	
9.7	9.7	111.4	Tr.	Tr.	3.64	31.3	40.9	12.5	0.6		Ore	
15.8	15.8	137.3	2.1	2.1	4.32						Ore	
7.9	7.9	119.3	Tr.		3.64	30.1	44.6	11.8	1.0		Ore	
6.6	6.6	143.9	*nil*		4.32	15.5	34.7	6.6	*nil*	75′ –77½′	Ore	
1.4	1.4	120.7	*nil*		3.64	2.0	3.8	1.4	*nil*	77½′–80′	Mineralized	
4.7	2.8	146.7	Tr.		4.32						Country	
0.6	0.4	121.1	*nil*		3.64	1.5	4.1	2.6	Tr.		Country	
3.0			*nil*									
1.5			*nil*									
3.0			*nil*									
1.0			*nil*									

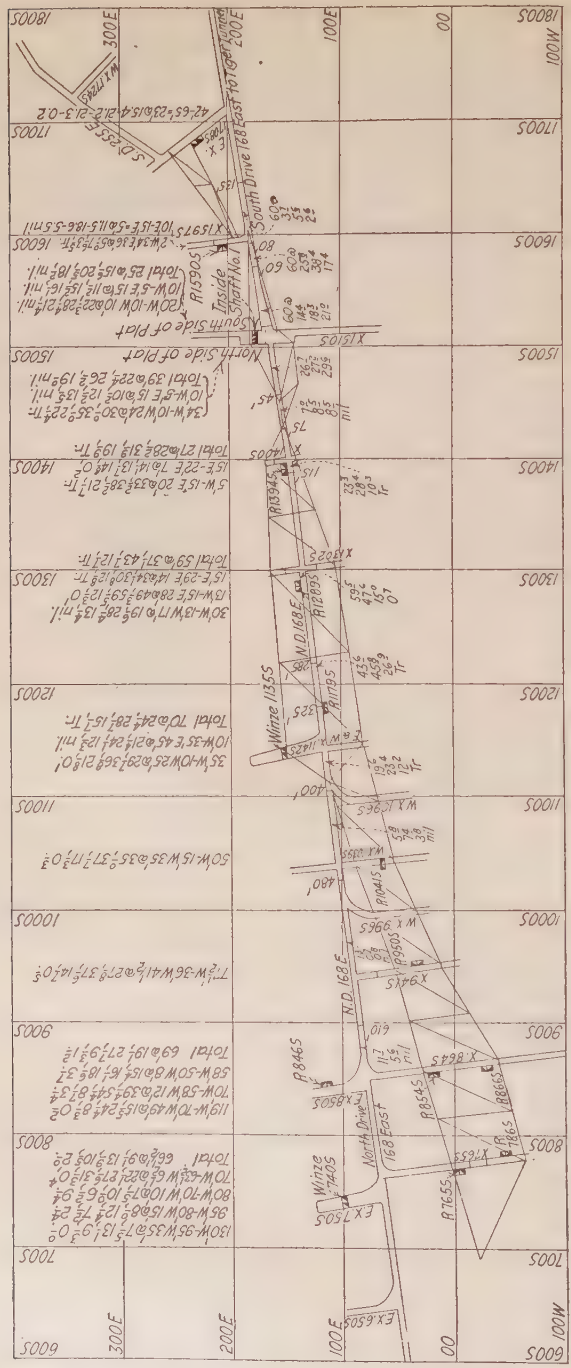

FIG. 7.—ORE ESTIMATE PLAN.

The next question to consider, and possibly the most important, is: what is commercial ore now and what will be commercial ore during the life of the property? This question is most difficult to answer because the mine is in a state of development and no one knows its life, or only approximately so, and furthermore due consideration must be given to the possibility of new metallurgical treatment and the fluctuation in price of the products. However, the result is only an estimate based on available data. The advisers for this mine decided on 20 per cent. combined lead and zinc, in whatever combination, with its accompanying silver; or in the case of copper ore, any of the combination containing lead and zinc with 3 per cent. copper and its accompanying silver.

By referring to the assay ledger, in preference to the assay plans, the limits of commercial ore can be marked on each crosscut. Due consideration must be given to the stoping method to be applied in order to know just what samples to include. Lines are then drawn on the plan from crosscut to crosscut enclosing the body of commercial ore. The area is divided into triangles, as shown in Fig. 7 and, with the aid of a scale, the base and altitude of each triangle is measured. The areas of triangles in the adjoining half of the space between the crosscuts (that is nominally 50 ft. on either side) are multiplied by the average value of the crosscut and give as a product ounces times square feet and per cent. times square feet. The sum of all ounces times feet and per cent. times feet divided by the total of all the areas between the crosscuts, which is the area of the level, gives the average value of that level. By calculating the Pb in form of PbS, Zn in form of ZnS, and the Cu in form of $CuFeS_2$, the total percentage of sulfides is obtained and the remainder is considered quartz. By deducting 5 per cent. for voids and using the specific gravity of the various minerals comprising the ore, the number of cubic feet per ton is obtained. The sum total of all the areas divided by this factor gives the number of ton-feet. Ton-feet multiplied by the distance up and down, which is generally half way to the next level, gives the total tonnage and value of the level; see Fig. 8. As the ratio of cubic feet to tons depends on the relative specific gravity, porosity, and moisture of the ore, the average value of the crosscuts as calculated is not exact, as they were calculated on volume and not tonnage. It is impossible and too expensive to get the specific gravity of each sample, so that this factor cuts down the metal contents per actual tons of ore 7 to 10 per cent.

Ore is classified as proved and probable. Proved is considered ore of practically no risk in estimating its value or continuity. As development work has shown the orebody to be fairly uniform from level to level with no abrupt changes in grade or size, as proved by raises every 100 ft., all ore between lines drawn from the boundaries of proved ore on one level to those above or below is considered proved; also ore 25 ft. above the

ORE RESERVE CALCULATIONS

653-FT. LEVEL. CHINAMAN LODE SECTION

Crosscut		Area	Silver		Lead		Zinc		Copper	
			Ounces per Ton	Ounces × Feet.	Per Cent.	Per Cent. × Feet.	Per Cent.	Per Cent. × Feet.	Per Cent.	Per Cent. × Feet.
1708		880	15.4	13,552.0	21.2	18,656.0	21.3	18,744.0	0.2	176.0
1597		537	11.5	6,175.5	18.6	9,988.2	5.5	2,953.5		
S. of 1510		1,150	15.6	17,940.0	20.5	23,575.0	18.2	20,930.0		
N. of 1510		1,690	22.4	37,856.0	26.2	44,278.0	19.0	32,110.0		
1400		2,261	28.2	63,760.2	31.9	72,125.9	19.9	44,993.9		
1302		7,259	37.1	269,308.9	43.7	317,218.3	12.7	92,189.3		
1142		7,869	24.4	192,003.6	28.7	225,840.3	15.7	123,543.3	Tr.	
1039	$\frac{70 \times 29}{2} + \frac{70 \times 26}{2} + \frac{70 \times 27}{2} + \frac{70 \times 23}{2} =$	3,675	35.0	128,625.0	37.7	138,547.5	17.3	63,577.5	0.3	1,102.5
941	$\frac{72 \times 28}{2} + \frac{72 \times 26}{2} + \frac{76 \times 32}{2} + \frac{76 \times 24}{2} =$	4,072	27.8	113,201.6	37.6	153,107.2	14.7	59,858.4	0.5	2,036.0
854	$\frac{88 \times 34}{2} + \frac{88 \times 28}{2} + \frac{89 \times 33}{2} + \frac{89 \times 36}{2} =$	5,798	19.1	110,741.8	27.7	160,604.6	9.3	53,921.4	1.2	6,957.6
765	$\frac{88 \times 34}{2} + \frac{88 \times 35}{2} + \frac{96 \times 56}{2} =$	5,724	9.1	52,088.4	13.9	79,563.6	10.5	60,102.0	2.0	11,448.0
		40,915	24.6	1,005,253.0	30.4	1,243,504.6	14.0	572,923.3	0.5	21,720.1

Wt-Ft^3

Pb = PbS PbS

30.4 × 1.16 = 35.26 ÷ 468 = 0.0754 per cent. cu. ft. per lb.

Zn ZnS ZnS

14.0 × 1.49 = 20.86 ÷ 250 = 0.0834 per cent. cu. ft. per lb.

Cu $CuFeS_2$ $CuFeS_2$

0.5 × 2.9 = 1.45 ÷ 262 = 0.0055 per cent. cu. ft. per lb.

Quartz

42.43 ÷ 154 = 0.2572 per cent. cu. ft. per lb.

100.00 0.4215 per cent. cu. ft. per lb.

5 per cent. voids 0.4215 ÷ 0.95 = 0.4437 per cent. cu. ft. per lb.

0.4437 × 2240 = 9.94 cu. ft. per ton.

$\frac{40,915}{9.94}$ = 4116 ton-feet.

FIG. 8.

Summary of Ore Reserve Calculations, July 1, 1920

Chinaman Lode Section

Levels	Feet			Ton-Feet	Total Tons	Silver		Lead		Zinc		Copper	
	Up	Down	Total			Ounces per Ton	Ounces	Per Cent.	Tons	Per Cent.	Tons	Per Cent.	Tons
0	25.0	25.0	50.0	1,228	61,400	11.0	675,400	12.9	7,920.6	11.5	7,061.0	2.4	1,473.6
1	50.5	39.5	70.0	4,420	318,240	12.0	3,818,880	14.8	47,099.5	20.7	65,875.7	0.3	954.7
2	39.5	62.0	95.0	6,056	575,320	19.4	11,161,208	24.1	138,652.1	20.9	120,241.8	0.1	575.3
3	62.1	67.1	129.2	8,060	1,041,352	26.1	27,179,287	28.9	300,950.7	18.9	196,815.5	0.7	7,289.5
4	67.1	55.0	122.1	8,179	998,656	32.2	32,156,723	32.3	322,565.9	23.2	231,688.2	0.7	6,990.6
5	55.0	56.9	111.9	5,540	619,926	24.4	15,126,194	26.0	161,180.8	15.9	98,568.2	0.3	1,859.8
6	56.9	25.1	82.0	4,116	337,512	24.6	8,302,795	30.4	102,603.6	14.0	47,251.7	0.5	1,687.6
Total proved					3,952,406	24.9	98,420,487	27.3	1,080,973.2	19.4	767,502.1	0.5	20,831.1
Probable between Nos. 1 and 2 levels				134	12,730	11.5	146,395	11.1	1,413.0	26.5	3,373.4	0.1	12.7
Probable below No. 6 at area and value of No. 6 level.		25 to 100	75.0	4,116	308,700	24.6	7,594,020	30.4	93,844.8	14.0	43,218.0	0.5	1,543.5
Total Chinaman lode proved and probable					4,273,836	24.8	106,160,902	27.5	1,176,231.0	19.0	814,093.5	0.5	22,387.3

Fig. 9.

SUMMARY OF ORE-RESERVE CALCULATIONS OF JAN. 1, 1922

	Total Tons	Silver		Lead		Zinc		Copper	
		Ounces per Ton	Ounces	Per Cent.	Tons	Per Cent.	Tons	Per Cent.	Tons
Total Chinaman lode proved and probable	4,273,836	24.8	106,160,902	27.5	1,176,231	19.0	814,093	0.5	22,387
Total Shan lode, vertical-shaft section proved and probable	570,294	19.3	10,991,091	17.0	96,965	11.0	62,862	6.0	34,280
Total proved and probable before extraction	4,844,130	24.2	117,151,993	26.3	1,273,196	18.1	876,955	1.2	56,667
Extraction	597,682	32.8	19,627,572	34.7	207,168	22.3	133,463	0.4	2,520
Reserve left in mine	4,246,448	23.0	97,524,421	25.1	1,066,028	17.5	743,492	1.3	54,147
Ore stocks	43,910	22.0	967,603	22.1	9,719	20.9	9,168	0.4	1,785
Total ore reserve	4,290,358	23.0	98,492,024	25.1	1,075,747	17.5	752,660	1.3	55,932
Less copper ore as of Jan. 1, 1921	335,681	23.2	7,755,845	12.8	42,983	7.7	25,819	11.0	37,082
Total	3,954,677	22.9	90,736,179	26.1	1,032,764	18.4	726,841	0.47	18,850

FIG. 10.

upper level and 25 ft. below the lowest. Probable ore is considered as containing some risk on account of the lagging behind of development on a level, particularly at the extremities of the orebody. From 25 to 100 ft. below the lowest level is considered probable. Probable ore would also be assumed 50 ft. ahead of the last crosscut, where geological evidence is favorable for continuity. It would be calculated on the value of the last crosscut and on an area equal to a triangle, having an altitude of 50 ft. and a base the width of the ore in the crosscut.

Accuracy of Methods of Estimation

As the mine is still in its infancy, it is not possible to check the method of estimating by actual mill yield plus tailings. In most mines, there is a loss of 10 per cent. or more in actual value. By the present method of estimating, using the algebraic average of a section, the metal contents per actual ton of ore is cut down 7 to 10 per cent. dependent on whether the high-grade or the low-grade predominates. This should take care of any mining loss, and dilution will add to the values recovered as the dilution will be low-grade non-commercial ore, instead of barren waste.

History of Mining Methods

The first method of stoping tried was the common flat-back square-set stope, which was carried very wide and long with a number of ore passes to shovel into. Fortunately, this method was stopped in time, as it is impossible to prevent such a large stope from caving and the cost of keeping so many timbered ore passes is excessive. Furthermore, the filling was laborious and expensive as all waste had to be wheeled and shoveled into position.

A narrow Gilman slice rill stope was tried in the hardest and most favorable ore, with only timber on the sides to confine the waste from the adjacent stopes. This was a failure and a death trap, as the solid sulfide ore (9 and 10 cu. ft. to the ton) would drop without any warning in large masses; and in other places the friable ore would run up in the form of a chimney and cause the stope to cave.

Present Mining Methods and Reasons for Adoption

Stoping

The upper levels of the mine have been riddled by Chinese workings; consequently the ore left is mixed with old filled stopes and gives a low-grade oxidized ore. The company does not care to mine and mill this ore at present as it plans to open-cut and quarry a large portion of it in connection with the regular waste-filling plan now being used to fill the stopes. Any method of mining the ore, must leave these upper levels undisturbed until the company is prepared to handle this ore. This eliminates several systems of stoping, for instance mill hole, top slice, and caving systems, and leaves only some form of a timbered and filled stope. The ore is so heavy and friable that it must be closely timbered to support any opening and the orebody is so wide that it requires mining in sections.

By elimination we have arrived at the square-set system of stoping with all its modifications. A flat-back square-set stope is costly and slow, on account of the laborious work in handling the ore and waste, and is not easy to ventilate. To meet these disadvantages, an efficient combination square-set rill system, of a variable width and length to meet any conditions that arise has been adopted, (see Fig. 11).

To lay out the system to the best advantage for longitudinal stoping, as required by the grades and class of ore, the development drives should be run outside and parallel to the orebody in the hard foot-wall rock and not in the ore, as were some of the earlier ones. At every 100 ft., crosscuts are driven to and through the ore to the hanging wall and continuous raises put up from level to level, and finally to the surface, so that waste may be taken directly into the stopes.

When commencing stoping operations, half way between the main crosscuts, auxiliary ones are put in with a 25-ft. radius curve to the main or extraction drive. The 50-ft. blocks on either side, with the exception of a 12-ft. pillar directly over the crosscut, comprise the stope. These rill stopes have their apexes at the main passes and slope down to the extraction crosscuts. In the early work, no pillar was left at the toe of the two stopes and one ore chute sufficed for two stopes. This toe, or section around the chute, would become very heavy, and it was difficult to prevent this from caving. A 12-ft. pillar is now left between the two stopes and individual ore chutes are carried up. The pillar is mined after the two stopes are finished.

Stopes are carried three or four sets wide, depending on the character of the ground. Four sets is the maximum width that the best ground will stand; and as the ground becomes heavier the stope is brought in to three, and sometimes two sets in exceedingly bad ground, but the length remains the same.

Where the vein is wide, a longitudinal section is first taken along the hanging wall; and when this is completed a second and third section alongside, retreating toward the foot wall. Under this arrangement, the crosscuts and drives are never in bad ground, under worked-out stopes, but in the solid ore or country rock. The part of the crosscut under the completed stope can be filled, thus doing away with all expensive repairs maintaining openings under old stoped areas. In order to get the proper rill and still not make it too steep to climb up, square sets must be of proper size and height. Those at Bawdwin mine are 5½ ft. square and 7 ft. 4 in. high.

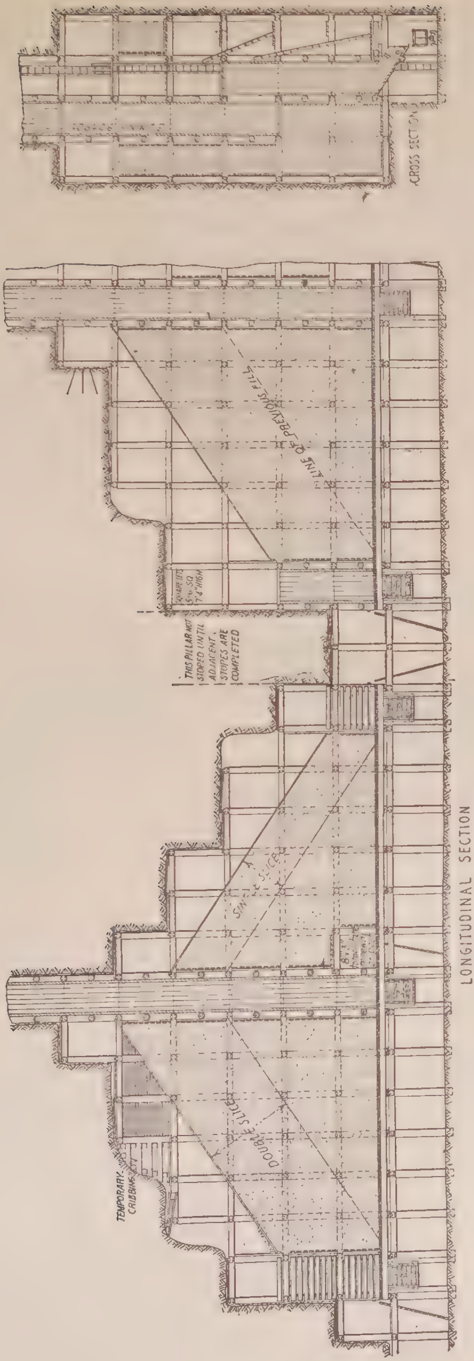

FIG. 11.—SQUARE-SET RILL METHOD OF STOPING.

Manways should always be on the hanging-wall side of chute and not along one side, as is the general custom. This gives access to either stope, from the manway or the chute without interfering with entance to the other. As all ore is dropped to the lowest level or adit, from which it is hauled out by electric locomotives, any system of continuous raises can be used as an ore pass, provided the raise is not being used for passing waste for a stope below. However, it has since been found cheaper to run up continuous untimbered ore passes from level to level in the foot wall when suitable hard rock can be found; this leaves the main passes for waste only. The lower repair cost of maintaining timbered chutes for the soft waste fill only and not for the heavy ore offsets the increased cost of tramming ore on the various levels to the (all rock) ore passes. Waste filling is obtained from the surface by quarrying or mill-holing around the top of these continuous mullock passes placed at intervals of 100 ft. along the strike of the orebody so that the filling can be passed directly into the stopes without any tramming.

Mine Openings, Shafts and Tunnels

The main opening to the mine, as at present developed, is the No 6. level adit, commonly known as Tiger tunnel; see Fig. 12. This is the main haulage and drainage adit. It is nearly 2 miles long and double-tracked from the portal to the inside shaft, a distance of 7400 ft. The

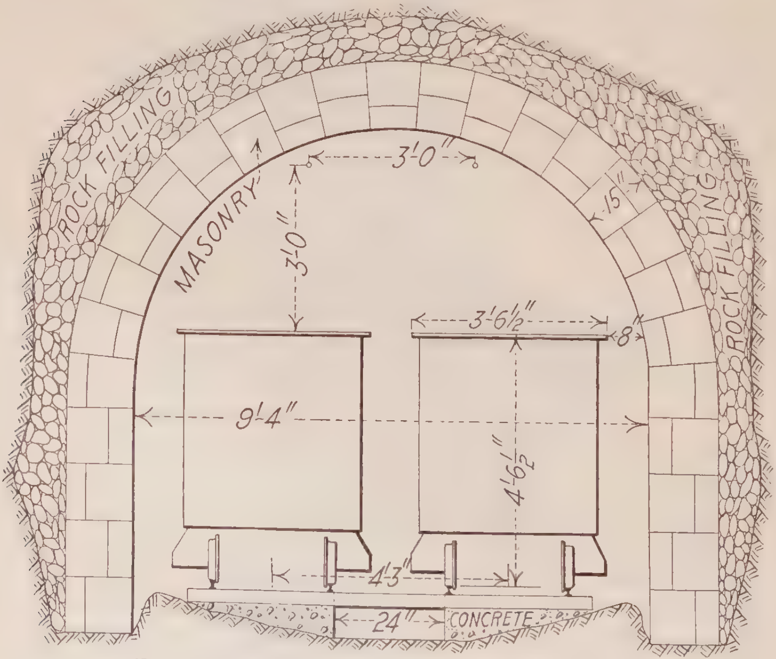

FIG. 12.—CROSS SECTION THROUGH TIGER TUNNEL.

section is 9 ft. wide by 8 ft. high in the clear. The ditch is carried in the middle in the space between the two tracks; long sleepers pass completely over it and support both tracks. The grade of the tunnel started at 0.6 per cent. but was increased to 0.7 per cent., to accommodate the large volume of water passing through the ditch, which had a tendency to silt up on the lower grade. The average flow is 40,000 gal. per hr. but this has been exceeded many times.

The double-track part of the tunnel was run from both ends; it was begun in April, 1914, and finished Sept. 21, 1916. The length of survey was 23,300 ft., consisting of forty-nine readings of the theodolite, exclusive of those in the tunnel itself. Of these readings, three were lines less than 10 ft. in length, four between 10 and 20 ft., and six between 20 and 30 ft. The actual error of closure was 1.26 ft.; giving an error of 1 ft. in 18,500. The distance between the center lines of the two ends at right angles to their length was 0.35 ft. In elevation, the error of closure was 0.35 ft. or 1 ft. in 100,000, the distance leveled being approximately 34,000 feet.

Tiger Tunnel Construction

The work was originally started with an outfit, picked up in the country, consisting of a Class A Sargent straight-line compressor (steam cylinder 22 by 24 in., air cylinder 22¼ by 24 in., giving approximately 960 cu. ft.) and two second-hand boilers run on wood fuel; 2½-in. piston machines were used for drilling. During the first year, on account of lack of organization, poor equipment and raw coolie labor, only 2448 ft. was driven—an average of 204 ft. per month. Work was then reorganized. A new equipment was obtained and eleven trained white men were employed to supervise as follows: One foreman, three shift bosses, one mechanic, one steel sharpener, one combined track layer and pipeman, and four miners. The new equipment comprised the following, with necessary accessories:

1 new Ingersoll-Rand compressor belt-driven,
  Imperial type 10, 950 cu. ft.
1 Garrett semiportable boiler and engine combined.
1 No. 5 Leyner drill sharpener.
6 No. 18A Leyner drills (three machines on a bar),
  replacing the piston machines.
1 British Westinghouse dynamo, direct-current 9 kw.
1 Root blower.

During the following 12 months and 21 days, the outside heading advanced 3814 ft. at the rate of 300 ft. per month, the highest footage for any month being 502 ft. An average month of 300 ft. required daily 227 men, working as follows: 151 underground, 40 mechanical department, 36 surface, all working in four 6-hr. shifts; of this number eight to eleven were white men acting as supervisors. There was 3050 ft. of tunnel timbered. Many large bursts of water occurred, the flow through the tunnel being from 40,000 to 73,000 gal. per hr. The total cost of the 7400 ft. of double-track tunnel amounted to Rs. 731,000, including all charges for labor and supplies and 8 per cent. for depreciation on the capital cost of equipment. As soon as the tunnel was finished the equipment was transferred to the mine account.

All running and heavy ground was timbered, but within a year after the tunnel was completed, the rapid deterioration of the local timber made necessary a large amount of repairs and many caves and obstructions to traffic occurred. It was decided to replace the timber with masonry, as good sandstone could be had on the railway line about ½ mile from the portal. Work was started and a 15-in. wall and arch replaced the timber. Any space above the arch was filled with dry rubble. As a large proportion of the timbered sections was in spiling ground, progress was slow. Only one set of timber could be removed at a time, for the iron forms had to be put in and masonry built around them and allowed to set before the next set of timber could be taken out. Nine steel forms were used 50 ft. or more apart, and an advance of 150 to 200 ft. per month was made. The forms were so constructed and work so managed that there was no serious delay to the mine haulage, and the trolley and lighting wires were never cut. The cost of the masonry was as follows:

	PER FOOT
Quarrying, transporting, and placing stone	Rs. 27.0
Cement ½ bbl. at Rs. 25 per bbl.	Rs. 12.5
Sand 10 cu. ft. at Rs. 14 per 100 cu. ft.	Rs. 1.4
Lime and other supplies	Rs. 1.1
Taking out old timber and masonry forms	Rs. 12.0
Total cost	Rs. 54.0

All ore mined is dropped to this level and hauled out by electric locomotives. Three smaller adits on the upper levels, namely Zero, No. 1 and No. 2 levels, are used for drainage and transportation of supplies. As there are only three levels that have no adits and as no ore or waste is hoisted or lowered in cages, an elaborate shaft and hoist equipment is not required until stoping operations begin below the lowest adit, or Tiger tunnel.

Two inside shafts, one of two and the other of three compartments 4 by 8 ft. and 4½ by 11 ft. in the clear, respectively, are used for hoisting and lowering men and supplies. The 8 by 8-in. timbers in the two-compartment shaft have shown considerable pressure and have been replaced by 10 by 10-in. Mandalay ingyin timber. The framing of the set has also been changed by making a bevel on the inside corners of the set; this prevents the wall plates crushing and splitting at the end plates. These shafts were sunk in the orebody as a means of development before the lower adit was driven, consequently they are only temporary and will be replaced by a large circular shaft 14 ft. in diameter and masonry lined, Fig. 13. This will be located in the hanging wall in country rock and at some distance from the orebody.

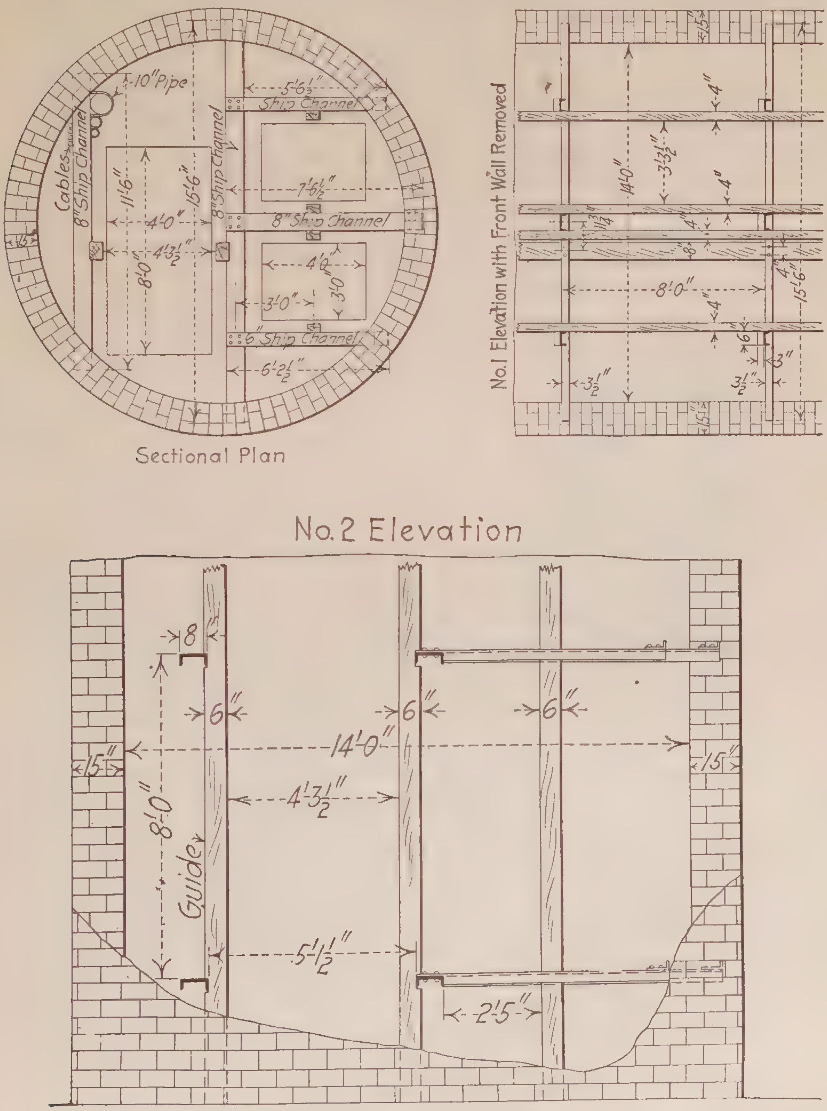

FIG. 13.—PROPOSED CIRCULAR SHAFT.

In sinking the three-compartment shaft, one European was put in charge of the coolie labor and advanced the shaft 30 to 35 ft. per month; 5000 to 5500 gal. of water was pumped per hour. An advance of 33 ft. per month required 461 coolies divided into three shifts.

The cost per foot was:

Labor	Rs. 58.0
Supervision	36.0
Explosives	34.0
Timber	28.0
Pumping	36.0
General expenses	4.5
Engineering	1.5
Mine ventilation	1.0
Repairs and maintenance	29.0
Lighting	8.0
Compressed air	3.0
Hoisting	2.5
Assaying and sampling	2.0
Steel	1.0
Sundries	10.0
Total	Rs. 254.5

Of the local wood in Burma, only ingyin and teak are satisfactory for shaft work and the latter is too expensive. The three-compartment shaft, which is of ingyin timber, has been completed over 6 years and the timber is still in good preservation, although a number of sets have had to be replaced on account of rock pressure. This is partly accounted for by the shaft being wet with acid water impregnated with zinc sulfate and there being a good circulation of air. Timber has been taken out of the submerged lower levels of the Chinese workings that is 50 to 100 years old and in a good state of preservation. When this old timber is brought to the surface, the absorbed water evaporates and leaves a $\frac{1}{8}$-in. coating of zinc sulfate over the entire stick. The smaller shaft, which was sunk about the same time and timbered with local sawn timber, has been practically retimbered twice on account of the rapid decay, as this shaft is dry. The second time 10 by 10-in. Mandalay ingyin timber replaced the local timber and we expect this to remain in good shape for two or three years, but not as long as if it were wet with the acid mine water.

Underground Development

In Figs. 14 and 15 are shown longitudinal sections of the different levels.

Drilling and Blasting

The corporation has standardized on the following drills, for the following reasons:

(*a*) The first ones were purchased about 7 or 8 years ago and tried out with the raw coolie labor; although they are subject to much abuse they stand up well.

(*b*) It has taken considerable time to teach the coolies to operate and repair these particular drills efficiently.

(*c*) On account of the long distance Burma is from the source of supply, a large stock of repair parts is necessary; this is quite an expense when about 100 drills are operating.

(*d*) By using only one type that has been found satisfactory, much labor is saved that would otherwise be wasted on a new type of drill, and the stock of repairs is kept to the minimum.

(*e*) Burma is not the country nor the Chinaman the man to try out new machines; consequently we leave that to others.

The drills in use are: 40 I. R. B. C. 21 stopers, 40 B. C. R. 430 Jackhamers, 13 18A Leyner drifters.

For Jackhamers and stopers ⅞-in. hexagonal hollow and 1-in. cruciform solid steel are used; 1⅛-in. round hollow steel is used for the Leyners. Ordinary cross bits are used with as few changes and as large a clearance as possible, because the coolie persists in running the steel until the machine is entirely cranked out. Drills are tempered by a separate fire by giving them a short heat not farther up than 1 in. from cutting point and then standing them on a grating, resting ½ in. below the surface of a tank of water. Sharpening is done on two Ingersoll-Rand No. 5 Leyner drill sharpeners in conjunction with oil furnaces. The average rock is not hard and requires no special bit; and as the cross bit

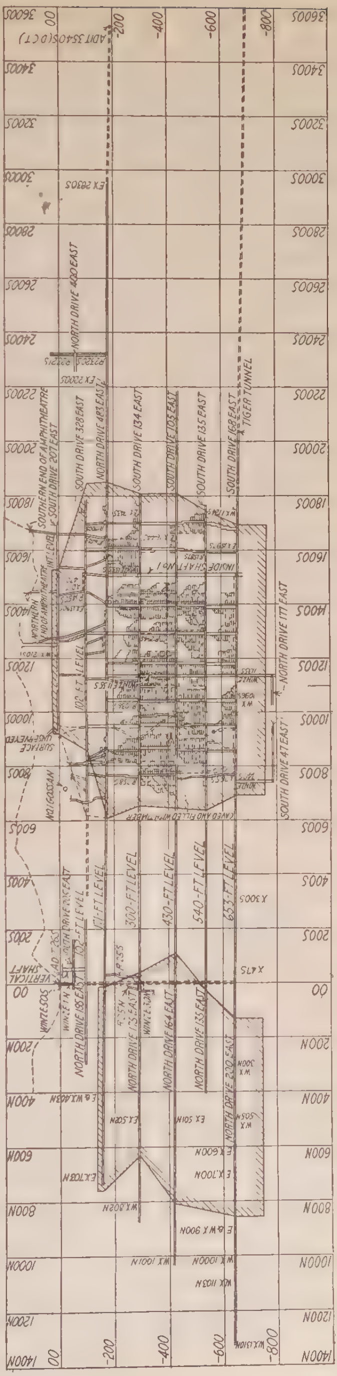

FIG. 14.—LONGITUDINAL SECTION ALONG SHAN AND CHINAMAN LODES.

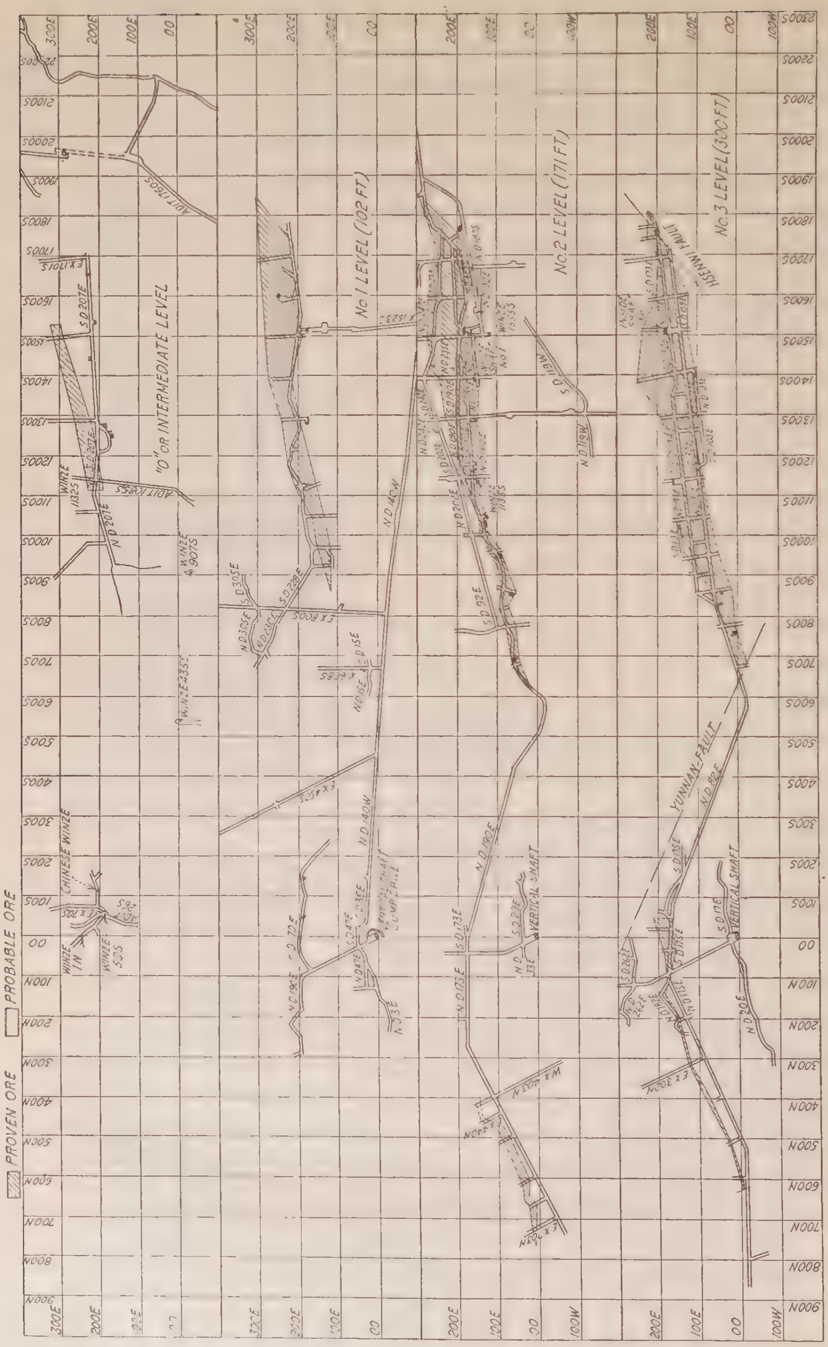

FIG. 15.—PLAN OF LEVELS 0 TO No. 3.

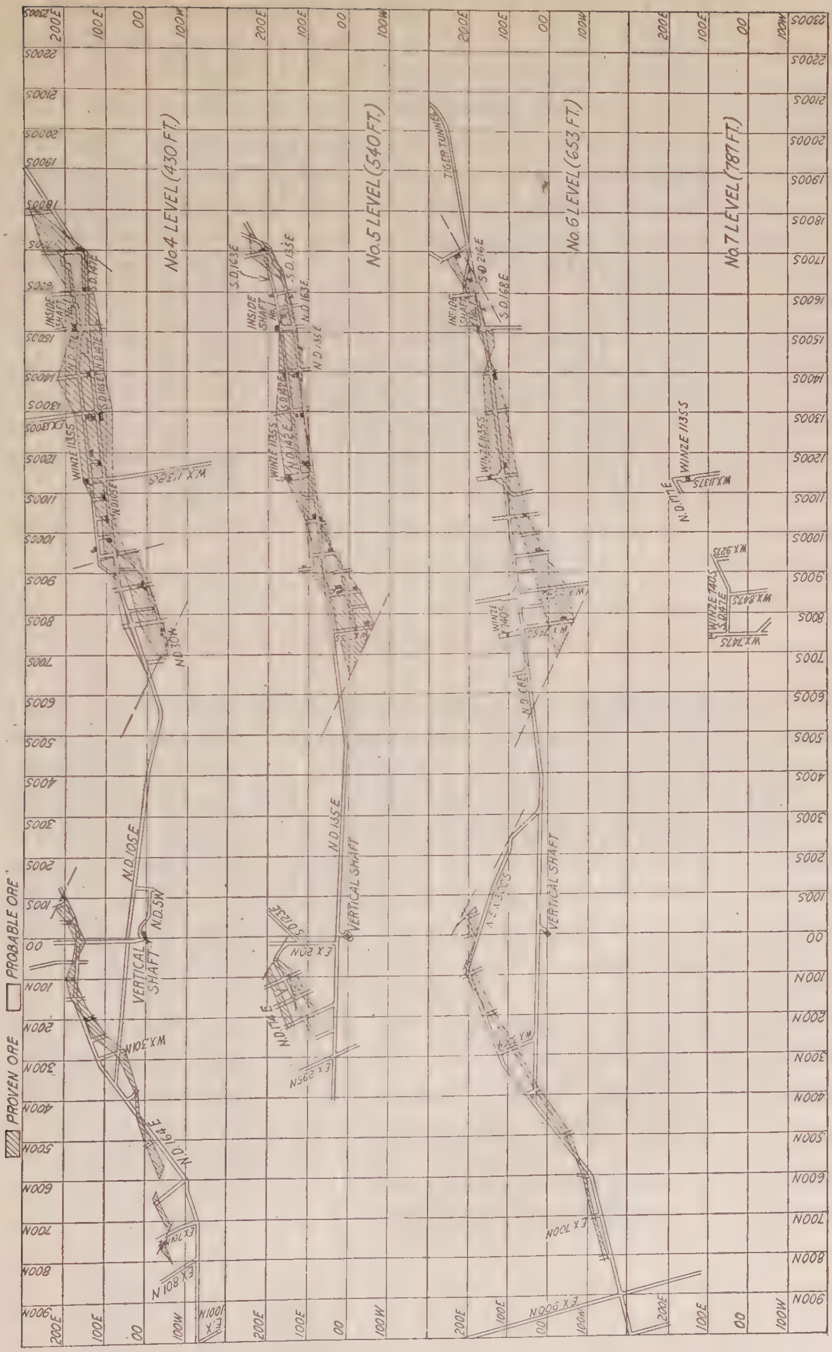

FIG. 15.—PLAN OF LEVELS NO. 4 TO NO. 7.

is the simplest and easiest to sharpen it is used on account of coolie labor. The simplest rounds are used as the placing of holes to the best advantage is the most difficult part of mining to teach the coolie.

Fifty-per cent. Nobels or Curtis & Harveys gelignite is used. This explosive is comparatively safe and only one fatal accident and two minor ones have occurred in the mine during the last 5 years and these were due to carelessness. Gelignite, a modification of blasting gelatine, resists the action of water, is safer than gelatine, and has all the plastic advantages of the latter over dynamite. It is a nitroglycerine compound to which is added nitrocotton, nitrate of potash, and wood meal. It, however, lacks the rapidity of explosion of the dynamite and does not produce the local shattering but has a larger rending action in being slower. Dynamite is more effective for bulldozing, etc., where gelignite is more effective for drill holes. Double tape fuse is used for dry work and gutta percha tripple tape for wet work. No. 7 detonators are used entirely. The coolies have very little trouble in spitting the fuse, as they roll a small piece of gelignite into a small pencil, which is quite effective especially in wet and drafty workings.

Drifting and Stoping

The ordinary methods of driving and timbering are used. In going through running ground, spiling with false set and face boards is used. The latter are, however, made in two pieces. One piece has two bolts and the other a slot to accommodate them. First, room is made for the half with the two bolts and the end shoved in front of the point of the side spiling, while the other end is supported by a temporary brace. Room is then made for the other half, and the board is put in with the slot in line with the two bolts. Large washers are slipped over the bolts and the nuts tightened. This makes a rigid face board that is easy to put in and take out.

For stoping, hardwood is the only available timber. This wood is so hard that nails cannot be driven into it unless the holes have first been drilled. Wire nails are of little value, so cut iron spikes must be used. The timber is heavier than water; consequently every stope is supplied with Holman or tugger hoists. Instead of the ordinary 2-in. lagging generally used to line the inside of a stope preparatory to filling, 4-in. bamboos 11 ft. long are used, much the same as pole lagging.

Timbering

Special timber is framed by hand but all other by machine.

COST OF FRAMING TIMBER

	HAND	MACHINE
Stope post	Rs. 0.563	Rs. 0.292
Stope cap	Rs. 0.563	Rs. 0.292
Stope girt	Rs. 0.313	Rs. 0.292
Shaft set complete	Rs. 15	
Winze set	Rs. 10	
Drive set (2 posts and cap)	Rs. 0.688	
Chute frames (2 posts, 1 cap and 1 center piece)	Rs. 0.875	

All framing is done on the surface and all timber, except that required for stopes, goes to the preserving plant. This consists as follows:

One creosote storage tank 5½ by 12 ft. in diameter.

Two open-treatment steel tanks, one for hot and the other for cold solution, size 5 by 5½ by 12 ft.

Three 1½-ton chain blocks and overhead crawl.

One vertical boiler, 9 hp., 100 lb. pressure.

The boiler is connected to a series of 1-in. pipe coils laid in the bottom of the hot tank, which raises the liquid to 180° or 220°.

The timber is left in this tank for 6 or 7 hr. The heat causes the wood to expand, expelling the air and moisture, and to absorb a small amount of creosote in the cellular spaces. The wood is then put in the cold tank where it is left over night. This causes contraction and a condensation of moisture, and the creosote is absorbed and forced into the wood. The wood is very hard and dense and only a ½-in. penetration is obtained. As most of the timber is thoroughly seasoned, the hot tank is not necessary except for green timber; by leaving the seasoned timber in the cold tank 12 to 24 hr. a penetration of ½ in. is obtained. The process has not been in operation more than 2 years so that we are not able to tell just how much longer life is given to the timber by the treatment. There is reason to believe from the experience of two years that the treatment will at least double the life of the timber, which will bring the chemical life of the timber close to its mechanical life, and that is as far as one is justified in going in expenditure for treatment.

The local Burmese timber is as follows:

Thitsee	3150 lb. per 50 cu. ft.	
Yindike	3000 lb. per 50 cu. ft.	
Thaukkyan	3075 lb. per 50 cu. ft.	
Thumalum	3037 lb. per 50 cu. ft.	
Kadee	2925 lb. per 50 cu. ft.	
Yang	3900 lb. per 50 cu. ft.	
Cedar	1500 lb.	not suitable for mining purposes.
Cotton wood	1440 lb.	

All timber is peeled in the forest, as the space between the bark and wood affords a starting place for fungi and is unsuitable for creosoting. Most of the timber is seasoned by cutting it the previous year and stacking it along the railway line; however, not more than enough for one season should be cut as the timber decays very rapidly in the forest and is badly damaged by ants and borers.

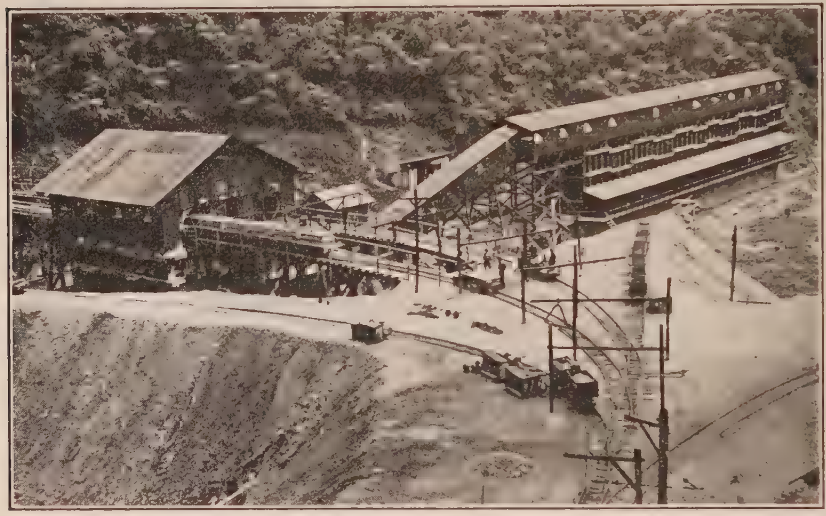

FIG. 16.—TIPPLE PLANT AND ORE BINS AT TIGER CAMP.

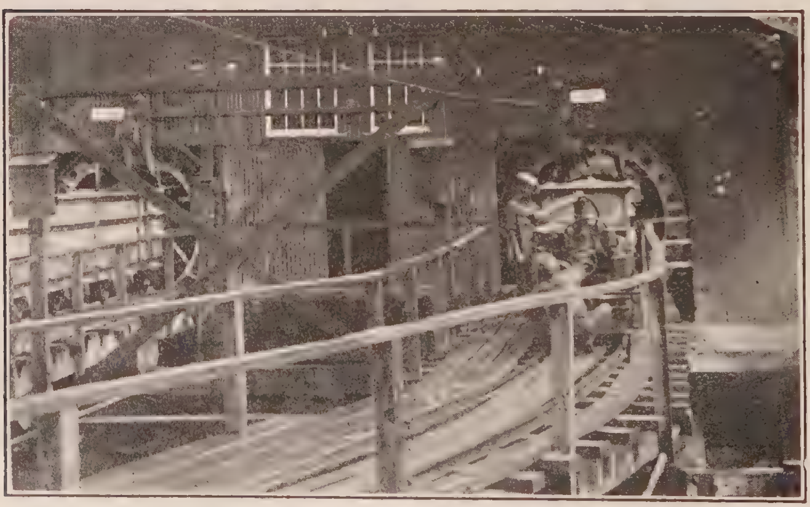

FIG. 17.—TIPPLE AT TIGER CAMP.

No timber is recovered from stoping operations, as the ground is heavy and the timber takes weight very rapidly; we are satisfied if the timber will hold until the filling is completed without requiring the additional expense of putting in doubling up sets and angle braces. Timber is loaded on special timber trucks 6½ by 3 ft. wide, 2-ft. gage and is hauled into the mine in special trains, by electric locomotives, to the inside shaft and winzes, where it is loaded into cages for transportation to the various levels. In the new circular shaft, these trucks will be run onto a large cage and hoisted to the various levels without any additional handling.

Underground Sampling

Sampling has been described under exploration work. As a check, crosscuts used for estimating the ore reserve are resampled by an independent sampler and have so far closely checked the original samples.

Every set of ground mined is located by the number of the stope and the number of sets north and south of the nearest 100-ft. crosscut and east or west of the drive and with respect to the floor above the level. For example, "4–14 S. stope 6th floor 3 W. & 4 S." locates a definite set of ground in the mine. This is interpreted as No. 4 level, 1400 S. coördinate, sixth floor, 3 west of drive, and 4 south of 1400 S. crosscut. There is great difficulty in sampling across the middle of the back of stope sets, as it is high up and, unless one is on the spot when the ground is taken out and before the lagging is placed, it is impossible to take the sample correctly. Often two or three sets fall out, leaving a dangerous hole, and the sampler thinks more of protecting his head than of getting a good sample. To avoid all trouble, danger, and careless work, samples are cut across the vein on the exposed side of the set about 4 ft. above the floor and after the timber is placed and lagged over; this arrangement gives the sampler two or three days time to cut his sample before the next set ahead is taken out.

Each level is divided into convenient sections and stope plans prepared of each floor in that section, showing all the ground taken out with its respective value. As the ore is thoroughly mixed in passing through continuous rock passes from the upper to the haulage level, ordinary hand sampling gives fairly close results. Each car is sampled as it leaves the portal of the mine. The ore then passes through the tipple, and by travelling belt, to the bins where it is discharged into 20-ton railway cars. Here it is again sampled by hand; and as it has been thoroughly mixed in going through the rock passes, cars, tipple plant, and bins, a fairly accurate result is obtained, which checks closely the sample taken at the concentrator.

Tramming and Haulage

On all levels, except Tiger tunnel or the haulage level, small ¾-ton cars of 20-in. gage are used to tram the ore from the stope chute to the nearest ore pass, which is seldom more than 200 ft. As the ore is very heavy it requires two men to a car, especially in tipping. Turntables were used at the intersection of the crosscuts and drives, as no extra ground had to be taken out for switches. As the tables used were unsatisfactory and a continual source of trouble, they were scrapped and switches put in. The mine is very heavy, and additional expense was incurred in continually replacing the long caps at the curve. However, a very satisfactory turntable was discovered at this smelter and installed in the mine, although it is more expensive than most others; it requires no repairs and is never out of order. For hand tramming, all right-angle turnings in heavy ground are provided with these turntables, which have simplified the timbering and cut down the repairs.

On the haulage level, 4-ton (56 cu. ft.) tipple cars are used. They are 6½ ft. long by 3 ft. wide by 2 ft. 11 in. deep and equipped with standard railway journals and brasses and the small size U. S. M. C. B. railway coupling. They are hauled, in trains of ten cars, by 4-ton Baldwin-Westinghouse electric locomotives of 2-ft. gage. Four of these locomotives handle the ore and supplies. Current is supplied by an overhead trolley line (one over each track), which is fed by two 42-kw. Westinghouse motor-generator sets. These sets take alternating current at 500 volts 63.3 amp., and deliver direct current at 250 volts 168 amp. The rails are 50 lb. and are bonded by General Electric twin-stud, flexible cable bonds, which have proved quite satisfactory both for safety from theft and for service. Gage of the track is 2 ft. and the grade is 0.7 per cent. in favor of the loaded train. On this grade no power is required to take out the loaded trains but considerable is necessary to take in the empties. A 4-ton locomotive will haul a train of ten empty (56 cu. ft.) cars up this grade without undue heating. A lighter grade would have been more efficient from the haulage standpoint but insufficient for drainage purposes.

Dumping

As the loaded train of ten cars (40 tons of ore) reaches the portal, it is pushed to the tipple. There are two revolving tipple drums 6 ft. 9 in. in diameter by 34 ft. long, each, geared to an Allis Chalmers, direct-current, 15-hp., 58-amp., 200-volt motor, which is fed from the trolley wire. Each drum is capable of receiving four 4-ton cars and discharging their contents in one revolution into the bin below. Four special revolving drums operated by 7.5-hp. motors are used for gates in delivering the ore from the bin to the incline belt conveyor. This belt, which conveys the ore to the storage bin, is 20 in. wide and 323 ft. long and runs on an angle of 16° 30′; it requires a 15-hp. motor and discharges on to a 20-in. distributor belt 406 ft. long equipped with an automatic tripper, which requires a 7.5-hp. motor. From the tripper, the ore is discharged into various storage bins, which supply the railway that hauls the ore to the mill at Namtu 7 miles away. As the ore passes over the end pulley of the elevator belt (just before it is discharged), it passes under a Cutler Hammer, type B, size 18, 2.75 amp. hot, 220-volt magnet, which takes out all iron.

Hoisting

As no ore is hoisted the hoisting plant, though small, is adequate and will suffice until the new 14-ft. circular shaft is sunk and equipped. Only timber supplies and coolies are hoisted at present. The hoisting is facilitated by the adits on Nos. 6, 2, 1, and 0 levels. The main inside shaft is equipped with a motor-driven double-drum electric hoist made by Allis Chalmers: one fixed and one loose drum; drum reel 4 ft. diameter by 2½ ft. inside flange, drum speed 34 r.p.m. or 408 ft. per min.; motor 50 cycles, three-phase; speed 725 r.p.m., 70 amp.; 500 volts, 60 horsepower.

The hoist is equipped with a solenoid brake on the motor and a Lilly brake (overwinding and speeding device) on the loose drum. The hoist motor is protected by an oil switch with an overload and no-volt release trip, which opens the circuit in case of a dangerous overload or stoppage of current and applies the solenoid brake on the motor. The Lilly brake controller is also connected to the no-volt release; and in case of overwinding or overspeeding, it not only applies the brakes on the hoist itself but also those on the motor. In addition, two limit switches located in the head frame open the circuit and apply all the brakes in case the Lilly brake controller does not function in an overwind. Close to the operator is a hand emergency switch for cutting off power and applying all brakes. The one fixed drum, instead of two loose ones, is a decided advantage when using coolie labor as there is just one-half the chance for an accident.

Double-cylinder 10 by 12-in. second-motion air hoists are used for operating winzes and 6 by 12-in. double-cylinder hoists for sinking. The large Holman stretcher-bar hoist is often used for the first 100 ft. in sinking. Small Holman 3 by 5-in. double-cylinder stretcher bar hoists and Little Tugger hoists are used for the stopes and rises.

Air Compressor

Compressed air is supplied by the following compressors:

1 Robey compressor, rope drive, 800 cu. ft. free air per min.; air cylinders 12 and 20½ by 30 in.

1 Ingersoll-Rand, imperial, type 10, 900 cu. ft. free air per min.; high-pressure cylinder 12 by 16 in., low-pressure 20 by 16 in.; belt driven.

1 Ingersoll-Rand, class P.R.E. 2, low-pressure cylinder 25 by 18 in., high-pressure 15¾ by 18 in., speed 214 r.p.m., motor 344 hp.; capacity of air in three stages: Full load 2370 cu. ft., 432 hp.; three-fourth load 1778 cu. ft., 321 hp.; one-half load 1185 cu. ft.; 255 hp.

The motors for the compressor plant are supplied from the low side of a substation receiving 33,000 volts and stepping it down to 550 volts; they are as follows:

INDUCTION MOTORS			
Robey compressor	200 hp.	226 amp.	500 volts
Imperial type	175 hp.	190 amp.	500 volts
SYNCHRONOUS MOTOR			
Ingersoll-Rand P.R.E. type	344 hp.	323 amp.	500 volts

On account of the large number of induction motors on the circuit of the mine, mill and smelter, this synchronous motor has been most beneficial and has brought up the power factor considerably. Another large synchronous motor has been ordered. One 350-kva. transformer at Tiger camp and two 350- and one 700-kva. at Bawdwin transform the necessary power for the mine.

Air is delivered into two air receivers (5 ft. 3 in. in diameter and 12 ft. long and 5 ft. in diameter and 13 ft. long) and then through a 10-in. steel pipe into the mine.

Ventilation

The ventilation of the mine is by natural draft assisted by mechanical means. Practically all air enters the mine through the Tiger tunnel and a shaft at the extreme north end of the Chinaman orebody, which is connected to each level. As the air enters the portal of the tunnel, it travels 7200 ft. before entering the actual mine workings. Here the current splits and passes up the various raises and winzes, spaced at least every 100 ft. along the vein, from level to level until it reaches No. 1 level, where it is accelerated by a 100-in. Sturtevant fan and discharged from the mine. This fan simply accelerates the natural draft and consequently increases the volume of air. It exhausts 70,000 cu. ft. per min. running at 579 rev. and driven by a 70-hp. motor. Although it is doing the required work, its speed is too great for efficiency and consequently consumes too much power.

As space is limited, the fan, which is a 100-in. double-inlet single-width, will be replaced by a 110-in. double-width occupying practically the same space and decreasing the power by about 50 per cent. for the same volume. At the same time it will be able to take care of a larger volume of air if required.

Lighting

Three General Electric lighting transformers, each 15 kva. 200 volts, located underground and at different parts of the plant supply the necessary electric lighting for bungalows, surface plant, main adits, haulage levels, shaft stations, and underground stores. The bulbs of all lamps are etched with the name of the corporation to prevent theft and sale in the local bazaars. Carbide lamps are used underground by the European shift bosses and candles by the coolies. These candles are of good quality and are made in the country by the Burmah Oil Co. They are colored green to prevent theft and sale but without avail. They cost Rs. 0.42 per pound.

The coolies were originally provided with three candles for an 8-hr. shift, but as they work in twos and fours it was observed that a group of four Chinamen working together would seldom have more than two candles burning at one time. The extra candles were taken home, collected, and finally sold in the bazaars of local towns. It is safe to say that the mine was furnishing light for the surrounding country. Candles have now been cut down to two for each coolie and still they have enough left to light their own homes and sell some to their neighbors. In a mine of cheap labor, the lighting account is large. In this particular mine it is the fourth largest separate account, being next to repairs and maintenance.

Telephones

A telephone system is installed at the supply stores on each level. A clerk is continually on duty and prevents meddling by the coolies. Originally, all the telephones were on the same circuit but this was found unsatisfactory for when one line became broken, which is often the case in a mine of heavy ground, all the telephones were out of commission; when defects or short circuits appeared they were more difficult to locate; the level clerks were continually using the phones for social conversation and it was difficult to get a message through. A small exchange was placed at No. 2 level store and separate lines run to each level; this system, although requiring a larger expenditure, is highly satisfactory.

Timekeeping and Store Checking

As many coolies have the same name and look alike to the new arrival (Europeans), it is necessary to give them a number; this is a round metal tag upon which the number is stamped. When he receives it, he is told in Chinese what number it bears and rarely does he forget his number or the appearance of the numerals on it. In a day or two, he can pick his number out of several disks and often will be able to call it out in English. Together with the brass disk, he is given a ticket bearing his number, rate of pay, and 15 spaces for the days of the half month. Each morning, before beginning his work, he appears at the time office and calls out his number; the disk is taken off the board and given to him. After the shift has passed through the time office, the number of tags remaining on the boards shows the timekeeper those who are absent and he can then make up his pay-roll sheet for subsequent checking. Checkers go underground and again check the coolies in their working places and make the required shift allocation. On coming off shift, the coolie is searched by the police (as he will steal anything no matter how small, as everything is of value to him if he gets it back to China) and presents himself again at the time office, where he hands in both his metal disk and his ticket. The former is put on the board and the latter (ticket) is punched for that day. In this way he can see at any time how many days he has worked and arguments at pay day are avoided. However, he is not paid on the ticket, that would be to his liking however, but from the actual pay roll which is kept up daily. The ticket is for his identification; if he loses it without reporting the fact, the company is not responsible if the pay envelope is delivered over to the presenter. New coolies use many ingenious schemes for punching their own tickets when absent, thinking they will be paid according to these, but when they find out it does not work and calls forth a penalty they do not do it the second time.

Tools of all kinds, dynamite, fuse, caps, etc. are of great value to the Chinese coolie when he goes home, consequently on issuing these articles certain precautions must be taken or they will not be returned. On every level there is a supply store where the coolie can get any tool or article he requires, but he must deliver to the issuing clerk his pay ticket as security until he returns the articles at the end of the shift. If the tools are not returned, his pay ticket is turned into the office with the list of tools and he is fined, or the matter is brought to the attention of the mine foreman.

The system of timekeeping and underground store checking has been gradually developed from our experiences, and is now the simplest and most satisfactory to all concerned. The Governments of India and Burma have stringent laws concerning the use of explosives and any quantity found in the possession of coolies is investigated; and the mine management is held responsible for any that gets out of its possession. Notwithstanding the punishment (which is both corporal and imprisonment) coolies are caught every week by the police guard, attempting to take explosives out of the mine.

WAGES	PER DAY
Carpenters	Rs. 3 to Rs. 4
Blacksmiths	Rs. 2 to Rs. 3
Machinists	Rs. 2 to Rs. 4
Fitters	Rs. 2 to Rs. 3
Pumpmen	Rs. 1/8
Electric fitters	Rs. 1 to Rs. 2
Motormen	Rs. 1 to Rs. 2
Miners and timbermen	Rs. 1/8 to Rs. 2/4
Muckers and truckers	Rs. 1/4 to Rs. 1/8
Subordinate clerks	Rs. 40 to Rs. 100 per month

RECORDS OF UNIT PRODUCTION

An average month is taken as the basis of the following calculations (14,643 long tons per month).

Stopemen are paid on a bonus system on the number of sets of ground taken out and timbered regardless of the tonnage contained therein as the ore may be low or high grade. Development is also paid on the bonus system, but on the footage advanced and timbered; other excavations, on the sedrum (100 cu. ft.); loading ore from bins, on tonnage.

It requires the following men (8-hr. shifts) to mine and timber one set of ground and muck the contents into the chute:

17.4 miners per set
6.6 muckers per set
24.0 men per set.

One set in ordinary ground removes 270 cu. ft. or 27 long tons of ore.

1.55 long tons per miner in stope for 8 hr.
4.10 long tons per shoveler in stope for 8 hr.
1.10 long tons per each man in stope for 8 hr.
11.00 long tons per each man on ore and rock in development for 8 hr.
0.54 long tons per each man underground for 8 hr.
1.16 long tons per each man engaged on surface including office.
0.37 long tons per each man of total organization.

CLASSIFICATION OF LABOR

As already mentioned, a large proportion of the labor is seasonal. The Chinese in large numbers come over at the beginning of the dry and go away at the beginning of the wet season. The turnover during the 6 months, November to April, is generally over 11 per cent.

	Europeans, Per Cent.	Asiatics	
		Clerks and Skilled Labor, Per Cent.	Unskilled Labor, Per Cent.
General office, hardware and P. & G. stores	0.25	0.92	0.25
Engineering, geology, assaying and sampling	0.25	0.12	0.44
Hospital and sanitary department		0.31	1.25
Mechanical department	0.06	5.00	5.00
Electrical department	0.06	0.75	0.91
General surface	0.06	1.22	15.25
Underground	1.00	1.00	65.90
Total	1.68	9.32	89.00

RECORDS OF UNITS OF SUPPLIES USED PER TON OF ORE PRODUCED

The explosives used per long ton of ore produced are 0.36 lb. 50 per cent. gelignite, 3 ft. Bickfords fuse, 0.55 No. 7 detonators.

The timber required per long ton of ore produced is 0.38 cu. ft. logs, 0.33 cu. ft. sawn timber, 0.30 cu. ft. 8 by 2-in. lagging, 0.27 cu. ft.

8 by 3-in. lining boards and cribbing or a total of 1.28 cu. ft. of timber.

The power required is as follows:

	Total Kw.-hr.	Kw.-hr. per Ton
Mining (compressed air)	178,120	12.16
Haulage and hoisting	38,000	2.60
Ventilation	25,000	1.71
Lighting, mine	4,000	
Surface	5,200	0.63
Workshop motor and laboratory	14,480	0.99
Framing plant	5,200	0.35
Total	270,000	18.44

Also, 10,000 cu. ft. free air compressed to 90 lb. were required per ton of ore

APPROXIMATE AVERAGE COST PER TON OF ORE

	Rupee
Ore breaking	6.542
Filling stopes	1.484
Tramming	2.516
Loading on railway cars	0.063
Mining development suspense	2.000
Tiger tunnel arching and maintenance suspense	0.125
Total	12.730

of which 8 per cent. is for explosives
17 per cent. is for timber
18 per cent. is for other supplies

Total......43 per cent. is for total supplies

DISTRIBUTION OF COST PER TON

	Ore Breaking, Rupee	Filling, Rupee	Tramming, Rupee
General expenses	0.349	0.057	0.083
Engineering	0.036		
Electric power			0.026
Repairs and maintenance	0.964	0.547	1.209
Lighting	0.396	0.120	0.089
Compressed air	0.068		
Hoisting	0.083	0.026	0.057
Mine ventilation	0.036		
Assaying and sampling	0.146		
Supervision	0.573	0.109	0.240
Miners and shovelers, etc	1.839	0.490	0.755
Explosives	0.786	0.021	
Timber	1.094		
Steel	0.016	0.010	0.005
Sundry supplies	0.156	0.104	0.052
Total	6.542	1.484	2.516

SAFETY AND WELFARE WORK

Notwithstanding so many coolies are employed who have never handled explosives or worked underground before, the fatal accidents are very few; there has been only one fatal accident due to explosives in five years.

No safety engineer is employed; but by making each European shift boss responsible for his level and by providing him with all necessary material and instructing him that the safety of the coolies comes first, the mine has been made comparatively safe. Extreme sanitary precautions are taken to avoid epidemics, which are so prevalent in the tropics—such as plague, cholera, relapsing fever, and other contagious diseases which wipe out villages in a few days. Fresh drinking water is piped to every level and many of the working places, and at suitable places on each level latrines are provided. At first, great difficulty was encountered in getting the different types of Indians and Chinamen to use the same latrines, but by most persistent work on the part of the European staff, with the aid of heavy fines and dismissals, the mine has been brought to a stage where for safety and sanitation it will rank with the best mines employing Europeans only.

Free medical attention is given to all natives and employees on the corporation property, and hospital accommodation for those that require it. The hospital and sanitary department is one of the largest and best in Burma.

For recreation, the corporation has provided clubs with tennis courts, both for Europeans and Asiatics, and is now trying to become a member of a cinematograph circuit. A first-class race-course has been built, to which everybody migrates on the only two holidays of the year—Christmas and Boxing Day. All the employees with their families are there: from the iron mines, a day's journey by train, from all parts of the forest, from the power plant (100 miles away), from Bawdwin (the lead mines), the mill, smelter and all the stations along the corporation's railway. They travel by train, ox-cart, pony, and afoot. Many of the Europeans and natives own their own racing ponies.

The Rove Tunnel*

By M. Mathieu,† Marseilles, France

(New York Meeting, February, 1923

The Rove tunnel is the means by which the canal from Marseilles to the Rhone River[1] penetrates the hills of Nerthe, lying between Marseilles harbor and Lake Berre, Fig. 1. The canal will communicate at sea level with both terminals. The original project planned a lock near the village of Marignane; but observations, pursued for more than three years, showed that the water surface gradient would average only 0.015

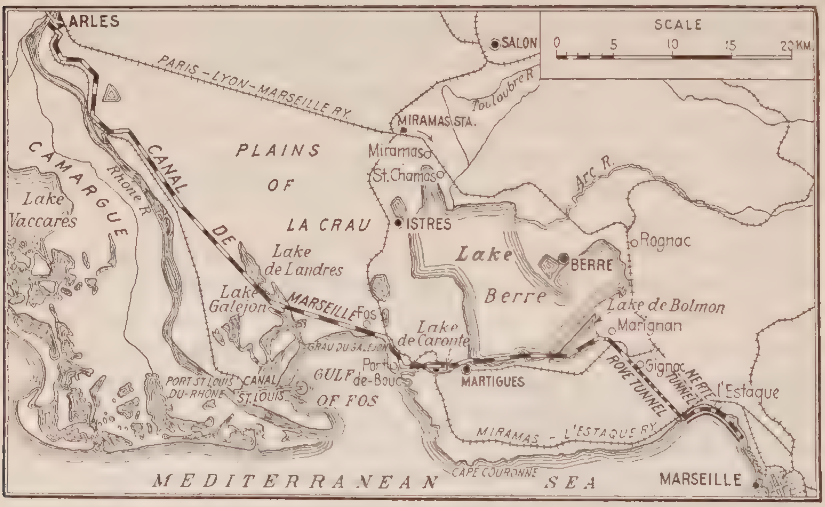

Fig. 1.—General map of the Rhone-Marseilles Canal.

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* Translated by R. R. Kirkpatrick.

† L'ingenieur ordinaire des Ponts et Chaussées.

[1] The canal from Marseilles to the Rhone River was the subject of a complete study, published by M. Bourgougon, chief engineer, in *Les Annales des Ponts et Chaussées* (November, 1914).

m. per kilometer during 321 days of the year, and would be greater than 0.03 m. during only two days of the year. The plans will be so arranged that the construction of a lock will be easy if its usefulness is recognized later.

General Description

The length of the tunnel is 7.12 km., and is on a straight line. Its cross-section is much larger than that of certain other tunnels, as shown by Fig. 2.

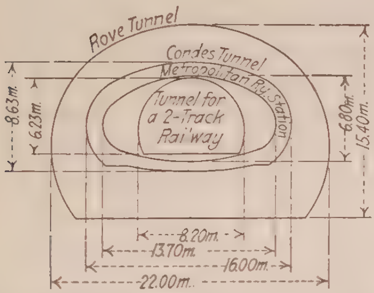

Fig. 2.—Comparison of several tunnel cross-sections.

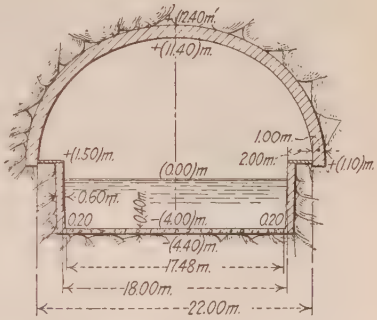

Fig. 3.—Tunnel cross-section, type 1 or 2.

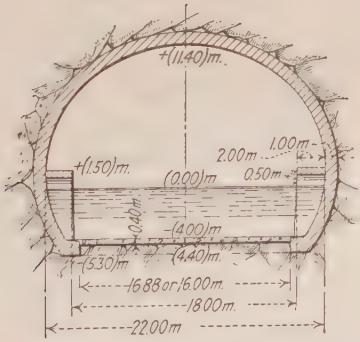

Fig. 4.—Tunnel cross-section, type 3 or 4.

The cross-section is of two types (Figs. 3 and 4), the choice depending on the nature of the ground; in certain parts, the bottom will be reinforced by a masonry floor down to the dotted line in Fig. 4. The average thickness of the arches was planned to vary from 0.70 to 1.25 m., but it has actually varied from 0.60 to 1.95 m.

The original plans called for a water-section 3 m. deep, but the final plans, adopted in 1919, increased this dimension to 4 m., since it would be impossible to enlarge the tunnel, after it had once gone into service, in order to accommodate the heavier traffic anticipated as a result of opening Lakes Caronte and Berre to ocean navigation. Furthermore, a boat traversing a canal sinks deeper than when it is moving at the same speed in the open sea; this fact was, observed by Inspector General Guerard on the Bouc canal at Martigues, on the Goulette canal at Tunis, and was verified in 1905 in the Suez canal. His experiments at Suez showed that the additional submergence varied, according to the shape of the boat, from 0.21 to 1.07 m., for boats 140 m. long, travelling at the rate of 14 km. per hour. Hence, the allowance of an additional meter in depth was thought to be no more than enough.

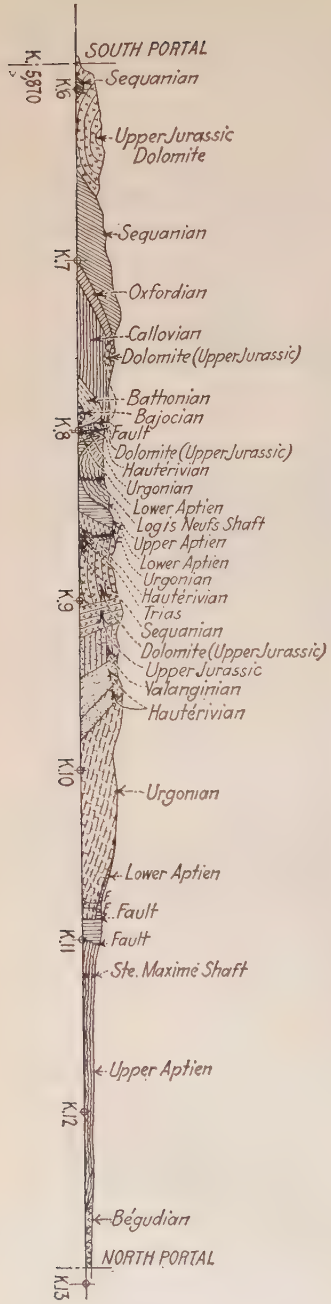

Fig. 5.—Geological section along the line of the tunnel (*Vasseur*).

Character of Rocks Penetrated

A careful study of the ground to be driven through was made in advance by several geologists. A cross-section, as now established, is shown in Fig. 5.

The Upper Jurassic dolomite consists of rocks of variable hardness in which are often found caverns with sandy pockets. The Sequanian is a hard lithographic limestone, in thin beds, with unimportant intercalations of chalky limestone. It is exploited at many quarries, in the region, and furnishes excellent building material. The Oxfordian and the Callovian are composed of compact limestones, which become marly at the base of the formation. The Bathonian and the Bajocian include some limestones and marls. The Hauterivian is formed, in its upper measures, of a yellowish limestone, in great beds of medium hardness, with alternations of marl; at the base it is more marly. The Urgonian is a massive white limestone of exceptional quality, much used in the region for cut stone.

The upper Aptien is a black marly limestone, marls predominating. Layers of hard gritty limestone are also found here. It is of mediocre appearance and breaks up easily in the air. In the region of Gignac the beds are folded, forming a syncline and anticline, which do not appear, at all, in the surface relief. The Lower Aptien is a hard limestone of medium quality. The Trias contains chambered dolomites and variegated marls; it is found in only small thickness. The Valanginian, in its upper part, includes a group of hard limestones in great regular benches; below are found, alternately, limestones and marls. The upper Jurassic is a crystalline limestone of excellent quality.

All of the strata encountered have been, in general, fairly strong; the only ground that has caused serious delays, which will be mentioned later, is the upper Aptien which is much folded and cut with numerous slip planes.

General Method for That Part of the Section above the Tow Paths

On the basis of competitive bidding, the contract was awarded to M. Chagnaud, who had already completed many remarkable projects, especially the first crossing of the Metropolitan under the Seine, the Lôtschberg Tunnel, and others.

The tunnelling operation comprised the following steps, as shown in Fig. 6: the successive driving of three advance headings (Nos. 1, 2, 3); enlargement and deepening of the top heading (No. 4); construction of the arch piers (No. 5); excavating the crown between the three headings (No. 6); construction of the arches (No. 7); removal of the center bench (No. 8).

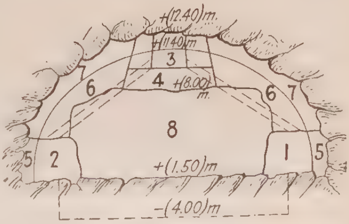

Fig. 6.—Sketch showing method of advancing tunnel.

The condition of the operations at each stage is shown in Fig. 7. The three headings communicate with one another: the bottom headings through cross-cuts, driven every 200 m.; the lower headings and the top heading by inclined raises, driven every 18 m. The material broken in the top heading slides through these raises into cars which travel in the bottom headings; thus the rock is entirely removed through these lower headings without having to displace the tracks.

A track is laid in the top heading after it has been enlarged (phase 4) and is joined to those of the lower headings by inclines on a 2.5-per cent. grade; this track is used mainly for bringing in material (timber and masonry). This provision of separate roads, the bottom for hauling away rock, the upper for bringing in material, maintains the operations in good order; furthermore, all materials are handled with the assistance of gravity.

The excavation of the crown (phase 6) has not been accomplished, generally, by continuous progress. In most cases, it would have been unwise to open such a large excavation, more than 22 m. wide, without limiting its length very closely. Hence the excavation is restricted to a total length of 6 m., and the masonry arch is erected as quickly as

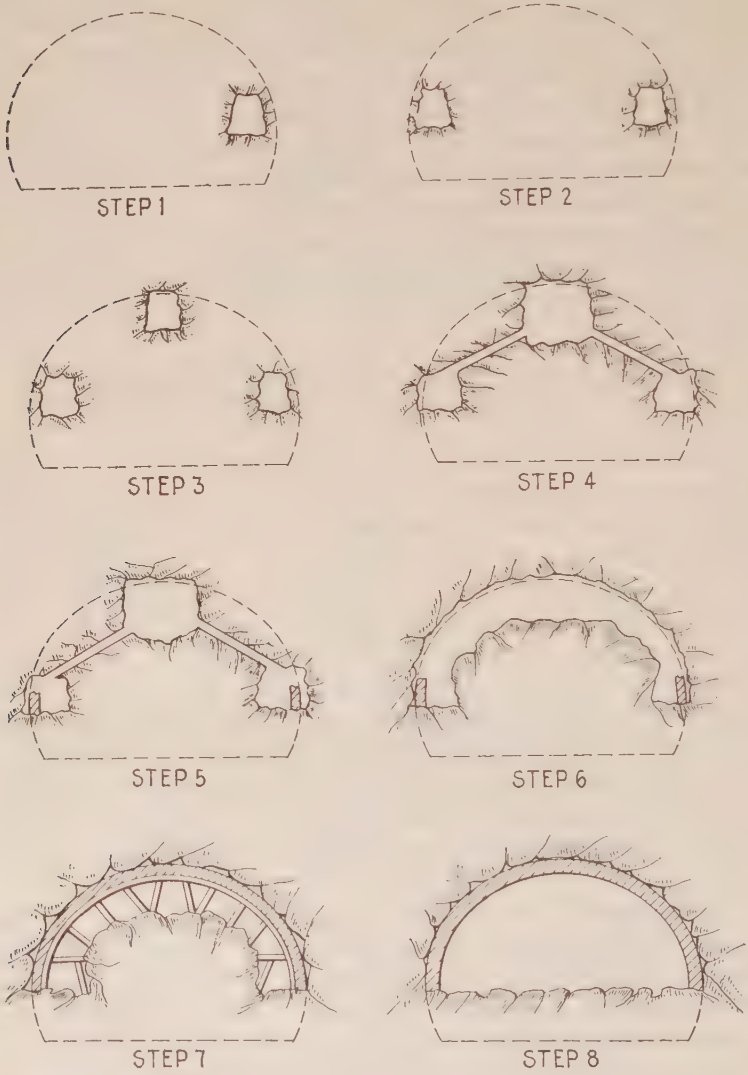

Fig. 7.—Sketch showing successive steps in advancing tunnel.

possible in this space. Inside of the masonry rings, so constructed, the natural rock remains in place, penetrated only by the three headings. This is then removed for a length of 6 m. so that the arch of the tunnel is formed of rings, 6 m. wide, which are not cemented together. This method has proved convenient and flexible, being easily adapted to the difficulties encountered; for example, the distance could be reduced to 3 m. when the ground was particularly bad.

The volume of excavation above the tow paths varies from 193 to 216 cu. m. per linear meter, depending on the thickness of the masonry, divided approximately as follows:

Lower headings (steps 1 and 2)	20 cu. m.
Upper heading (step 3)	5
Enlargement of upper heading (step 4)	20
Excavation of the crown (steps 5 and 6)	75 to 95
Removal of the center (step 8)	75
Total	195 to 215 cu. m.

The tunnel was started at the south end, on March 7, 1911. The headings have been driven from the portals without using the shaft, 142 m. deep, which had been sunk at a point 2300 m. from the portal.

The work at the north end was delayed by the fact that the tunnel there emerges in a cut 30 m. deep, which was started in August, 1912. In July, 1914, when war mobilization upset the work, this cut was not far enough advanced to allow advantageous working in the tunnel at the north portal. In the meantime, it was considered desirable to ascertain quickly the nature of the difficulties that would be encountered. A heading was therefore started in both directions from a shaft 69 m. deep, 3.50 m. in diameter, situated 1800 m. from the north portal. Connection between the two main headings occurred on Feb. 18, 1916, at a distance of 4708 m. from the south portal. In July, 1922, only a few meters remained to be arched, and the entire project is expected to be finished in 1925.

DRILLING AND BLASTING

The work is divided into three shifts, with a rest of 24 hours over Sunday. The drilling equipment consists entirely of compressed-air drills, working at an average pressure of 4 to 5 kg. per sq. cm. (57–71 lb. per sq. in.); they consume, including pipe-line losses, from 80 to 100 cu. m. (2825–3531 cu. ft.) of free air per drill, per hour. In this case, the use of individual drills was thought to have great advantage over any system of mounting.

The drill bits are of various forms, depending upon the hardness of the rock (augers, Z-shape, chisel-edged, etc.); their diameter varies from 25 to 32 mm. In general, the number of drills used was around 11 or 12 in each of the lower headings, 3 or 4 in the upper heading, and as many as 80 in excavating the crown. While taking out the center, the number varied widely, depending on the progress of the large heading; altogether, they have totalled 120 to 130 in the south headings. Each of them drilled 10 to 15 m. of hole per 24 hours, distributed in 10 or 12 holes. The depth of the holes averaged 1.20 m. in the top headings (section 5 sq. m.), 1.10 m. in the lower headings (10 sq. m.), and 0.90 m. in excavating the crown and taking up the center. The power consumed per cubic meter of excavation, the net cost of which is given at the end of this paper, was:

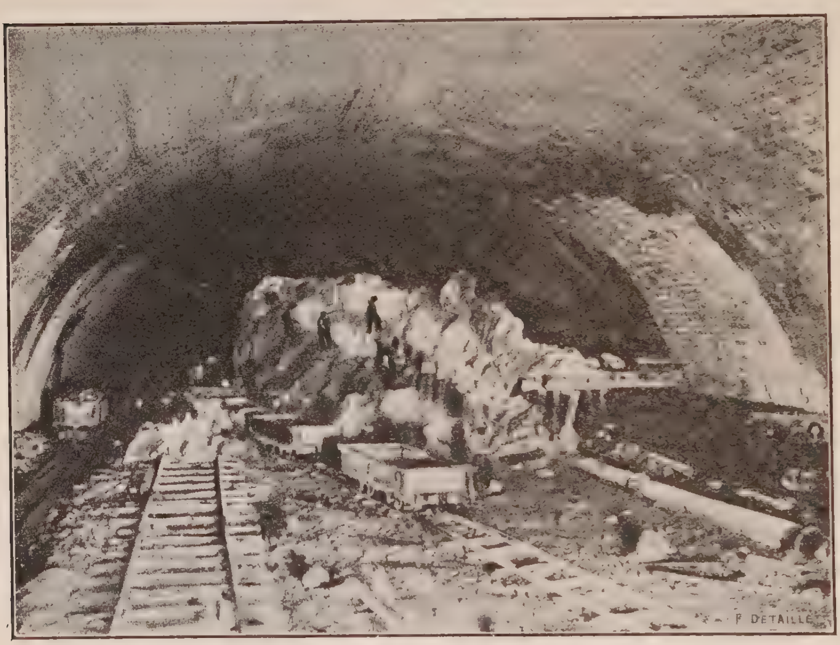

Fig. 8.—Excavation of the center bench.

Headings	58 kw.-hr.
Excavating the arch	18 kw.-hr.
Taking up the center	17 kw.-hr.

Different explosives have been used; those which have given the best results are 72 per cent. dynamite for the headings and the hard rock, and 57 per cent. for the softer rock. The consumption of explosives naturally has varied within wide limits; the following figures represent the average in compact limestone, per cubic meter of excavation:

Headings	1.6 kg.
Excavating the arch	0.5 kg.
Taking up the center	0.3 kg.

The load for each shot varied from 0.5 kg. in the headings, to 0.2 kg. for the arch and the center.

Advance in the headings often attained 5.50 m. per 24 hours; it was 1200 m. per year before the war. The maximum excavation in the arch was 470 cu. m. in 24 hours; it has been 144,000 cu. m. in a normal year. The maximum excavation in the center bench in 24 hours has reached 520 cu. m. The total excavation from the tunnel (north and south portals combined) is from 1200 to 1300 cu. m. per day.

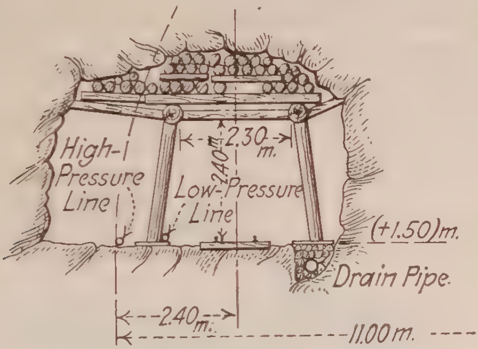

FIG. 9.—TIMBERING OF THE BOTTOM HEADINGS, AFTER WIDENING.

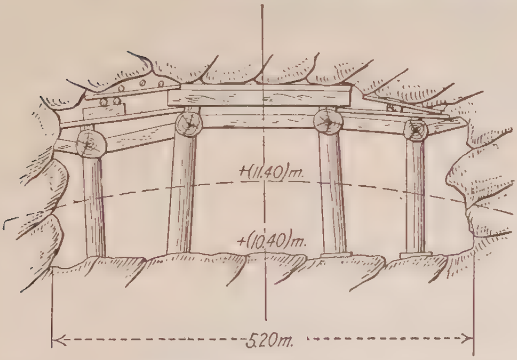

FIG. 10.—TIMBERING OF THE TOP HEADING, AFTER WIDENING.

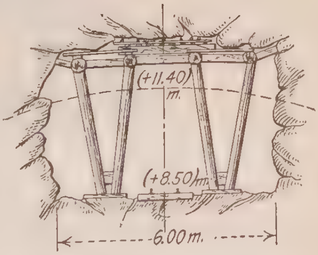

FIG. 11.—TIMBERING OF THE TOP HEADING, AFTER DEEPENING.

TIMBERING

Nearly everywhere, it has been possible to maintain the headings without timber; this has been used only during the enlargements (steps 5 and 6 of Fig. 7). Figs. 9, 10 and 11 represent the timbering for the enlarged top and bottom headings.

During the cutting of the arch, the arrangement shown in Fig. 11 is commonly used in the top heading. The timber is native pine posts, from 0.20 to 0.40 m. in diameter. They were spaced according to the nature of the rocks; in some places, for considerable distances and even during the cutting of the arch, no timbering was necessary. In other places it was necessary to sustain the entire area; Fig. 12 shows the most extensive form of timbering required.

HAULAGE

The excavation from the south portal (more than two-thirds of the total) is lightered to the site of some work under construction for the Port of Marseilles, a distance of 4 to 5 km. from the mouth of the tunnel. The excavation from the north portal is hauled along the south bank of Lake Bolmon, where ground is being made, an average distance of 4 km. from the end of the tunnel. Haulage inside the tunnel has been carried on with compressed air and ordinary steam locomotives, the choice depending on ventilation conditions provided by the shafts. At the beginning of 1922, the steam locomotives at the south end went as far as 5 km. from the portal.

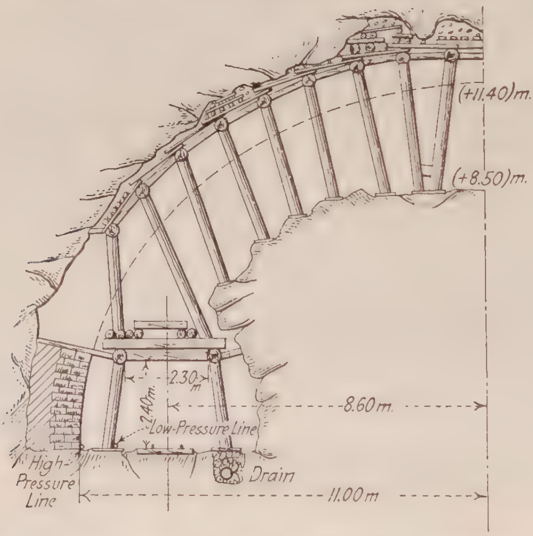

FIG. 12.—TIMBERING AFTER EXCAVATION OF THE CROWN.

The compressed-air locomotives (7 in number) use air at a pressure of 100 kg. per sq. cm. (1422 lb. per sq. in.); at present, the air conduits are overworked and the pressure is not over 70 to 80 kg. These locomotives weigh from 12 to 24 tons, and haul trains with an average of 25 cars. The corresponding power consumption is about 50 kw.-hr. per cubic meter-kilometer, including all losses.

The steam locomotives (5 in number) weigh 20 to 25 tons, and haul trains of 50 to 70 cars, each containing 2.5 cu. m. of rock. Nearly 500 cars are required for a daily extraction of 1200 cu. m.

The steel rails weigh 20 to 25 kg. per meter, and are laid on wooden ties with a gage of 0.75 m. The total length of track, not including that at the face of the headings, has reached 30 kilometers.

SURFACE PLANTS

Each of the approaches requires an independent plant, each of which includes an air compressor, a repair shop, a rock crusher, a store house for lime and cement, and an office building. The air compressors deliver at a pressure of 10 kg. per sq. cm. for the drills and at 100 kg. for the locomotives.

The power, installed, is 2000 kw. at the south portal and 1000 kw. at the north portal; the largest unit at the south approach is of 600 kw. At the south approach, current is received in three-phase, 50-cycle, 190 and 5000 volts, the latter being used for the air-compressor motors. At the north end, 25-cycle, 440-volt current is used.

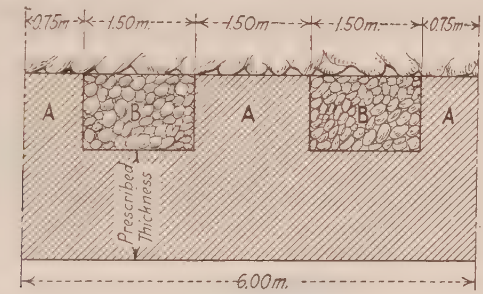

FIG. 13.—PLAN OF MASONRY CONSTRUCTION, WHERE DISTANCE BETWEEN INTRADOS AND ROCK WALL WAS GREAT.

MASONRY

The arch of the tunnel is walled for its entire length. In the original plans the average thickness of the walls was defined but the form of the extrados was to be determined during the course of the work. In practice, the masonry has been filled out to the solid rock, being therefore irregular in thickness, although the excavation has been maintained at as uniform a width as possible. When, for any reason, the excavation would require too great a thickness, the masonry is extended to the solid rock only in small pillars, *A* in Fig. 13, the empty spaces *B* being filled with dry stone.

Arches having a thickness of less than 1 m. consist of one course of rough stone, dressed on their bed and joint faces, and with a thickness of alternately 0.30 or 0.45 m. Arches with greater thickness are built of two courses of dressed stone. The rest of the masonry is constructed, in the usual way, of successive covings whose faces are perpendicular to the intrados.

The dressed stone is of hard limestone, obtained from quarries outside the tunnel and delivered at the heading ready cut. Part of the rough stone comes from the same source, but some is obtained from suitable strata met in the tunnel.

The sand is prepared by crushing limestone, and has the following screen analysis:

Apertures per Sq. Cm.	Per Cent.
On 1	5.24
1 to 4	28.78
4 to 25	31.64
25 to 400	22.22
400 to 900	4.50
Below 900	7.62

The lime is a hydraulic lime, from the "Société des Chaux et Ciments de Lafarge et du Teil." Six laboratory tests on a mixture of 1 part of lime to 3.5 parts of sand crushed from rock taken from the headings gave the following results, in kilograms per square centimeter:

	Tensile Strength after		Compression Strength after	
	7 Days	28 Days	7 Days	28 Days
Minimum	6.5	19.0	99.0	151.7
Maximum	8.0	24.0	124.6	174.1
Average	7.6	21.8	108.3	165.3

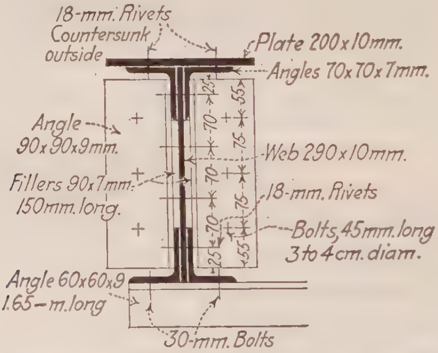

Fig. 14.—Cross-section of metal arch supports and forms.

Other tests on a 1 to 2.5 mixture with Lencate sand gave: in tension, after 7 days, 8 kg.; after 28 days, 16 kg.; in compression, after 7 days, 60 kg.; after 28 days, 105 kg. per sq. centimeter.

After the enlargement of the lower heading, the arch foundations are built to a height of 2.75 m., as high as they can be made without forms (step 5 of Fig. 7). When the cutting of the arch space is complete, the arch centers are installed. These forms are made of steel I-beams, as shown in Fig. 14. Each truss is composed of nine pieces, each weighing 300 kg.; there are five trusses for each 6 m. length of arch. The arches are supported, every 2 m. by posts set on the center bench, and resting in sand boxes.

The masonry arch rests on the truss for about 12 days ordinarily, but with thin arches in strong ground, the forms have been taken down after 7 days.

The average time required for the construction of one ring of the arch, 6 m. long, is from 7 to 8 weeks, for example:

Excavation of the arch space	2 to 3 weeks
Building and setting the form	1 week
Masonry	3 to 4 weeks

General Method for That Part of the Section below the Tow Paths

This part of the work was begun in earnest in 1920 at the south portal, and in 1922 at the north portal. By reason of the different nature of the formations, the method varies from place to place; the system employed at the south end is represented in Fig. 3. A cuvette or trench, 20 sq. m. in cross-section, was begun the excavation being loaded by hand into boxes; these were hoisted by a crane standing on a platform, and dumped into cars running along the same side. The cranes were operated by compressed air from the same air line as the rock drills, and consumed about 200 cu. m. of free air per hour, each crane handling approximately 6 cu. m. of rock per hour. This cuvette was then enlarged to make space for the lining walls. Then the excavated material was loaded onto cars running in the bottom of the cuvette, joined to the upper tracks by inclines. Several such workings were begun at intervals in the five kilometer length.

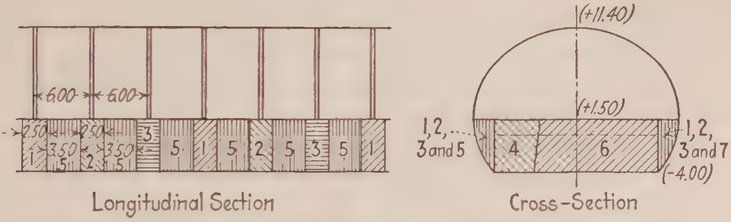

Fig. 15.—Method of attacking the water section, north end.

The cross-section adopted in the north end is shown in Fig. 4. The foundation wall was made in small sections, 2.50 m. long (No. 1 of Fig. 15) in a pit under a joint of the masonry ring; successive pits were spaced every 18 m. The wall was then built under all the joints (Nos. 2 and 3 of Fig. 15). Afterwards a cuvette (No. 4) was dug as for the south end of the tunnel, and that part of the walls (No. 5) left between the pits at the side of the cuvette was constructed. In difficult parts of the work, inverted arches were set between the pits, opposite each other (Fig. 4). Then the center bench was taken up (No. 6) and the wall was finished on the side opposite the cuvette (No. 7). The numbers in Fig. 15 show the progress of the work.

The original plans allowed for no masonry on the floor, except at points where some heaving may be expected; there inverted arches 1 to 1.2 m. thick must be built. Undoubtedly, a large portion of the canal bottom must be walled where it penetrates the Aptien, because this rock is rapidly decomposed by the action of water and air. The lining will probably be of concrete with a thickness of 0.30 to 0.40 m. This question is yet to be settled.

The Problem of Drainage

From the geological studies, very little water had been expected. The contractor had estimated the amount of power required for drainage at only 50 h.p., any additional expense for pumping power to be shared equally between the contractor and the Administration.

These very optimistic forecasts have not been realized. At 130 m. from the south portal, a spring running 60 l. per sec. was encountered. At 3008 m. from the south portal, the east heading met a spring with a continuous flow of 60 l. per sec. with a pressure of 3 kg. per sq. cm. The number of pumps in service soon went from three to eight and then to 12, in spite of which the advance headings were drowned out for many days. The flow continues to this day. At first they tried to force a passage by driving ahead either the west or the top heading, which up to that time had been a little behind the east heading. This work was pushed with numerous difficulties for two months, until the west heading had advanced as far as the east one, and was inundated in turn. The flow of water reached 100 l. per second.

They then decided to proceed by injection of cement. The two advance headings were walled up; in the walls were placed pipes with bolted covers to which valves were fastened. The injections were made first with cement, then with the same hydraulic lime that was used in the masonry. The hydraulic lime gave the best results because it sets more slowly and fills the fissures far enough from the point of application to form a barrier on the perimeter of the tunnel. Injections began with a pressure of 3 kg. and finished with 5 kg. They were continued for three months and required 400 tons of cement and lime. As one set of fissures were closed, the water found its way through other passages and it was necessary to come back to stop them. After 5 months the flow of water fell to 35 l. per sec., when it became possible to begin driving again, with occasional repeated injections. The water-bearing zone measured 100 m. along the tunnel, and was situated in the dolomite (Fig. 5).

The principal drainage trouble consisted in the disposal of water through a level ditch. It was difficult to place pipes in the advance headings, since they would be damaged when the enlargements were made. On the other hand, space was lacking for sufficiently large ditches, and these often filled up with debris. The contractor, however, at first adopted this latter solution. The ditch was stopped at intervals (400 to 500 m.) by a dam; a pump then lifted the water over the dam, and the difference in level, so created, assured the flowing of the water. However, when the flow of water into a pond became greater than the capacity of the discharging pump, the water gradient diminished, the current decreased, and flooding resulted.

In spite of these inconveniences, this system was used for a long time on account of the difficulty during the war of obtaining the necessary materials.

In 1916, a large sheet-iron ventilating pipe became unengaged and was installed as a water line in that part of the tunnel where the excavation was finished, a distance of 3100 m. from the south portal. But this was laid without grade, and the reservoirs into which the water was pumped were so low so that the pressure was very weak. In spite of its large diameter (0.60 m.) this system took away only 100 l. per sec., an insufficient amount in flood times. A little later, another pipe with diameter of 0.40 m. was laid between the points 3100 and 4200 m. from the south portal; but as this pipe did not discharge directly into the first, it was necessary to handle the water again with another pump.

This layout was greatly improved in 1920 and 1921. The advance of the excavation below the level of the tow path (+1.50 m.) forced the contractor to protect himself against water with much more adequate methods. The following scheme has been used: water encountered at a distance greater than 5000 m. from the south portal is sent out by the north opening; it is not abundant and a 15-cm. pipe is sufficient. Water from the advance headings is impounded and pumped into the 0.40- and 0.60-m. pipes mentioned above. These pipes have been repaired and connected in such a way that all of the water, in normal times, can be discharged without pumping.

The exceptional rains of October, 1920, and April, 1921, greatly increased the flow of water from the springs. The advance headings were inundated many times, and for 15 days in October all work in the tunnel was stopped, the flow at the outlet attaining 580 l. per sec. The drainage system, as finally installed, has never been overloaded on account of the light rainfall since the date of its completion. The total discharge is now only 160 l. per sec., of which 60 l., coming from the first spring, flows naturally to the sea.

The amount of pumping power, paid for by the Administration, from the beginning of work to the end of 1921, has reached 10,500,000 hp.-hr., representing a total expenditure of 1,700,000 francs. During the dry season, the total power used for pumping is about 125 horsepower.

CAVING AT RING No. 103

In November, 1919, the headings started from the north portal had just passed ring No. 103. (That part of the arch which is constructed in a single piece is called a ring. These parts are 6 m. long; ring No. 103 is therefore at 6 × 103 = 618 m. from the north portal.) At this point, the rock is the Upper Aptien. There were no signs of water at ring 103; on the contrary, the rock was very much crumpled by faults in various directions, the planes of which were smooth and slippery. The cutting of the crown had been started and five headers had been placed on each side of the axis of the tunnel; the timbering was thus much less extensive than that shown in Fig. 12, which has nine headers.

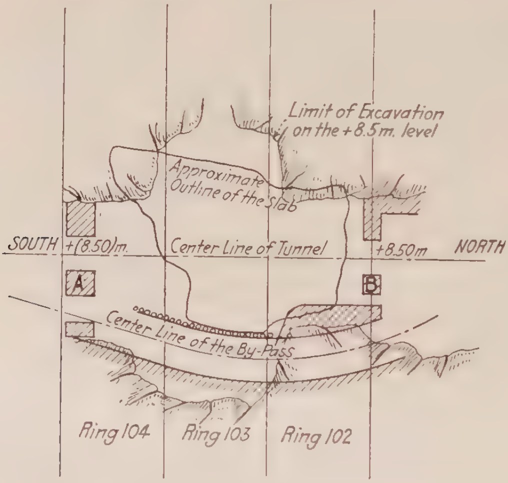
Limit of Excavation on the +8.5m. level
Approximate Outline of the Slab
SOUTH
+(8.50)m.
Center Line of Tunnel
+8.50m
NORTH
A
B
Center Line of the By-Pass
Ring 104
Ring 103
Ring 102

FIG. 16.—GENERAL PLAN AT RING 103, DURING CONSTRUCTION OF THE REINFORCED CONCRETE SLAB.

During the night of Nov. 21, a cave of 150 cu. m. occurred; the timbers were crushed and the top heading was filled with muck. The cavity produced was in the form of a three sided pyramid; the north face was smooth, but the south face (the advancing side) showed irregular breakage. Some irregular fissures soon appeared above ring 102 (to the north of 103), showing clearly that haste was necessary in consolidating the whole. The contractor began to place headers above the cave, then to lay lagging, upon these, supporting the edges of the caved hole. This work was almost finished when on Dec. 18, a second caving occurred, larger than the first (250 cu. m.). All the work was lost; the timbering under erection was crushed as well as that in ring 102, which had stood up the first time. The debris completely filled the top heading and the raises between this and the lower headings.

It was impossible to continue the application of ordinary methods.

In the bell-shaped hole, more than 400 cu. m. in volume above the crown of the tunnel, rock fell continuously, rendering the work very dangerous. It would have been necessary to use timber of such great length that it would have had very little strength unless an elaborate structure were erected, which would have taken too much time. In fact, further movement was feared; in rings 106, 109, and 112, fissures appeared and soon became larger.

Unquestionably, the first thing was to prevent extension of the caving. For this purpose, cross walls were built in rings 101, 104, and 109. These walls, about 1 m. thick, rested on the center bench and supported the roof of the top heading. Holes large enough to permit the passage of cars were left in these walls, two of which are shown at *A* and *B*, Fig. 16. Between the walls in rings 101 and 104, the space was filled as high as possible with material from the cave or with posts piled above the by-pass heading (Fig. 17). The middle bench was sustained by temporary walls, *C* and *D* of Fig. 17. Tunneling was then resumed on the east side which seemed more solid. At the level of the top heading a by-pass was driven, permitting the resumption of car movement and the supplying of the main headings with material.

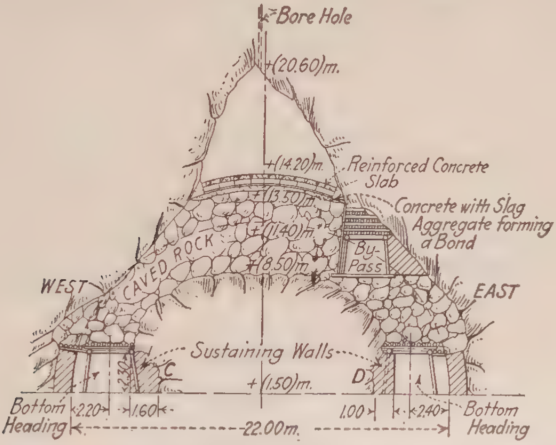

FIG. 17.—CROSS-SECTION OF RING 103, DURING CONSTRUCTION OF THE REINFORCED CONCRETE SLAB.

At the same time, profiting from the fact that the thickness of cover over the cave was only 22 m. a bore hole with a diameter of 0.20 m. was put down from the surface to the opening above the cave. The surface of the fallen rock was then quickly smoothed into a dome shape and a slab of concrete, 0.70 m. thick, was poured. The concrete, composed of 500 kg. of cement to 800 l. of crushed rock and 400 l. of sand, was mixed at the surface and dropped through the bore hole. Workmen, entering the cavern through a narrow raise driven through the caved ground, leveled the concrete.

On top of this slab of concrete, a longitudinal heading was kept open (Fig. 19), but the rest of the space was filled with a concrete containing 200 kg. of hydraulic lime per cubic meter of slag; slag was chosen because of its lighter weight. Part of the pressure was carried over to the solid rock by two abutments (*E* and *F*, Fig. 19) of concrete.

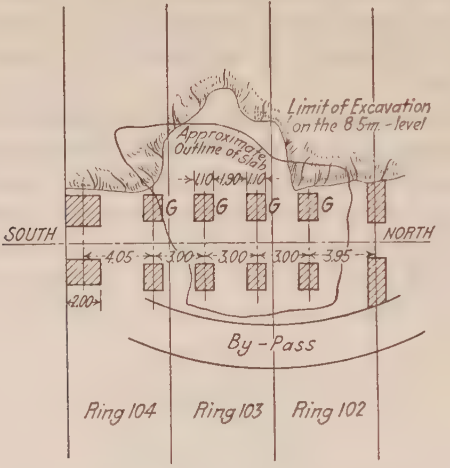

FIG. 18.—PLAN OF RINGS 102, 103 AND 104, DURING CONSTRUCTION OF THE PERMANENT ARCH.

After the completion of this protective work, which lasted until the end of August, 1920, excavation of the central heading under the concrete slab was resumed. Masonry pillars (*G* of Fig. 18) were built to sustain the slab *E*, and then the arch was constructed; on the intrados, the ordinary arch form was used, but on the extrados a half circle was made.

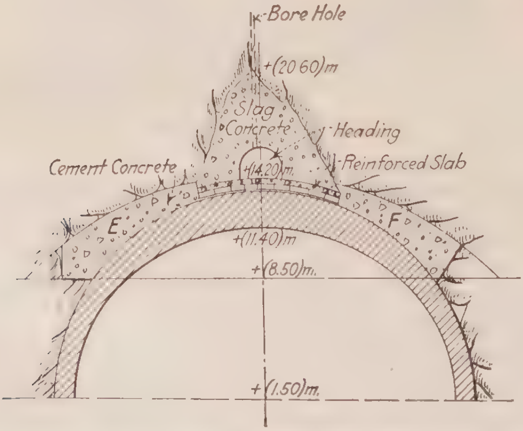

FIG. 19.—CROSS-SECTION OF RING 103, AFTER THE PERMANENT ARCH WAS BUILT.

It was erected in sections, 3 m. long. All was finished in June, 1921, without further incident, since when no movement has been noticed.

Minor Cavings

The caving in the neighborhood of ring 103 caused stresses on the neighboring rings. On April 9, 1920, while the forms were being taken from under ring 105, movements injured ring 106 (which had its supports removed Feb. 11) and ring 107 (which was being excavated); In both rings, the masonry seemed to have been badly damaged. Rings 106 and 107 were now solidly propped and the supporting forms of ring 105 were again wedged up. When the bell-shaped cavity of ring 103 had been filled, the damaged parts of the masonry were taken out. In the solid rock could be noticed a nearly vertical fault plane along which the movement had taken place. The masonry was rebuilt and for two years, nothing abnormal has been observed.

In April, 1920, a ring fell in, three days after its arch support had been taken away. A rock slab, nearly 2.50 m. thick, fell with it. During the construction of the arch, nothing unusual had been noticed; the ground did not appear bad, so that the thickness of the masonry had been reduced to 1.07 m. The failure seemed to be due to some big load concentrated around the key of the arch, that had not been suspected during construction. It is possible that the firing of shots in the neighboring excavation had some part in the failure. The reconstruction of the arch took place without incident.

In two other cases, arches collapsed while their supports were being taken down, but it is known that these failures were due to faulty construction. Their reconstruction involved no difficulty.

Since the spring of 1922, important movements have been noticed in the zone between 1700 and 1800 m. from the north portal, occupied by the upper Aptien rocks in which the trouble with ring 103 occurred. These rocks are much folded and contain numerous fissures. Certain schist strata, also, dipping from south to north and composed of small blocks without cohesion, cut the mass. These layers of pulverulent matter, no doubt, transmit in all directions the pressure of the overlying strata.

The first movements were noticed at the time when the north headings were connecting with that part of the tunnel in which the arch had already been constructed in the south headings. In the advance rings, the masonry was built without trouble, for a distance of 18 m. at a time. When excavation of the neighboring rings was started, it was noticed that fissures had occurred in the earlier rings, from some of which the forms had been removed for several weeks. At the same time, the foundations of the rings in course of excavation were cracked and showed a tendency to be overthrown, as if they were being submitted to a horizontal thrust.

The masonry of the advance rings was supported by putting back the forms and by building some piers on the middle bench. Also the construction of the neighboring rings has been hurried (today the arch keys have been set) and the damaged piers have been rebuilt. This work has followed without special difficulty, other than the necessity of reducing the length of the advance from 6 to 3 m. The same precautions will be taken in the arches yet to be built, and the cracked arches will be repaired. This work is necessarily quite slow but it presents no particular difficulty.

Cost Analysis

For an undertaking so complex as a tunnel, where the work extends for more than ten years, it is impossible to give data with any degree of precision, unless some one particular situation is taken as an example. These considerations are much more important in this case, because during and since the war, wages and the cost of material have been subjected to rapid and enormous variations (from 1 to more than 10 for coal).

At the outbreak of the war, operations had reached the following stage:

Average length of headings from portal	2100 m.
Average length of arch excavation	1600 m.
Average length of masonry	1550 m.
Average distance from portal to the middle bench	650 m.

The rock then being excavated was limestone of good quality, requiring very little timbering.

Table 1 gives the prices of the various items used in the calculation of costs. Table 2 gives the amount of labor and material required per cubic meter or excavation. From this, the cost of excavation and masonry is computed, as shown in Tables 3 and 4.

This cost does not include the great overhead expense of the undertaking, comprising amortization of the equipment (stationary, rolling stock, machines, buildings, etc.); operation and repair of this equipment; management; taxes, interest on money advanced, etc. The exact total of these latter charges is difficult to obtain, precisely; as an approximation, they can be estimated as one-fifth to one-fourth of the operating expenses.

The cost of operation has been established under other conditions; they will not be given here, to avoid complicating the problem by examining the differences in the methods of driving the headings, or in the scales of wages and prices.

For excavation of the water section, no cost was established before the war, as this work was not conducted to any great extent until 1921. Comparison with the preceding work, therefore, being difficult, only the following general statement can be made.

In the south approach, removal of the water section, the excavation being hoisted by crane, is 20 to 40 per cent. less expensive than the cutting of the arch. The enlargement of the water section is comparable to the taking up of the middle bench. In the north approach, the excavation of the pits under the arch joints is comparable to the cutting of the arch.

At the beginning of the work, the cost of the tunnel was estimated as follows:

	Francs
Work by the contractor	40,000,000
Furnishing of lime by the Administration	1,500,000
Inspection, contingencies, etc	4,000,000
	45,500,000

This did not include the expense of landing the excavated rock on the site of the harbor work in the port of Marseilles (1,000,000 fr.), nor that for the approach channel at the south end (104 m. long—900,000 fr.), nor that for the cut at the north end (1900 m.—6,500,000 fr.). The construction of the tunnel alone cost a little less than 6400 fr. per linear meter, on the average. Table 5 gives the principal prices paid the contractor; these costs relate to the south half; the cost of the north half was 10 to 12 per cent. greater. The conditions under which the work was conducted until the middle of 1914 showed that the actual cost would have been only a very little more than the estimate; notable economies, to the extent of 1,000,000 fr. had been realized by the general use of thinner arches. But the war upset all operating conditions.

It has been necessary to reimburse the contractor with additions to the contract price, due to the difficulties of all sorts that have been encountered, in recruiting labor; delays in supplying materials (especially explosives, coal, and wood); increases in prices; irregular working force due to successive mobilizations—the French and then the Italian mobilizations twice have disorganized the work. To rectify this injustice, the contractor was first paid an indemnity for all past work and then the remuneration per linear meter of completed arch was determined at the end of each year. These last payments varied from 1000 fr. in 1917 to 4600 fr. in 1919 for one linear meter of arch, built of one course of dressed stone; and from 1250 to 6000 fr. for the arches with two courses of rubble stone. The amount so paid was about 4,500,000 francs.

By 1920, the disparity between the original price and the cost at that date was such that it became necessary to make a general re-adjustment. The price was increased from 280 to 300 per cent., resulting in an increase of expenditures of 64,000,000 fr. in round numbers. This includes the 7,000,000 fr. entailed by deepening the water section from 3 to 4 m. The actual cost will thus be raised to the following total:

	Francs
Initial expenditures	45,500,000
Various indemnities paid during the war	2,000,000
Increase in contract price after the war	57,000,000
Deepening of the water section	7,000,000
Total	111,500,000

It seems probable that this estimate will not be greatly exceeded, in spite of recent difficulties encountered, and the additional expenditures which have resulted. In fact, the price paid to the contractor will not be more than the fixed price. By reason of rapid variations in wages and cost of material, these costs have varied, since 1920, in a very complicated manner. The price now being paid is 10 per cent. less than in 1920.

Table 1.—*Factors Involved in the Operating Cost*

A—Labor

Kind of Labor	Wages per Day, Fr.
Chief mason	11.65
Chief timbermen	8.50
Chief miners	8.50
Masons	7.45
Muckers	5.10
Common labor	5.10
Timbermen	7.10
Timbermen helpers	5.10
Miners	6.20
Engineers	5.20
Firemen	5.20
Brakemen	5.00
Boys	3.50

B—Supplies

Kind of Material	Unit	Prices per Unit Fr.
Sand	Cubic meter	2.20
Ordinary rough stone	Cubic meter	4.30
Dressed stone	Cubic meter	31.00
Timber (see note)	Cubic meter	35.00
Coal	Ton	37.20
Lime (furnished by Administration)	Ton	31.00
Power	Kilowatt-hour	0.04
Explosives—Dynamite	Kilogram	4.30
Cheddite	Kilogram	2.80
Caps	Each	0.025
Fuse	Linear meter	0.025

Note.—The timbers are mostly 5- to 6-dm. round posts; they are generally used twice. The price, 35 fr. per cu. m., takes account of the double use.

TABLE 2.—*Consumption of Principal Materials, per Cubic Meter of Excavation*

Item		Headings	Cutting Crown	Taking up Center
Labor (in hours)	Extraction and loading	10.0	8.0	4.0
	Timbering		1.0	
Power (in kw.-hours).	Drilling	58.0	18.0	17.0
	Drainage	1.5	1.5	1.5
	Ventilation	0.5	0.5	0.5
	Haulage	1.0	1.0	0.5
Material	Timber (in cu. m.)		0.035	
	Explosives (in kg.)	1.6	0.5	0.3

TABLE 3.—*Cost of Excavation, Francs per Cubic Meter*

(General expense not included)

Item	Headings	Cutting Arch	Taking up Center
Labor	7.50	6.00	2.30
Power	2.35	0.70	0.70
Timbering	0.30	0.90	
Explosives	7.20	2.25	1.00
Drainage	0.15	0.15	
Ventilation	0.05	0.05	0.05
Haulage	0.95	0.80	0.50
Total	18.25	10.85	4.55

NOTE.—Haulage, outside the tunnel, is not included.

TABLE 4.—*Cost of Masonry, Francs per Cubic Meter*

(General expense not included)

Item		Cut Stone	Ordinary Stone
Labor	Dressing of stone	5.15	6.70
	Setting forms	2.15	2.15
	Handling material	0.45	0.45
Material	Sand	0.65	0.90
	Dressed stone	31.00	
	Ordinary stone		4.75
	Lime	4.35	4.35
Supplies	Timber	1.05	1.05
	Coal and fuel	0.30	0.30
	Power	0.45	0.45
Total		45.55	21.10

NOTE.—Haulage, outside the tunnel, is not included.

TABLE 5.—*Prices Paid to the Contractor*

(Before the war; south half of tunnel)

	AVERAGE THICKNESS OF WALLS, M.	PRICE PER LINEAR METER, FR.
Completed tunnel, profile No. 1	0.70	4227.24
Completed tunnel, profile No. 2	1.00	4748.18
Completed tunnel, profile No. 3	1.00	5530.90
Completed tunnel, profile No. 4	1.25	6447.51
Floor masonry, profile No. 1	1.00	573.22
Floor masonry, profile No. 2	1.20	739.84

Liquid-oxygen Explosives at Pachuca

By Michael H. Kuryla and Galen H. Clevenger.

(New York Meeting, February, 1923)

Some years after Nobel made his epoch-making contribution to the knowledge of high explosives, Sprengel[1] described a new class of detonating explosives consisting of mixtures, made immediately before use, of an oxidizing agent and a combustible substance.[2] The novel feature of the Sprengel explosives is that the constituents alone are non-explosive during manufacture, storage and transport. Despite this advantage, and although some of these have been used in important work, they are now little used, possibly on account of the greater inconvenience to the user, compared with explosives supplied by the manufacturer in final form.

Liquid-oxygen explosives are a type of Sprengel explosive in that the oxidizing agent (liquid oxygen) and the combustible substance or substances (solid carbonaceous matter alone or in combination with liquid hydrocarbons or even metallic powders and, at times inert absorbents) are brought together immediately before use. The components separately are non-explosive. The possibility of using liquid oxygen as one of the components of an explosive was perhaps first suggested by Sprengel, who stated that the *beau ideal* of a detonating explosive was a mixture of eight parts of liquid oxygen with one part of liquid hydrogen. Although the liquefaction of air[3] had been accomplished in the laboratory as early as 1877, it was not until 1895 that Linde[4] evolved a method whereby liquid air could be produced commercially. The separation of liquid oxygen from liquid air by the general method now extensively employed was suggested, if not actually accomplished in the laboratory, before Linde[5] demonstrated in 1902 that this could be economically done on a large scale.

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[1] Herman Sprengel: *Jnl.* Chem. Soc. London [N. S.] (1873) **11**, 796.

[2] Briefly, commercial explosives consist of a combustible substance and an oxidizing agent. Frequently an absorbent is required, which may be one of the active constituents or an inert substance, added solely for this purpose. The violent reaction between the combustible and the oxidizer at the moment of firing or detonation produces a large volume of highly heated gas, the sudden evolution of which gives rise to the explosive effect.

[3] Georges Claude: "Liquid Air, Oxygen and Nitrogen," 54–64. English Translation by Henry E. P. Cottrell, 1913.

The possibilities of liquid-oxygen or liquid-air explosives had been foreseen by Sprengel, and perhaps others at a later date, but they obviously could not be given serious consideration until liquid air could be produced in commercial quantities. However, it is not surprising that Linde[6] introduced a liquid-air explosive "oxyliquit" as early as 1897, or five years before liquid oxygen was available for this purpose. The early oxyliquit cartridges were said to consist of a pasty mixture of finely ground charcoal and petroleum contained in a paper wrapper. The cartridges were cooled to the temperature of the liquid air, dipped therein, and after becoming thoroughly saturated were loaded into the drill holes and fired as soon as possible. The most important, although not the first, practical use of oxyliquit was at the north end of the Simplon tunnel[7] in 1899. Although oxyliquit was said to compare favorably in some ways with other explosives used in this work it was finally abandoned.

Liquid-air Explosives

It is well at this point to call attention to the important difference between liquid-air and liquid-oxygen explosives. In the early work of Linde with oxyliquit, as well as much that immediately followed, liquid air was used, whereas in modern European and Pachuca practice liquid oxygen of a fair degree of purity is employed. It is said that a liquid-air explosive cannot be detonated unless the liquid air has been considerably enriched in oxygen. Apparently what little success was achieved with oxyliquit was due to the use of enriched air which probably resulted from the more rapid evaporation of the nitrogen in the less efficient containers then available. Even when enriched air was accidentally or intentionally used, denotation was uncertain and combustion of the carbonaceous absorbent incomplete. This gave rise to poor explosive effect and formation of poisonous carbon monoxide. Confusion has arisen through the making of direct comparisons between the performance of the early liquid-air explosives where the oxygen content of the liquid air varied greatly and modern liquid-oxygen explosives, employing oxygen of fair purity. Our experience at Pachuca, as recorded in this paper, is confined to the use of *liquid oxygen* of 94 to 95 per cent. purity and *not liquid air*. Liquid-oxygen explosives, comprising both the absorbent and the oxidizing agent (liquid-oxygen), have been abbreviated locally L.O.X., which designation, for the sake of brevity, will be used throughout this paper.

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[4] *Ibid.*, 88.

[5] *Ibid.*, 321.

[6] Richard Pabst: "Flüssiger Sauerstoff und seine Verwendung als Sprengstoff im Bergbau," München und Berlin, 1917.

[7] *Ibid.*, 12.

Following the earlier trials, and up to the beginning of the European war, little interest was manifested abroad in L.O.X., but important developments were taking place during this period which were to have an important influence upon the later development of this type of explosive. Since the early trial of oxyliquit at the Simplon tunnel, liquid oxygen had become a commercial reality and important improvements had been made in the equipment for producing both liquid air and liquid oxygen. Equally important were the great improvements which had been made in the containers for holding liquefied gases. With liquid oxygen instead of liquid air, and highly efficient metallic containers available, it is not surprising that the development of L.O.X. should have been rapid once there was the proper incentive, and this came with the beginning of the European war. The war had not long been under way before Germany discovered that she would be cut off from outside sources of fixed nitrogen, and indeed she had anticipated this, for her scientific men had developed methods of nitrogen fixation which assured the military a supply of nitrate explosives. Should the war last more than a few months it was evident that the precious supply of nitrogen would have to be carefully conserved and the use of nitrate explosives, for industrial purposes, either greatly restricted or cut off entirely. This condition resulted in immediate and great stimulation to the development of L.O.X. with the result that by the end of the war these explosives were extensively used throughout Germany and the occupied portions of France, in mining operations, tunneling, quarrying and, during the war, for the demolition of French metallurgical and manufacturing plants. Pabst[8] in a monograph, published in 1917, gives a list of 160 coal, metal and potash mines using L.O.X. The size of the oxygen plants ranged from 5 to 75 l. per hr. and the total hourly capacity of all these installations was 4862 l. The development of L.O.X. is here only sketched in hasty outline; the reader is referred for more definite information concerning their development and the prior state of the art to the papers and monographs by Diederichs,[9] Pabst[10] and Rice.[11]

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[8] *Ibid.*, 81–86.

[9] H. Diederichs: "Die Erzeugung und Verwendung flüssiger Luft zu Sprengzwecken." *Stahl und Eisen* (1915) **35**, 1146–1151, 1177–1181.

[10] Richard Pabst: "Flüssiger Sauerstoff und seine Verwendung als Sprengstoff im Bergbau," 1–101. München und Berlin, 1917.

[11] George S. Rice: "Development of Liquid Oxygen Explosives During the War." U. S. Bureau of Mines, *Tech.* Paper 243, pp. 1–44.

Outside of Europe, liquid-oxygen explosives have remained a curiosity until comparatively recently.

George S. Rice, chief mining engineer of the U. S. Bureau of Mines, on the entry of the United States into the war, with a knowledge of the conditions in Germany, foresaw that if the war continued long the same need might arise for a substitute for at least a part of the large amount of nitrate explosives normally required in the United States for non-military purposes. He accordingly initiated and caused to be carried forward at the Pittsburgh station of the Bureau of Mines preliminary tests with L.O.X. early in 1917. Aside from the practical demonstration of their effectiveness and more refined measurements of their power in the laboratory, he sought to develop suitable absorbents. These tests were carried on under the serious disadvantage that the Bureau did not have equipment for producing liquid oxygen and this could only be secured in small quantities at a distance of several miles from the various points at which the tests were made, through the courtesy of a company producing compressed oxygen in cylinders. They, however, indicated that, if it became necessary, L.O.X. could be used in the United States during the war, and considerable information was developed concerning absorbents, but they did not prove that L.O.X. could be economically used in mining operations in this country under peace conditions, for this could only be definitely demonstrated by the installation of a working unit at an operating mine.

It was known that L.O.X. had come into extensive and successful use in Germany during the war. But was this a temporary expedient which would, after the exigencies growing out of the war had passed, be abandoned? Again this question could only be answered by a working test. The early information which we secured upon the position of L.O.X. in Germany following the war was of a contradictory character. Moreover, the blasting problem in the German coal, iron, and potash mines is quite different from that encountered in metal mines, such as those at Pachuca. In general, larger drill holes are used and in many cases the ground is more readily broken. The larger drill holes are particularly favorable to L.O.X. since a cartridge of greater diameter can be used. It was, therefore, uncertain what the outcome would be with the smaller holes and harder rock at the mines at Pachuca.

The large-scale demonstration and commercial operation of the method at the mines of the Compania de Real del Monte y Pachuca, involving the mining of a large tonnage of ore, shows that the method is feasible under most of the conditions existing in the metal mines of the world, and in many cases gives promise of resulting in a distinct saving over the use of other explosives. The work at Pachuca is the first practical demonstration of the method outside of Europe. The taking up of this project, involving as it did considerable risk and expenditure, in the face of oft-repeated statements from certain quarters that L.O.X. was being discarded in Germany, came about through a fortunate train of circumstances.

Early in 1919 a process was under consideration in which it was thought that a large volume of gaseous nitrogen might be required and in looking for possible uses for the byproduct oxygen, the use of L.O.X. in Europe at once came to mind. Since the company was carrying on extensive mining operations it was evident that this would perhaps give the greatest outlet for oxygen, provided that it proved feasible to use L.O.X., and gaseous nitrogen could be economically produced with liquid oxygen as a byproduct.

This culminated in a definite suggestion that at least a preliminary investigation of L.O.X. be made. A short time before this a German manufacturer had shipped to Mexico a liquid oxygen plant of 30 l. per hr. capacity, a set of carrying and dipping containers, and a small stock of cartridges, on the chance that the equipment could be sold to one of the mining companies operating in Mexico. At about the time that the suggestion came from within the company, the Director of the Compania de Real del Monte y Pachuca was approached by Enrique Huber, agent of the manufacturer, for the purpose of negotiating a sale. Soon after this, L. A. Barton, then in the employ of the company as a geologist, who had been overseas with the United States Army and had had an opportunity of seeing L.O.X. in use at the Moyeuvre mine at Hayange, in Lorraine, gave the management the benefit of his observations.

George S. Rice, chief mining engineer of the U. S. Bureau of Mines, placed at our disposal all the data which he had secured abroad, shortly after the signing of the Armistice, and through the experimental work which the Bureau had carried on at Pittsburgh. He also arranged for a series of tests under the direction of J. W. Paul, chief coal mining engineer of the Bureau, early in January, 1921. These were witnessed by two of the company's engineers, who, after a study of all the data, submitted a report late in February stating that there appeared to be no insuperable difficulties in handling these explosives and with proper manipulation that they appeared to be fully as powerful as 40 per cent. dynamite. It was recommended that a liquid-oxygen plant be secured and a thorough investigation of the subject undertaken under the conditions existing at Pachuca. The German liquid-oxygen plant then in Mexico was accordingly acquired, installed, placed in operation and the tests were under way by August, 1921.

Manufacture of Liquid Oxygen

It is desirable to make one of the vital constituents of L.O.X. at or near the place used. Air is a mechanical mixture of gases, more or less contaminated by water vapor and dust particles. The composition of pure air is given in Table 1, and the important physical constants of its constituent gases are given in Table 2.

TABLE 1.—*Principal Constituents of Air*

Constituent	Per Cent. by Weight	Per Cent. by Volume
Nitrogen	75.74 to 75.58	78.115 to 78.026
Oxygen	22.92 to 23.08	20.913 to 20.999
Argon	about 1.3	about 0.94
Neon	about 0.00125	0.001 to 0.002
Helium	about 0.0000738	0.0004 to 0.00054
Carbon dioxide	about 0.04	about 0.03
Water vapor	Variable	Variable
Dust	Variable	Variable

NOTE.—The composition of air varies slightly from day to day and is also affected by the locality; the values given are the result of analyses made upon a large number of samples of clean dry air from points all over the world.

TABLE 2.—*Principal Physical Constants for Air and Its Important Constituent Gases*

Gas	Melting Point of Solid, Degrees C.	Boiling Point at Atmospheric Pressure (760 mm.), Degrees C.	Critical Temperature, Degrees C.	Critical Pressure, Atmospheres	Specific Gravity of Liquid at Boiling Point and 760 mm., Water = 1.00
Air	−218.00[a]	193.00[a]	−140.63[b]	37.17[b]	0.933[a]
Nitrogen	−210.03[b]	−195.78[b]	−147.13[b]	33.490[b]	0.8043[c]
Oxygen	−218.4[b]	−182.93[b]	−118.82[b]	49.713[b]	*1.142[c]
Argon	−189.30[b]	−185.84[b]	−122.44[b]	47.996[b]	1.3975[d]
Neon	−248.67[b]	−245.92[b]	−228.35[b]	26.86[b]	1.204[b]
Helium	Below −271.00[b]	−268.74[b]	−267.74[b]	2.26[b]	0.122[b]
Carbon dioxide	− 56.24[g]	− 78.2 sublimes[f]	+31.20[e, f]	34.6[e]	1.057[e] (at −34° C.)

NOTE.—Absolute zero = −273.09° C.

[a] These values have not been accurately determined, due to the difficulty of preventing separation of the oxygen and nitrogen.

[b] Onnes and co-workers at University of Leiden.

[c] Mathias.

[d] Baly and Donner.

[e] Cardoza.

[f] Zelery and Smith.

[g] Kennon and Robson.

* The 94 to 95 per cent. liquid oxygen produced at Pachuca has been found to have a specific gravity of about 1.1.

The general steps necessary to produce liquid oxygen of the purity usually employed for L.O.X. are:

1. *Dust particles, water vapor and carbon dioxide initially present in the air and the oil mist introduced through the lubrication of the compressors are eliminated so far as possible.* Efficient removal of these is of great importance, for at the low temperatures attained, water vapor and

carbon dioxide become solids, which together with the dust particles would rapidly clog the small tubes and passages of the apparatus, soon bringing the whole operation to a standstill.

2. *The purified air is liquefied by application of pressure and lowering of the temperature below the critical point.* All the so-called permanent gases have critical temperatures above which it is impossible to liquefy them by pressure alone. Table 2 shows that the critical temperature of air is −140.63° C. and the critical pressure is 37.17 atmospheres. This means that the air must be cooled to at least −140.63° C. before it can be liquefied by application of pressure. At this temperature it begins to liquefy when the critical pressure (37.17 atmospheres) is reached. The failure of early investigators to appreciate the importance of cooling below the critical temperature long retarded the production of liquid air. In practice, the pressure is produced by suitable compressors and the necessary lowering of the temperature is secured by expansion of the air. The two systems commonly in use are the Linde and the Claude. In the Linde system the air is expanded through a special nozzle whereas in the original conception of the Claude system the air was caused to do work in expanding by passing it through the cylinder of an expansion engine. As a matter of fact, expansion through a nozzle in conjunction with expansion in a cylinder is generally used in this system. The Claude system is characterized by its ability to operate at a much lower pressure than the Linde. Other and more elaborate cycles have been proposed which have promised to give important economies in the expenditure of power but as yet none of these is commercial. Most of the plants producing liquid oxygen for L.O.X. are of the Linde type although a few of the Claude type are in use.

3. *Liquid oxygen is separated from the liquid air by passing it through a rectifying still.* The boiling point of liquid oxygen at atmospheric pressure is −182.93° C. and of liquid nitrogen −195.78° C. This difference in the boiling points of the two liquids comprising liquid air renders it possible to effect a separation of liquid oxygen by employing precisely the same principle that is applied to separate mixtures of water and alcohol. While the design of the apparatus may vary somewhat, the general type of rectifying still, or still and rectifier, is common to both systems.

The equipment installed by the Compania de Real del Monte y Pachuca is of the Linde type, and was made by the Maschinefabrik Sürth, de Cie. Lindé, Sürth, b Coln. a Rhein, of the Deutsche Oxhydric, A. G. It has a rated capacity of 30 l. per hr. of liquid oxygen; but under the conditions obtaining at Pachuca its actual capacity is 25 l. per hr. The plant is housed in a wooden frame building, covered with corrugated sheet iron, 42.5 ft. long, 24.5 ft. wide, and 18 ft. high (13 by 7.5 by 5.5 m.). A 100-kw. motor drives the main compressor, the ammonia machine, and the caustic-solution pump; a 7.5-kw. motor drives the precompressor and a similar motor the cooling-water pump. The total average power consumption is 84 kw. The sequence of apparatus as shown by Fig. 3 is as follows:

Air at atmospheric pressure enters the system at a_1, passes through the carbon-dioxide separator *a*, caustic-solution separator *b*, single-stage compressor *d*, four-stage compressor d_1, oil and moisture separator *e*, nitrogen air cooler *f*, oil and moisture separator *g*, air rewarmer *h*, caustic dryer *i*, ammonia cooler *k*, and liquefier *l*.

Carbon-dioxide Separator

The separator *a* is a steel cylinder 3.77 ft. in diameter by 18 ft. high (1.15 by 5.5 m.), about half filled with 1- to 4-in. (25- to 102-mm.) lumps of coke. The caustic-potash solution (35 per cent., 37° Bé.) is circulated through the coke mass by a 1.5-in. (38-mm.) rotary pump *c*. The dust and much of the carbon dioxide in the air are removed at this point; whatever solution is carried over is trapped in the separator *b*. The strength of this solution is maintained by the caustic potash discarded from the caustic dryer *i*.

Air Compressors

Pachuca is at an altitude of 8000 ft. (2240 m.), therefore the precompressor *d* was provided to deliver air to the main compressor at or about normal atmospheric pressure (760 mm.). It is a horizontal, single-stage, double-acting, belt-driven machine having the following specifications:

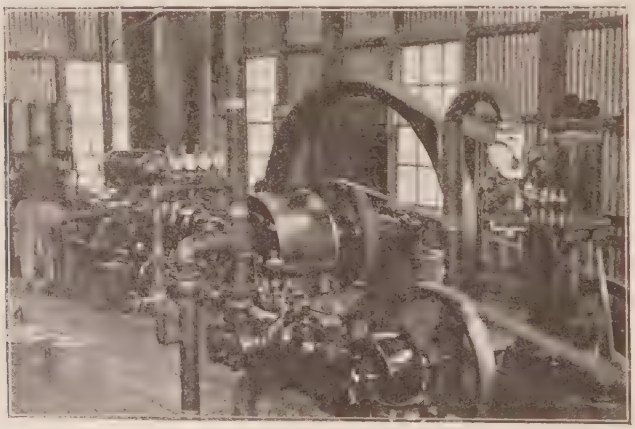


FIG. 1.—AIR AND AMMONIA COMPRESSORS.

Cylinder diameter, 15.75 in. (400 mm.); stroke, 5.91 in. (150 mm.).
Piston displacement, 266 cu. ft. per min. (452 cu. m. per hr.).
Rated output, 235 cu. ft. per min. (400 cu. m. per hr.) at sea level.
Discharge pressure, 6 lb. per sq. in. (0.42 kg. per sq. cm.).
Revolutions per minute, 200.
Power input of motor, 6.7 kw.

The cylinder is water cooled and the discharge pipe is coiled through a well of cooling water.

The main compressor d_1 is a horizontal, four-stage, belt-driven machine having the following specifications:

Cylinder diameters, 12.62 in., 9.19 in., 5.99 in. and 2.05 in. (320, 233, 152 and 52 mm.).
Discharge pressures, 55, 232, 796, and 2700 lb. per sq. in. (3.9, 16.3, 56 and 190 kg. per sq. cm.).
Stroke, 13.8 in. (350 mm.).
Piston displacement, 208 cu. ft. per min. (354 cu. m. per hr.).
Rated output, 162 cu. ft. per min. (275 cu. m. per hr.) at sea level.
Revolutions per minute, 145.
Power input of motor, 69.5 kw.

Each cylinder is water cooled and water-intercoolers located on the compressor frame cool air after each stage of compression. The high-pressure air leaves the compressor at a temperature of about 32° C. The cylinders are lubricated by force feed with an oil having a flash point of about 232° C.

Oil-and-moisture Separators

Separators *e* and *g* are simple steel traps, sometimes called purge bottles, 7 in. diameter by 44 in. high (178 by 1118 mm.); they are flushed through a needle valve every half hour.

Nitrogen Air Cooler

To assist in the elimination of water vapor, the temperature of the high-pressure air is reduced about 12° C. by means of the cool nitrogen gas from the liquefier *l*. The air passes through the inner and the nitrogen through the outer pipe of a counter-current concentric-pipe coil *f*.

Air Rewarmer

The rewarmer *h* consists of a series of thin, steel disks attached to the high-pressure air pipe. The air is rewarmed to approximately room temperature in order to facilitate the action of the caustic potash.

Caustic Dryer

The dryer *i* removes, as completely as possible, the remaining carbon dioxide and water vapor from the air. The six heavy steel cylinders, 7 in. in diameter by 60 in. high, (178 by 1525 mm.) are provided with removable steel baskets filled with caustic potash lumps about 2 in. (51 mm.) in diameter. A layer of cotton waste is placed on top the caustic potash to retain dust particles. Bolted covers permit inspection and removal of the baskets. Each cylinder is provided with a needle valve, which is opened for flushing every half hour. The air pressure drops from about 2700 lb. to about 2530 lb. per sq. in. (190 to 178 kg. per sq. cm.) in passing through this purifying system. It is necessary to shut down the plant every week for about 6 hr., to "condition" the caustic potash; this is done by removing the baskets and brushing and washing the lumps. The coarser pieces are returned to the baskets; the finer are used in maintaining the strength of the circulating solution in the separator *A*. After sufficient fresh caustic potash is added to fill the baskets, they are returned to the cylinders. The average consumption of commercial caustic potash for this purpose is 29 lb. (13.1 kg.) per 24 hr. The proper functioning of the caustic dryer is vital to the satisfactory operation of the liquefying plant. Caustic soda was tried for this purpose but there was more frequent freezing of the liquefier. The delay caused by the conditioning of the caustic potash could be avoided by providing an extra battery of dryers.

Ammonia Air Cooler

The purified high-pressure air is cooled by ammonia down to about −12° C. The object of this, in addition to securing precooling of the air, is to freeze out as much as possible of any remaining traces of moisture. The importance of as complete removal of dust particles, carbon dioxide, and moisture as possible has already been emphasized.

This equipment consists of a dry-compression ammonia machine k_1, and a counter-current concentric-tube cooler, or refrigerator, coil k. The compressed ammonia gas passes through the outer tube of the condenser k_2 where it is liquefied by means of the circulating cooling water. The liquid ammonia, controlled by a valve, enters the outer tube of the cooler. In the heat exchange that takes place between the liquid ammonia and the high-pressure air, the ammonia expands to the gaseous state and is returned to the inlet of the compressor k_1, which is a vertical, single-acting, belt-driven machine, having a cylinder 4.93 in. in diameter and a 5.91-in. stroke (125 by 150 mm.). It is rated at 8000 calories per hour (530 B.t.u. per min., or 2.4 metric tons of refrigerating capacity per 24 hr.). The water available for cooling has a temperature of about 25° C., whereas the manufacturer based the plant design upon cooling water at 11° C. A cooling tower was installed and the compressor speeded up from 160 to 180 r.p.m. With these changes the high-pressure air is cooled to −12° C; the power input is 3.5 kw. The gage pressure on the condenser is 153 lb. per sq. in. (10.8 kg. per sq. cm.), and the temperature 29° C. The suction gage pressure is 21 lb. per sq. in. (1.5 kg. per sq. cm.), and the temperature −14° C. The loss of ammonia averages 0.9 lb. (0.4 kg.) per 24 hr.; doubtless more careful attention to preventing leakage would decrease the consumption of ammonia.

Liquefier and Rectifying Still

Inside of a wooden container *l*, similar to an ordinary wooden tank, are located the counter-current heat exchange coil *o*, the cooling coil o_1, the vaporizer coil *q*, the expansion valve *r*, and the rectification column *s*. The sheath is 5 ft. in diameter and 10.6 ft. in height (1.52 by 3.23 m.). All the free space within it is packed with mineral wool which retards the passage of heat from the outside. The purified air entering the liquefier at *n* is under a pressure of 2530 lb. per sq. in. (178 kg. per sq. cm.) and at a temperature of −12° C. The gaseous nitrogen, intermixed with gaseous oxygen, passes out of the apparatus at *w* and the liquid oxygen is discharged at q_1.

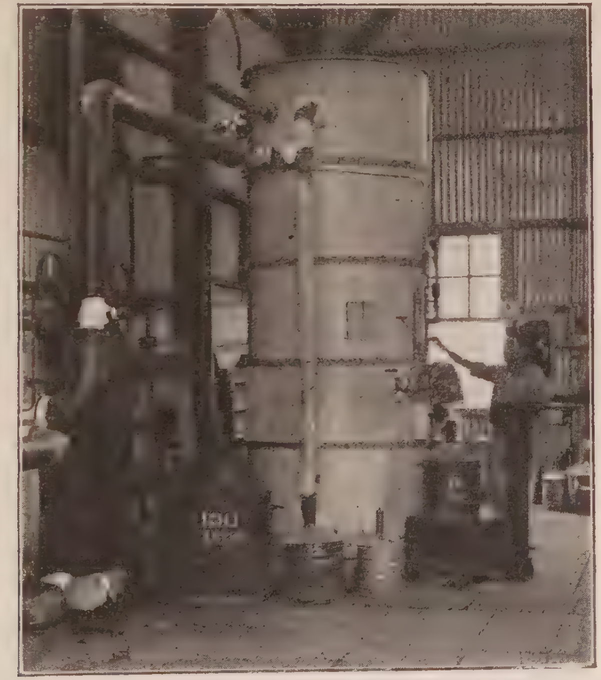


FIG. 2.—LIQUEFIER AND RECTIFYING STILL.

The heat exchanger *o* is a 100-ft. (30.5 m.) helical coil of 4-in. (102-mm.) lead pipe within which is a nest of twenty-four ¼-in. (6-mm.) copper tubes. The high-pressure air enters at *n*, passes downward through the copper tubes in *o*, leaving the exchanger at v_2, thence through a copper pipe to the cooling coil o_1 located at the top of the rectification column *s*. By reason of the cooling in *o* and o_1 caused by the gaseous nitrogen from the rectifier, the air enters the vaporizer coil *q* at a temperature of about −140° C.

From the vaporizer coil q, the air is conducted through a copper pipe to the expansion needle valve r, where it is expanded from a pressure slightly less than 2530 lb. per sq. in. down to a pressure of about 7 lb. per sq. in. (178 to 0.5 kg. per sq. cm.). This pressure is merely that required to force the nitrogen back through coils o_1, o, and the nitrogen air cooler f to the atmosphere. The additional refrigeration produced by reason of the Joule-Thompson effect of expansion through the orifice r liquefies the air at a temperature slightly above $-195.5°$ C.

The rectifying column s consists of a mass of ⅛-in. (3-mm.) copper tubes, arranged in the form of a vertical helix, and a number of baffles, the purpose being to cause the liquid to remain stratified so far as possible as it passes downward. The incoming high-pressure air in expanding through valve r is cooled and escapes through the heat exchanger o where it successively cools more of the incoming air. This cumulative effect continues until the temperature at which liquid air is formed is reached (about $-193°$ C.). The liquid air, as rapidly as formed, trickles down the rectifying column s and accumulates around the vaporizer coil q. The incoming high-pressure air within the vaporizer coil is warmer than the surrounding bath of liquid air; this temperature difference causes vapor to form which ascends against the descending liquid air. As the vapor tension of the nitrogen is higher than that of the oxygen, there is a gradual concentration of the oxygen in the vaporizer-coil bath. The rising vapors of oxygen intermingle with the descending, stratified stream of liquid air and the oxygen is condensed by it and falls back to the vaporizer-coil bath. The most of the nitrogen and part of the oxygen pass off in the gaseous state.

The cold-gas mixture passes over the cooling coils o_1, leaves the column at v, enters the bottom of the heat-exchanger o at v_2 and leaves o at n. During the counter-current flow, the cold gases absorb a great deal of heat from the incoming high-pressure air. The gases enter the air cooler f at a temperature of about $-2°$ C.; they escape into the atmosphere from w_4 at a temperature of about $+22°$ C.

In starting the plant, a slightly higher air pressure is required but after about 1.5 hr. normal operation is established and liquid oxygen begins to accumulate in the vaporizer-coil bath whence it is drawn off through the needle valve q_1 directly into the carrying containers. Notwithstanding that the construction of the liquefier and rectifying still is somewhat intricate and certain of the parts are inaccessible, the operation of this unit of the apparatus has proved to be simple. Regulation is accomplished by closely watching the high- and low-pressure gages p and p_1. The high pressure is controlled by the air compressors and the low pressure by the expansion valve r. The level of the liquid oxygen in q_2 is shown by the indicator p_2.

Thawing Device.—Despite the elaborate provisions, both mechanical and chemical, made for removing carbon dioxide and water vapor, minute quantities of these remain in the high-pressure purified air passing into the liquefier and rectifying still, with the result that a layer of the frozen solids slowly builds up at different points, particularly upon the inside of the copper tubes comprising the heat exchanger *o*. The layer of insulation thus formed, as it becomes thicker, reduces the rate of heat transfer between the incoming high-pressure air and the outgoing cold gases, until there is a marked decrease in the rate of production of liquid oxygen. To restore the efficiency of the apparatus, it becomes necessary to thaw out.

The counter-current concentric pipe coil *t* is so connected that hot water, at about 75° C., can be circulated through the outer pipe and compressed air at 75 lb. per sq. in. (5.3 kg. per sq. cm.) passed through the inner. The hot water is supplied by a boiler 20 in. in diameter by 32 in. high (50.8 by 81.3 cm.). The air produced by the oxygen-plant compressors passes through the caustic dryer before entering the heating coil *t*, where its temperature is raised to about 70° C. The tubing connecting the various gages is removed, the blowoff cock *x* in the rectifying column is opened; the nitrogen line w_2 and the expansion valve *r* are closed. The three-way thawing bypass is turned so that the air will pass through the caustic dryer, into the heating coil. Thence the hot dry air enters the apparatus at *j*, passing through the heat exchanger *o*, it enters the rectifying column at *v* blowing out through the gage pipes and blow-off cock *x*. This operation is continued from 3 to 6 hr., or until warm dry air blows out continuously, from these openings. The valve j_1 is opened and the valve *j* closed, thus causing the hot air to pass through the perforated pipe inside of and at the bottom of the wooden shell of the apparatus. The last step in the operation is to remove the moisture remaining in the piping by high-pressure dry air. The thawing bypass is closed and the pressure at the compressor is raised from 75 lb. per sq. in. (5.3 kg. per sq. cm.) to about 1000 lb. per sq. in. (70 kg. per sq. cm.). The high-pressure blow-off valve *z* is then quickly opened and closed, until there are no signs of moisture in the escaping air.

The entire thawing operation requires from 6 to 10 hr. During the early months of operation, thawing had to be resorted to about every two weeks. More recently, using only caustic potash of suitable size in the air dryers, and paying more attention to the frequent draining of the oil-moisture separators *e* and *g* and the caustic dryer *i*, it is possible to operate the plant for 8 weeks without thawing.

Labor Required

A native operator and one native helper are provided for each shift, but where higher grade labor is available one man per shift will doubtless be able to take care of a much larger installation. The plant is not as complicated to operate as might be inferred from the detailed description of the various steps. It has been demonstrated that the better class of native labor can be rapidly trained to care for the routine operations. A new operator, who preferably should have had previous experience in the operation of air compressors, can be trained in about a week.

Performance

The plant, as previously stated, has a rated capacity of 30 l. per hr. (33.54 kg., 73.95 lb.) and an actual capacity under the conditions obtaining at Pachuca, of 25 l. (27.5 kg., 60.6 lb.) of liquid oxygen. The manufacturers' specifications provide for 44 gal. of cooling water at 11° C. per min., whereas the mine water available for this purpose is at about 25° C.; this doubtless is one of the important causes of the decrease in capacity. It was thought that, with continuous operation, the output might improve but this did not prove to be the case. The makers state that a higher purity of the liquid oxygen and additional recovery of oxygen in the gaseous form can be secured by modifying the liquefier and rectifying still.

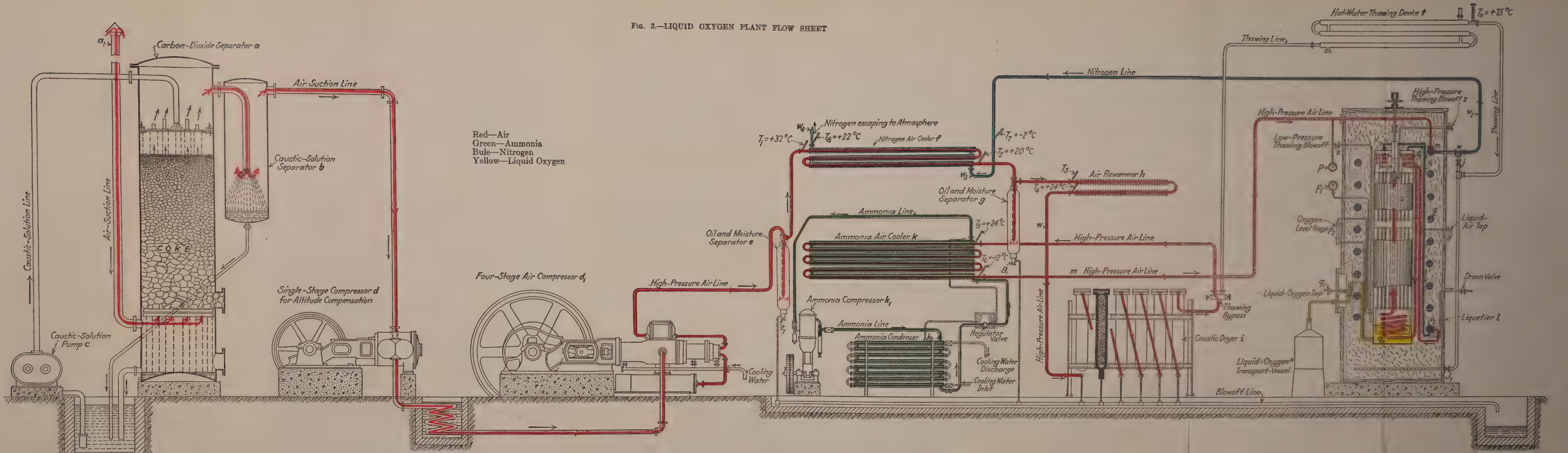
It is now generally thought that liquid oxygen of about 95 per cent. purity meets the average requirement for L.O.X., and our experience tends to bear this out. The liquid oxygen produced at Pachuca contains, on an average, approximately 94 per cent. of oxygen, the balance being largely dissolved nitrogen, with possibly some of the rarer gases. The gases leaving the liquefier and rectifying still contain 85 to 88 per cent. of nitrogen and 12 to 15 per cent. of oxygen.

George S. Rice has called our attention to tests made at Pittsburgh by the Bureau of Mines with liquid oxygen that is a byproduct of the manufacture of argon and neon by one of the electric-lamp companies at Cleveland, Ohio. Cartridges prepared with this oxygen detonated in the open to a degree not experienced with the liquid oxygen previously used in the Bureau's experimental work, which had a purity of about 97 per cent. J. W. Paul, who conducted the tests, thinks that this special property is due to the greater purity of the oxygen; it is, therefore, conceivable that for some blasting purposes purer oxygen might prove advantageous.

Repairs

From August to December, 1921, the plant was operated one shift; from January to May, 1922, two shifts; and since June 1, three shifts per day. The only repair work of importance was occasioned by the choking of several of the small tubes in the liquefier and rectifying still. This was probably caused by improper attention to the draining of the oil-moisture separators and the caustic dryer and to the use of caustic soda in place of caustic potash. The item for repairs, which appears in the cost of production, is therefore an estimate.

FIG. 3.—LIQUID OXYGEN PLANT FLOW SHEET



Red—Air
Green—Ammonia
Bule—Nitrogen
Yellow—Liquid Oxygen

Cost of Liquid Oxygen

Detailed costs of producing liquid oxygen, under present conditions at Pachuca are as follows: Actual capacity of plant 25 l. per hr. at 95 per cent. running time = 570 l., 627 kg. or 1382 lb. per 24 hr. Plant operated 6 days per week and the cost of power 0.63 c. per kw.-hr. Capital charges and overhead expense are not included.

	Kilowatts	Cost per 24 Hr. Dollars U. S. Cy.	Cost per Liter Cents, U. S. Cy.
Power, precompressor	6.7		
Main compressor	69.5		
Ammonia compressor	3.5		
Caustic pump	0.6		
Cooling-water pump	3.7		
Total	84.0		
2016 kw.-hr. (3.53 kw.-hr. per l.) at 0.63 c.	....	12.70	2.23
Labor, 3 operators at $3	$9.00		
3 helpers at $1	3.00		
Two shifts per week, changing chemicals, minor repairs, and thawing	1.50		
		13.50	2.37
Chemicals, caustic potash, 29 lb. at $0.11	3.19		
anhydrous ammonia, 0.9 lb. at $0.65	0.59		
		3.78	0.66
Lubricants, 3.4 gal. oil at $1	3.40		
Grease and waste	0.20		
		3.60	0.63
Miscellaneous supplies, packing, tools, lights		1.00	0.18
Repairs to machinery and oxygen containers (estimated)		4.00	0.70
Total direct, or operating cost		38.58	6.77
Total direct, per kilogram of liquid oxygen			6.16
Total direct, per pound of liquid oxygen			2.79

With a large plant, say 100 l. per hr., there would be an important reduction in the per-liter cost for labor and power. It is estimated that at the same unit prices taken for the small plant, the total direct, or operating, cost would not exceed 4.5 c. per l. (1.86 c. per lb.).

TRANSPORT AND DIPPING CONTAINERS

The boiling point of pure liquid oxygen at 760 mm. is −182.9° C.; therefore under even the most favorable conditions, which would be winter weather in a northern latitude, there would be a great temperature difference between the liquid oxygen within the container and the outside air. The temperature of the air at Pachuca has a mean average of about 25° C. Therefore, if undue evaporation is to be prevented, the containers must be insulated as completely as possible against the passage of heat through their walls. Closed containers cannot be used, for, with the best of insulation, there is an appreciable transfer of heat from the outside to the liquid within, which causes it to vaporize and develop dangerous pressures if confined. The modern metal-walled vacuum container, having a small neck and a charcoal chamber, fulfills the condition of having a low oxygen loss and is sufficiently rugged to withstand the rather rough handling to which these containers are subjected around a mine. Attention is called to the fact that a satisfactory metal container had not been developed for liquefied gases at the time that oxyliquit explosives were used at the Simplon tunnel. The lack of a proper container was doubtless one of the important factors contributing to the failure of these explosives on this work. As efficient and rugged containers are necessary to the successful use of L.O.X., the fundamental principles involved and the development of the present-day container are briefly outlined to show the relationship of the container to the whole development of L.O.X.

Many years ago, certain physicists,[12] probably Dulong and Petit were the first, discovered that the passage of heat through glass by conduction was greatly reduced by use of a vacuum wall. Later, it was discovered that a vacuum was no barrier to the passage of radiant heat but that this was largely influenced by the nature of the solid surfaces. Although Dewar used vacuum insulation for calorimeters as early as 1874, it was not until 1887 that d'Arsonval made practical use of glass-walled vacuum containers for holding liquefied gases, thereby reducing evaporation losses to one-tenth what they were in plain glass vessels. The next improvement, which again made use of old knowledge, was introduced by Dewar, about 1892, when by silvering the inner glass walls, he greatly lessened the transfer of radiant heat thereby reducing evaporation losses to 1/200 of what they are in plain glass containers. The vacuum silvered-wall glass container, when properly constructed, is highly efficient and well suited for use in a laboratory, but being very fragile, is unsuited for the rough handling it would receive in a mine.

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[12] A good historical account of the development of vacuum containers is given in "Liquid Air, Oxygen and Nitrogen" by Georges Claude. English translation by Henry E. P. Cottrell, London, 1917. This information is given concisely and even more completely by J. L. Sherrick of the U. S. Bureau of Mines in an unpublished paper entitled "The Development of Vacuum Walled Containers for Liquefied Gases."

All metals conduct heat much more readily than glass and are porous and permit gases, including air, to pass slowly through them, thus rapidly destroying the vacuum, whereas glass does not. The first fault is not serious but the second is vital and until some way was found of preserving a high vacuum between metal walls, a satisfactory metal container for liquefied gases could not be constructed. Dewar, in the course of his experiments with liquefied gases discovered that cocoanut charcoal, at the temperature of liquid air, had the property of adsorbing large amounts of various gases, including air, so completely as to produce a vacuum through which an electrical discharge would not pass. He secured an English patent on this discovery in 1904, and early applied the idea to the production of the high vacuum required in the then well known glass containers for liquefied gases; but it was not until 1906 that he pointed out that a high vacuum could be satisfactorily maintained by this means between the walls of a metal container. Later in 1904, A. J. Levy took out a French patent in which the use of absorbent material in the vacuum chamber of metallic liquid-air containers was claimed. It is not clear who first began the manufacture of metallic liquid-air containers, possibly it was Claude, for he states that he made copper receptacles of 5 l. capacity which preserved liquid air as long as a week but gave up making them on account of infringing Dewar's patent of 1904. In any event, these were not available until some time later than 1904.

The transport and dipping containers used at Pachuca were manufactured by the Gefässfabrik Ahrendt Co. for the Sprengluft-Gesellschaft. The construction of these is shown, in detail in Figs. 4 and 5, respectively. They are of the same type as previously described by Diederichs,[13] Rice,[14] and Pabst.[15]

The transport container, Fig. 4, made in various sizes, consists of a spherical, narrow-necked flask of spun brass surrounded by a slightly larger flask, so as to leave a free space between the bodies and necks of the two flasks. The neck of the inner flask is joined by a vacuum-tight joint to the inside of the neck of the outer flask near the top, but in such a way that when the assembly is tilted from the vertical position the inner flask, which normally does not touch the outer flask, comes into contact, over a small portion of its area, with the outer flask. To the outside of the bottom of the inner flask is attached a receptacle *f*, which conforms to the bottom of the outer flask but does not touch it. Five ¼-in. (6 mm.) holes covered with 100-mesh screen connect the chamber thus formed with the free space between the two flasks, which is evacuated before the flask is placed in commission. The chamber is filled with charcoal,[16] which has been heated and subjected to a high vacuum, in order to give it the highest possible adsorptive power. A short length of ¼-in. (6 mm.) lead tubing for connecting the vacuum pump is soldered to the opening *d*, which communicates with the space between the two flasks. After the vacuum is established, the lead tube is closed off by bending over and soldering. The stub is protected by a steel cap *e*. The flask assembly is placed inside of a heavy galvanized-iron outer protecting sheath *b* of the section shown in the drawing. The free space between the outer flask and the sheath is packed with ordinary wood excelsior. Carrying handles are firmly riveted to the outer shell near the top. The smaller sizes have a handle, similar to a bucket handle, provided for convenience in handling.

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[13] H. Diederichs: "Die Erzeugung und Verwendung flüssiger Luft. zu Sprengzwecken." *Stahl und Eisen* (1915) **35,** 1151.

[14] George S. Rice: "Development of Liquid Oxygen Explosives During the War." U. S. Bureau of Mines *Tech. Paper* 243, p. 32.

[15] Richard Pabst: "Flüssiger Sauerstoff und seine Verwendung als Sprengstoff im Bergbau," 52, 53, 56. München und Berlin, 1917.

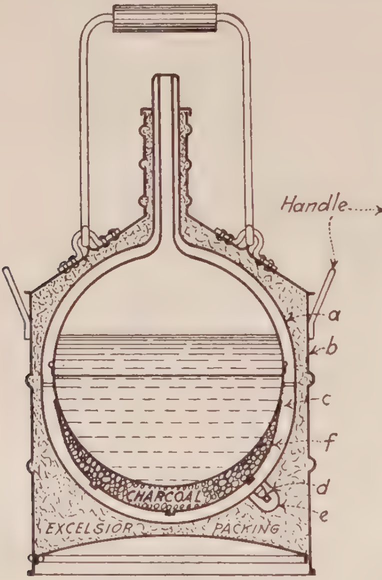

FIG. 4.—TRANSPORT CONTAINER.

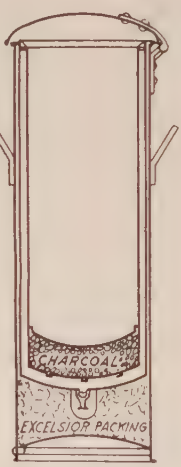

FIG. 5.—DIPPING CONTAINER.

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[16] It is most important that charcoal which contains metallic oxides be avoided. A number of explosions of liquid-oxygen containers made by an Austrian manufacturer were traced to the presence of iron oxide in the charcoal. The oxides of other metals, such as copper, zinc, etc., when finely divided and coating the charcoal are also likely to cause an explosion. Normally, the liquid oxygen does not come into contact with the charcoal but should a leak develop in the inner flask the liquid oxygen at once saturates the charcoal. See, S. Wöhler: *Zeits. Komprimflüss Gase*, **20,** 109–112, 121–124, 133–137.

The double-walled neck of the flask is made long and slender, 8 in. long by 0.56 in. inside diameter (203 mm. by 14 mm.), to minimize the loss of liquid oxygen by evaporation. Pouring through such a neck would be difficult but for the ingenious arrangement described. So long as the container remains in a vertical position, the inner flask is completely surrounded by the high-vacuum space *c*, which affords good insulation against the transfer of heat to the inner flask containing the oxygen, but when the container is tilted, as in pouring, the inner flask swings and a small area of its surface touches the outer flask and sufficient heat is transferred to cause a small amount of the oxygen to vaporize, thereby creating sufficient pressure within the flask to cause the oxygen to flow smoothly through the narrow neck When the container is returned to the vertical position, the inner flask assumes its normal position out of contact with the wall of the outer flask. As there is always a slight evaporation of the liquid oxygen, care is taken never to close the neck of the flasks tightly in order to avoid development of a dangerous pressure.

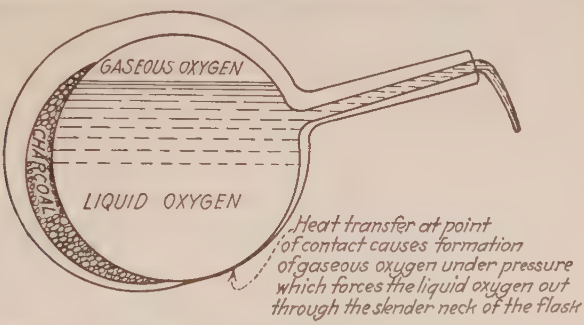
GASEOUS OXYGEN
CHARCOAL
LIQUID OXYGEN
Heat transfer at point of contact causes formation of gaseous oxygen under pressure which forces the liquid oxygen out through the slender neck of the flask

FIG. 6.—POURING POSITION OF TRANSPORT CONTAINER.

The dipping containers, Fig. 5, although of different section, are of the same construction as the carrying containers, except that the inner flask is held rigidly in place and the top is provided with a loose fitting cover.

There were supplied with the plant 270 transport containers having a capacity of 5 l. (5.71 kg., 12.59 lb.), 20 having a capacity of 15 l. (17.18 kg., 37.76 lb.) and 2 having a capacity of 100 l. (114.15 kg., 251.7 lb.). Much larger transport and storage containers have been used in Europe. Special tanks mounted upon motor trucks, similar to those used for transporting oil in this country, are not uncommon. Railway tank cars have been proposed, if not actually used, for transporting large amounts of liquid oxygen considerable distances. Large containers are especially desirable for storage as the percentage of evaporation loss decreases with the size. There are, in general, rather severe limitations upon the size of transport and dipping containers that can be conveniently used in a mine. We have found that 5-l. containers are well adapted for use in most of the stopes, for they can be readily handled through ladderways and small openings; a 10-l. container may prove advantageous in stopes where the working faces are fairly accessible. The 15-l. containers are used in drifts and crosscuts. For large tunnel work, 25-l. containers or larger, set in a tilting cradle mounted on a car or set of car wheels, would be the best practice. The 100-l. containers were intended for storage at the liquid-oxygen plant or for transport to the mine mouth, where their contents would be transferred into smaller containers for distribution to the working faces. Complete data for transport containers of various sizes is given in Table 3.

TABLE 3.—*Transport Containers*

Capacity			Height to Top of Neck		Outside Diameter		Weight Empty		Evaporation Loss per Hour		
Rated	Actual										
Liters	Kg.	Lb.	Mm.	In.	Mm.	In.	Kg.	Lb.	Kg.	Lb.	Per Cent.
5	6.2	13.7	550	21.7	285	11.2	6	13.2	0.040	0.09	0.65
10	11.7	25.8	568	22.4	335	13.2	10	22.0	0.060	0.13	0.51
15	17.2	38.0	610	24.0	380	15.0	14	30.9	0.075	0.16	0.44
25	27.0	59.6	800	31.5	455	17.9	20	44.1	0.090	0.20	0.33
50	55.0	121.5	900	35.5	600	23.6	35	77.2	0.115	0.25	0.21
100	110.0	243.0	1050	41.4	675	26.6	60	132.2	0.125	0.28	0.11

NOTE.—Capacity and evaporation loss as determined at Pachuca.

The dipping containers supplied with the plant were: 120 of 3.9 in. diameter, 10 of 7.9 in. diameter, and 20 of 9.9 in. diameter. The 3.9-in. size is convenient for small stope rounds, bulldozing, and the starting of chutes; the larger sizes are used in the drifts and crosscuts and for large stope rounds.

Owing to the relatively large surface of liquid oxygen exposed to the air while the dipping container is in service, the evaporation loss is heavy; but fortunately the soaking of a lot of cartridges requires less than 10 min., after which the remaining liquid oxygen is returned to the transport containers by means of sheet-iron funnel having a long stem. Recently, the funnels have been constructed with a double wall, the space between the walls being filled with kieselguhr; this has somewhat diminished the loss occurring during this transfer. The necessity for care during the soaking operation cannot be too strongly emphasized; at best considerable oxygen is lost through spillage and evaporation, and if the men are careless the loss may become excessive.

At first, the returning of the unused oxygen from the dipping containers to the carrying containers appeared to be hazardous, as the oxygen so returned usually contains more or less absorbent from the leakage and accidental breakage of the cartridges. It was thought that the proximity of an open light might ignite the mixture, causing it to burn violently or even explode; furthermore, there appeared to be some chance that the rapidly evaporating oxygen might carry with it extremely finely divided carbon black which would give with the gaseous oxygen an explosive mixture.

Tests made by the U. S. Bureau of Mines[17] at Pittsburgh indicate that the hazard from any of these causes may be small; however, the subject is so important that the results of these tests cannot be taken as being final. Liquid oxygen containing 10 per cent. of carbon black was placed in a flask fitted with a rubber cork, in which was inserted the stem of a glass funnel. A white filter paper was fastened over the large open end of the funnel so that all the oxygen that evaporated had to pass through the filter paper. After 4 hr. the filter paper remained perfectly white, indicating that none of the carbon black was carried over. Attempts were made to ignite liquid oxygen containing 10 per cent. of carbon black by various methods. In one test a hot wire was placed just below the rim of the container and in another the hot wire was at the surface of the mixture. Two tests were made with an electric igniter firing across the mouth of the container, one test with the igniter 1¼ in. below the rim, and one test with the igniter at the surface of the mixture; in no case did the mixture ignite. In three tests 1¾ l. of the mixture was poured over a lighted carbide lamp, without showing any tendency to ignite; after the oxygen evaporated, there was some carbon black on the reflector and although the lamp was covered with a layer ¼ in. thick it was still burning. These tests demonstrate that liquid oxygen-carbon black mixtures are, to say the least, difficult to ignite. Mr. Allison, of the Bureau of Mines, states that the reason that ignition did not take place in any of these tests was that the large excess of liquid oxygen kept the whole of the mixture below its kindling temperature. Complete data for dipping containers of various sizes is given in Table 4.

TABLE 4.—*Dipping Containers*

Capacity			Outside Diameter		Inside Diameter		Weight Empty		Evaporation Loss per Hour		
Number of Cartridges 1.1 by 12 in.	Oxygen Content		Mm.	In.	Mm.	In.	Kg.	Lb.	Kg.	Lb.	Per Cent
	Kg.	Lb.									
6	2.6	5.7	145	5.7	100	3.9	4.1	9.0	0.20	0.44	7.69
15	6.0	13.2	200	7.9	150	5.9	7.3	16.1	0.40	0.88	6.67
27	10.7	23.6	255	10.0	200	7.9	10.3	22.7	0.60	1.32	5.61
40	16.6	36.6	310	12.2	250	9.8	14.6	32.2	0.85	1.87	5.12

NOTE.—The outside depth of all the containers is 500 mm. (19.7 in.); inside, 340 mm. (13.4 in.).

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[17] George S. Rice: Personal communication, covering tests made by Messrs. Allison, Gardner and Fene, at Pittsburgh in May, 1922.

Life and Repair of Containers

The liquid-oxygen containers were placed in service during August, 1921, or about one year after they had left the factory. At this time, 18 per cent. of them were found to have a poor vacuum. Of those in first-class condition, about one-half have been in continuous service for 10 months and the balance for 3 months. It has been stated that metal vacuum containers for liquefied gases remain efficient for about one year, after which it is necessary to remove the adsorbed gases from the charcoal and restore the vacuum in the insulating chamber. Although none of the containers has been in actual use for this length of time, there is every indication that they will give at least this service before requiring restoration. The brass of which the outer and inner flasks are made is less pervious to air than many other metals and alloys, but nevertheless once the charcoal becomes saturated with air the vacuum rapidly decreases and the container soon becomes inefficient. The period of service is therefore determined by the time that the charcoal remains active. A personal communication from the U. S. Bureau of Standards states that vacuum containers should be evacuated to a pressure of 0.00001 mm. of mercury or lower. With higher pressures, the insulating properties of the container are correspondingly lowered. For example, at a pressure of 0.01 mm., the rate of heat transfer might be four times as great. The restoration of a container can be accomplished in less than an hour. Some loss of containers through falling rock and other causes is to be expected, but so far only ten have been badly damaged in underground use.

Selection of Absorbents

We have so far dealt with the production and handling of the oxidizing agent; the selection and preparation of the combustible substance that forms the other component of L.O.X. is equally important. In the case of the oxidizing agent we are limited to the consideration of a single substance—liquid oxygen, the production of which is purely a mechanical problem. The devices used and the underlying physical principles may appear somewhat complex, but there is the great advantage that all efforts can be centered upon the economical production and handling of a single substance.

The case with the combustible component is quite different. Once a satisfactory substance or combination of substances has been selected, the forming of it into cartridges is a simple matter. The number of single substances, and possible combinations of them, appears to be almost endless, and herein lies the difficulty. Fortunately the problem can be greatly simplified by enumerating the conditions that should be met by a satisfactory combustible, or absorbent, as it is generally termed.

1. The cost should be low. This condition is perhaps best met by a substance which is locally available or which can be produced on the ground from low-cost material which, if not available in the district, can be shipped in on a low freight rate. The good absorbents, in general, are bulky and freight charges may prove excessive. While L.O.X. have some important advantages over other explosives, as well as disadvantages, the determining factor as to whether or not they are used will, in most cases, be the possible saving in cost over other explosives. The cost of the absorbent has a most important influence on costs and a low cost efficient absorbent is, therefore, of the greatest moment to the future of L.O.X.

2. The absorbent should preferably be a single uniform material; the difficulty of thorough mixing, as well as all chance of later segregation of the ingredients, is thereby avoided. If it is necessary to use several materials, these should be such as are easily mixed and not likely to segregate, as might be the case when a heavy powder is mixed with a light one. The material should lend itself to easy and uniform filling of the cartridge wrapper.

3. There should be sufficient absorption and retention of liquid oxygen by the cartridge so there is complete combustion at the moment of detonation, despite the rapid evaporation of oxygen that unavoidably takes place during the interval between loading and firing. On this depends the avoidance of the formation of poisonous carbon monoxide and, to a large extent, the explosive strength of the cartridge.

4. When small-diameter drill holes are used in tough ground and a powerful cartridge is desired, there should be the maximum amount of active combustible per unit of cartridge diameter it is possible to secure and still fulfill the conditions enumerated under 3.

5. There should be unfailing detonation in order that there be no missed holes. While missed holes with L.O.X. are not dangerous, they tend to lessen efficiency.

6. The ingredients should be of such a nature that they do not evaporate or ooze through the wrapper if the cartridges are stored before using. This would appear to be a difficulty that might arise if oils were used in the mixtures, but probably could be avoided by making up the cartridge with the other ingredients and dipping them in the oil at the working face just prior to soaking in the liquid oxygen.

General Considerations and Testing of Cartridges

L.O.X. are unlike other explosives in that a time factor is involved in their use. Dynamite is a finished product when it leaves the hands of the manufacturer, and all that is required to explode it is proper detonation. It can be transported long distances, stored under proper conditions for some time, and drill holes can be loaded long in advance of firing, if there is any advantage in so doing; therefore no time factor need be considered in connection with its use. On the other hand, with L.O.X., it is only the cartridge of absorbent material before soaking in liquid oxygen that can be stored or transported any distance. After the cartridges are removed from the liquid oxygen, it is necessary to load and fire the holes promptly on account of the rapid evaporation of the oxygen. This limitation, which is one of the disadvantages of L.O.X., is in a measure compensated by the fact that the cartridges are harmless, except for a period of a few minutes after removal from the liquid oxygen. It is important, therefore, that the absorbent used should have the ability of taking up a large proportion of liquid oxygen, so as to supply the excess necessary to compensate for the evaporation that unavoidably takes place, under the most favorable conditions, in the interval between removal of the cartridges from the liquid oxygen and detonation.

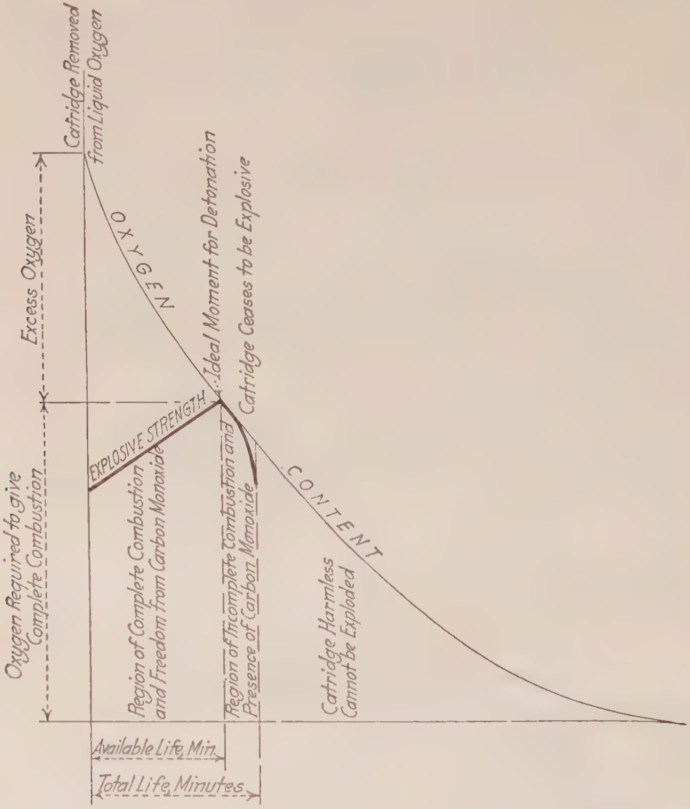

FIG. 7.—RELATION OF LIFE TO OXYGEN CONTENT OF CARTRIDGE.

Carbonaceous matter comprises the wrapper and the active absorbent material of the cartridges; therefore, reduced to the simplest terms, the elements to be considered in the reactions taking place with oxygen at the moment of explosion are carbon, hydrogen, and sulfur. The carbon and hydrogen are the normal and desired constituents of the absorbent, whereas the sulfur is an impurity likely to occur in carbon black, the most generally used absorbent. The ash, which represents the inorganic material present in the absorbent and wrapper, is always present to a certain degree but does not enter into the chemical reactions. Inert organic absorbents, such as infusorial earth, which are sometimes added to reduce the carbon content of the cartridge and increase the oxygen content, are likewise inert chemically. It has been assumed that the ideal moment for detonation is when there is just sufficient oxygen in the cartridge to give complete reaction between it and the combustibles of the cartridge. The carbon should be oxidized to carbon dioxide and to fulfill this condition requires 2.66 parts of liquid oxygen for each part of carbon. The result of reaction with the hydrogen is water, and each part of hydrogen requires 8 parts of oxygen. Sulfur, which is an accidental constituent, would probably be oxidized to sulfur trioxide and 1.5 parts of oxygen would be required for each part of sulfur.

Cartridges of L.O.X. may be detonated after removal from the liquid oxygen and for several minutes past the ideal moment for detonation. If the cartridge is exploded before the ideal moment of detonation, the explosive effect is less powerful by reason of the energy consumed in evaporating the excess of oxygen. If the cartridge is exploded after the ideal moment of detonation, there is the risk of incomplete combustion of the carbon with formation of noxious carbon monoxide.

The "*life*" of the cartridge, as we have used it in this paper, is the period of time, expressed in minutes, that elapses from the time the cartridge is removed from the dipping container until it ceases to be explosive. This might also be termed the "explosive life."

The "*available life*" is the period of time, expressed in minutes, which elapses from the time the cartridge is removed from the dipping container until the point is reached where there is the proper proportion of oxygen to give complete combustion of the active absorbents and the paper of the wrapper. The available life, which is the important consideration, is influenced by the nature of both the absorbent and the wrapper, and the conditions under which its determination is made. The test of available life, as made in air, although very simple, affords a reasonably satisfactory basis for the rapid comparison of absorbents and wrappers. The cartridge is weighed upon an ordinary postal scale, soaked in liquid oxygen until thoroughly saturated, and again placed on the scale. The weight is immediately noted and thereafter, at intervals of 1 min., until the oxygen

TABLE 5.—*Typical Evaporation Tests for Determining the Available Life of L.O.X. Cartridges*

Test number	101	102	103	104	105
Where made	Open air	Open air	Open air	Open air	Open air
Absorbent	Mexican carbon black	Mexican carbon black	Sprengluft carbon black	Sprengluft carbon black	Sprengluft carbon black
Size of cartridge	1.1 by 12 in.	1.38 by 12 in.	1.1 by 12 in.	1.38 by 12 in.	1.57 by 12 in.
Wrapper, "Kraft," paper, 45 lb. to ream, not perforated	Double	Double	Double	Double	Double
Weight of wrapper, in grams	3.6	4.6	3.6	4.6	5.0
Weight of absorbent, in grams	40.0	58.4	40.0	58.4	77.0
Weight of cartridge before soaking in liquid oxygen, in grams	43.6	63.0	43.6	63.0	82.0
Weight of cartridge after soaking in liquid oxygen, in grams	220.0	319.0	225.0	327.0	426.0
Weight at ideal moment for detonation, in grams	154.3	223.8	154.3	223.8	292.7
Available life, or time to reach ideal moment of detonation, in minutes	7.5	8.25	9.5	10.25	11.5
Ratio of oxygen to absorbent in cartridge when saturated	4.05	4.07	4.15	4.20	4.20
Calculated ratio of oxygen to absorbent in cartridge at ideal moment for detonation	2.5	2.5	2.5	2.5	2.5

content of the cartridge has fallen well below the amount calculated to be necessary for perfect combustion.

On account of the lack of definite information, at the outset, as to the chemical composition of the absorbents and wrappers used, it was necessary, in calculating the weight of oxygen to give complete combustion with any particular absorbent and wrapper, to make assumptions which are somewhat inaccurate. It was assumed that the paper wrappers are cellulose ($C_{12}H_{20}O_{10}$) and that each gram would require 1.18 gm. of oxygen for complete combustion. The various carbon blacks used are assumed to be pure carbon requiring 2.66 gm. of oxygen for the complete combustion of each gram. Paper is not strictly cellulose; and carbon blacks, in addition to carbon, contain varying amounts of tarry matter, hydrocarbons, and adsorbed gases as well as sulfur and a small amount of ash. In practice, after a wrapper and absorbent have been adopted, elemental analyses can be made of each and more accurate factors developed for calculating the oxygen requirement.

The available life in the drill hole is generally from 2 to 3 min. longer than indicated by the determination made in the air. Effective breaking of the ground has been found possible as long as 2 to 4 min. after the ideal moment for detonation, but carbon monoxide may be formed through incomplete combustion of carbon. Perfect practice, which of course can only be approximated, is to fire exactly at the end of the available life, thereby securing maximum breaking effect with no formation of noxious gases. To the inexperienced, it might appear difficult even to approximate this but experience has shown that this is not the case; in the mining operations it has not been difficult to secure good breaking effect without formation of noxious gases.

After 30-min. evaporation, the cartridges used at Pachuca contain little oxygen and are, therefore, non-explosive and perfectly safe. If desired, such a cartridge can be removed from the drill hole and used after re-soaking. The strength of cartridges is now determined by using them for blasting typical drift and stope rounds. This method, although far from perfect, has proved helpful in developing the present cartridge. Again, it was a case of securing practical results in advance of purely scientific study of the cartridge. It is expected that institutions, such as the Bureau of Mines, that are especially equipped for the refined study of explosives will enlarge the scope of their study of L.O.X. Table 5 gives the results of a few evaporation tests, which show the influence of the diameter of the cartridge on the available life. Evaporation tests have been made on cartridges made from Pachuca materials by Messrs. Gardner, Allison, and Fene, at the Pittsburgh station of the Bureau of Mines. The results of these tests are shown by the curves in Fig. 8. The data covering these cartridges is as follows:

	GRAMS
Cartridge No. 1.	
Weight of cardboard tube	15.6
Weight of Pittsburgh carbon black	19.2
Weight of condensed moisture	0.7
Cartridge No. 2.	
Weight one layer Pachuca paper	3.5
Weight of Pittsburgh carbon black	28.9
Weight of condensed moisture	0.2
Cartridge No. 3.	
Weight three layers Pachuca paper	10.5
Weight of Pachuca carbon black	31.2
Weight of condensed moisture	1.7
Cartridge No. 4.	
Weight two layers Pachuca paper	7.0
Weight of Pachuca carbon black	32.4
Weight of condensed moisture	1.4
Cartridge No. 5.	
Weight one layer Pachuca paper	3.5
Weight of Pachuca carbon black	39.8
Weight of condensed moisture	0.7

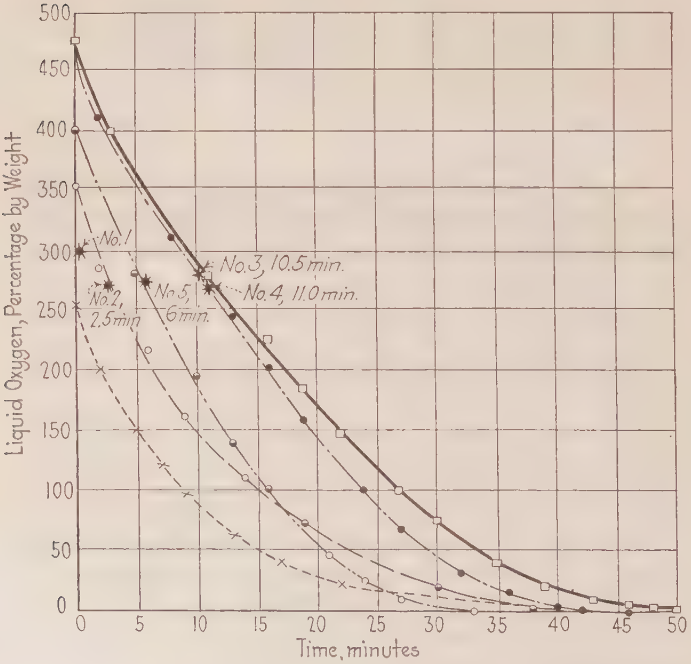

FIG. 8.—TESTS ON LIFE OF CARTRIDGES WITH VARIOUS WRAPPERS; AVAILABLE LIFE IS INDICATED BY STAR.

It is at once apparent that the cardboard tube used for No. 1 is not a satisfactory wrapper for the cartridge falls short of absorbing sufficient oxygen to give complete combustion. No. 2, a somewhat larger cartridge, with a single layer of Pachuca paper for a wrapper and Pittsburgh carbon black, had a life limit of 2.5 min., which is too low. Cartridge No. 3 was a regulation 28- by 300-mm. Pachuca cartridge to which two layers of paper were added. The paper wrapper was perforated before soaking. This cartridge had an available life of 10.5 min. Cartridge No. 4 was similar to No. 3 except that it had but one extra layer of paper; it had an available life of 11.0 min. Cartridge No. 5 was similar to No. 2. It is of interest to note that although the third layer of paper retarded the evaporation appreciably, it did not increase the available life of the cartridge on account of the additional oxygen required to give complete combustion of the third layer of paper. It is well to point out that the density and uniformity of the packing of the absorbent into the cartridges also has an important influence upon the available life. Apparently sufficient attention was not paid to this detail in making these comparative tests of wrappers. The present practice at Pachuca is to use but two layers of paper in the wrapper.

Results of Absorbent Tests

With us, the thorough investigation of absorbents has been subordinated to the main issue of determining the feasibility and economy of L.O.X. under the conditions obtaining in the mining operations of Compania de Real del Monte y Pachuca. We, accordingly, in the beginning, used the carbon-black cartridges supplied by the manufacturer of the equipment, thinking that these, everything considered, represented the highest state of the art resulting from the extensive experience with L.O.X. in Germany. If satisfactory results could be secured with the best German cartridges, we thought that there was a strong incentive for continuing the investigation, thereby developing the whole technique, as well as the cartridge, to best meet local conditions. On the other hand, if satisfactory results could not be secured with the best that the Germans had to offer, we were not justified in continuing the experiment.

The German carbon-black cartridge proved satisfactory in every respect save one, that of cost. On account of the great bulk of the carbon black in relation to its weight, transportation charges are excessive, which with a rather high initial cost, renders this absorbent expensive. The feasibility of L.O.X. having been demonstrated, we were at once forced to find a cheaper absorbent. The thorough investigation of this subject is a large task, in fact so large that the work which we have so far done may be regarded as but a beginning. The results of a large number of practical tests are briefly summarized below:

The sawdusts and pulps of various woods as well as the residue or bagasse from guayule (Mexican rubber shrub) showed a fair absorption and "available life" and were inexpensive, but the breaking effect in the mine was poor or the cartridge failed to detonate. Kerosene, gas oil, and fuel oil were added to secure greater explosive strength, but these reduced the available life to a point that was too low. Cotton, cotton waste, and kapok (a native tree cotton) showed high absorption, but the explosive strength was low, the cartridge difficult to fill, and the cost excessive. When oils were added, there was too great a decrease in the available life. Pulverized wood charcoals, bituminous coal, and coke gave very low available life. Admixture of kieselguhr with these increased the available life, but the strength was too low.

A Sprengluft cartridge, consisting of ground cork and a coal-tar product (presumably anthracene) has a satisfactory available life and good explosive strength. The wood of certain Mexican balsa trees is lighter than cork and shows high absorption of liquid oxygen but neither balsa sawdust nor balsa charcoal has shown sufficient explosive strength. It is, however, thought that the sawdust or pulp might form a satisfactory and inexpensive ingredient of a mixture. Possibly a mixture of balsa pulp with anthracene would give an absorbent approximating the Sprengluft cartridge containing ground cork.

Metallic powders, as well as inert substances such as iron oxide, as constituents of the absorbent, have not been investigated. The former would be expensive but would probably increase the explosive strength of a carbon-black absorbent. In fact there is a German cartridge upon the market in which a small amount of aluminum dust is mixed with the carbon-black absorbent. Inasmuch as iron oxide is not a combustible nor a good absorbent, its effect would appear to be solely that of a catalyst. The use of iron oxide with some forms of carbon, such as charcoal, for example, might, under certain conditions, prove dangerous.[18] Cartridges in which both aluminum dust and iron oxide are used have been developed by Weber.[19] These are said to be successfully used at the iron mines of MM. de Wendel, at Hayange, Lorraine.

There is a great variety of carbon blacks on the market for various purposes. Some of these are too expensive to merit investigation but representative samples of both natural-gas and tar-oil blacks, within a reasonable price range, are being investigated to determine whether any of them have special properties which would make them especially well suited as absorbents. Experiments are also under way with acetylene black and activated charcoal. Although some of the carbon blacks considered give little promise of becoming low-cost absorbents, it is thought that these tests will contribute in a useful way to the general knowledge of absorbents and their properties. A single absorbent, *i.e.*, carbon black, is used exclusively in all the regular mining operations at Pachuca employing L.O.X.

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[18] The cause of a number of explosions of metallic liquid-oxygen containers in Europe has been traced to the presence of iron oxide in the charcoal used in the vacuum chamber of the container. L. Wöhler: *Zeits. Komprimflüss Gase*, **20**, 109–112, 121–124, 133–137.

[19] Explosifs à base d'air liquid (Procédé Weber). Les Petits-Fils de Fois de Wendel & Cie. Hayange, Lorraine.

The carbon-black absorbent of the Sprengluft cartridge, used in most of the early work, was light, fluffy, and uniform. This black was manufactured in Germany and was probably produced from a tar oil. A carbon black manufactured in the United States by the German process from a tar oil containing some naphthalene proved less satisfactory and because of the high transportation charges, the cost delivered at the mine was excessive.

The absorbent now used is a carbon black made, by natives, from chapopote (Mexican fuel oil) by a crude process at a number of primitive

TABLE 6.—*Analyses of Certain Carbon Blacks*

SAMPLE	A	B	C
PROXIMATE ANALYSIS			
Moisture	4.2	3.6	3.2
Volatile matter	8.9	15.1	6.8
Fixed carbon	86.8	80.3	90.0
Ash	0.1	1.0	0.0
Total	100.0	100.0	100.0
ULTIMATE ANALYSIS			
Hydrogen	1.3	1.9	1.3
Carbon	88.8	81.8	91.6
Nitrogen	0.1	0.4	0.3
Oxygen	8.6	11.1	6.2
Sulfur	1.1	3.8	0.6
Ash	0.1	1.0	0.0
Total	100.0	100.0	100.0
MISCELLANEOUS DATA			
Calorific value, calories per kg	7,333.0	7,050.0	7,600.0
Calorific value, B.t.u. per lb	13,200.0	12,690.0	13,680.0
Specific gravity	1.73	1.30	1.32

Sample A is an American carbon black made by the German process from a tar oil containing naphthalene; it is less satisfactory than the German product.

Sample B is a Mexican carbon black made by crude native process from Mexican fuel oil; it is less satisfactory than German product but is used on account of lower cost.

Sample C is a German carbon black (contents of Sprengluft cartridge) probably made from a tar oil; it is more satisfactory than any absorbent so far used.

plants near Mexico City. The oil is allowed to drop slowly into pots, where it is burned with an incomplete supply of air. The carbon black is collected from the gases in connecting cement-lined masonry chambers. About 6 lb. of oil are burned to produce 1 lb. of carbon black. This material is fairly absorbent, but is inferior to the German carbon black as regards available life, uniformity, and strength. We feel confident that a better absorbent, which can be produced at a reasonable cost, will be developed to meet our conditions. In Table 6 are given some of the more important characteristics of the three carbon blacks that have been used on a large scale.

Preparation of Cartridges

The local practice is to make up the absorbent, or mixture of absorbents, into cartridges at a convenient central point on the surface. These may be either conveyed directly to the mine or placed in storage. Unlike the liquid oxygen, there is no reason why cartridges cannot be manufactured at a distance by companies who specialize in their production; this would doubtless insure greater uniformity, but there would always be the handicap of a high freight rate on a bulky material. It therefore appears that for isolated locations or for large operations, the greatest economy would result when the absorbents are produced and formed into cartridges locally.

The cartridge is simple, as it consists merely of a paper wrapper or container, within which the absorbent is placed. The selection of a suitable wrapper is important. It should have the following characteristics:

1. It should be inexpensive.
2. It should be readily penetrated by the liquid oxygen.
3. It should be sufficiently strong, both before and after soaking in liquid oxygen, to withstand any ordinary, and even rough, handling that the cartridges may receive.
4. It should be a good heat insulator, so as to increase the available life of the cartridge.

Although other wrappers, such as fabrics and especially thick cellular pasteboard (on account of its supposed superior insulating properties), have been used abroad and have been investigated to some extent by the U. S. Bureau of Mines, our experience indicates that medium-weight paper of the proper grade is the best wrapper, everything considered, we have so far found and is fairly satisfactory. We believe that present practice abroad is along similar lines. The Sprengluft cartridges had a special paper wrapper that contained numerous very small, in fact almost imperceptible, perforations such as would be made with the point of a needle. Paper chemists who have examined this wrapper state that it is a high-grade absorbent paper free from sizing or loading materials and that a fiber analysis indicates that it is 65 to 75 per cent. rag, 25 to 30 per cent.

sulfite pulp, and 5 per cent. wood flour. The mineral matter or ash amounts to 0.55 per cent. When we began to make our cartridges, fairly absorbent papers such as towel, mimeograph, etc., were used but after soaking, the cartridges were easily torn and broken during handling and loading. Other paper wrappers were also discarded on account of poor oxygen absorption, too rapid evaporation of oxygen, excessive thickness which reduced the space for absorbent, or weakness when saturated with liquid oxygen.

The wrapper now used is a Kraft wrapping paper weighing 45 lb. per ream of 500 sheets, 28 by 40 in. (700 by 1000 mm., weighing 60 gm. per sq. m.). This paper is made from partly refined pine fiber in which there remains part of the resin. This, together with the sizing of resin soap the paper is given in finishing, makes it somewhat resistant to water, a desirable quality for wrapping paper. This cannot, in any sense, be regarded as an absorbent paper; but it absorbs liquid oxygen fairly well, has good insulating properties, is strong, easily handled, and inexpensive. This subject is being investigated further for there is opportunity for improvement. One ream will make 6000 wrappers 7 by 13.3 in. (178 by 338 mm.) for 1.1 by 12 in. cartridges, or 4500 wrappers 9 by 13.3 in. (228 by 338 mm.) for the 1.38 by 12 in. cartridges. The wrappers are formed by rolling the small sheets of paper twice around a wooden cylinder of proper size. The paper is held in position by sticking the outer edge with starch paste. It costs about $6 U. S. Cy. per ream delivered at Pachuca and a boy working 8 hr. at $0.75 U. S. Cy. makes 1000 paper cylinders. At first the cartridges made with the Kraft paper were perforated, after being filled with absorbent, by rolling them over a board the surface of which was covered with fine, metal points. Perforating hastened the soaking but it also increased the rate of evaporation and so was abandoned.

Recent experiments indicate that the available life of the cartridge can be increased about 40 per cent. over the double wrapper cartridge previously used by forming the cartridge with a single wrapper and then after removing from the liquid oxygen slipping over it a single dry wrapper. This requires no more paper and in effect is but a modification of the double wrapper.

Filling of Cartridges

The arrangement used for filling the blank paper tubes with absorbent, shown in Fig. 9, consists of a vertically split wooden form or mold with recesses for ten wrappers. At a higher level, and to one side, is a bin for the absorbent and below is a bin to receive the cartridges. Above the mold is a tamper having ten fingers that register with each recess. Wrappers are placed in each recess and the gate to the bin opened so as to allow the absorbent to fall into the enlarged space at the top of the mold. The tamper is moved up and down by hand until the absorbent is worked into the wrappers and packed to the desired density. The cartridges are discharged by opening the bottom gate and swinging the two halves of the mold on the pivots so as to separate them and thus permit the cartridge to drop out. A small amount of the absorbent is squeezed out of each end of the cartridge and the projecting ends of the wrapper are folded over. The boys who do this work soon become proficient in producing uniformly filled cartridges of the correct density.

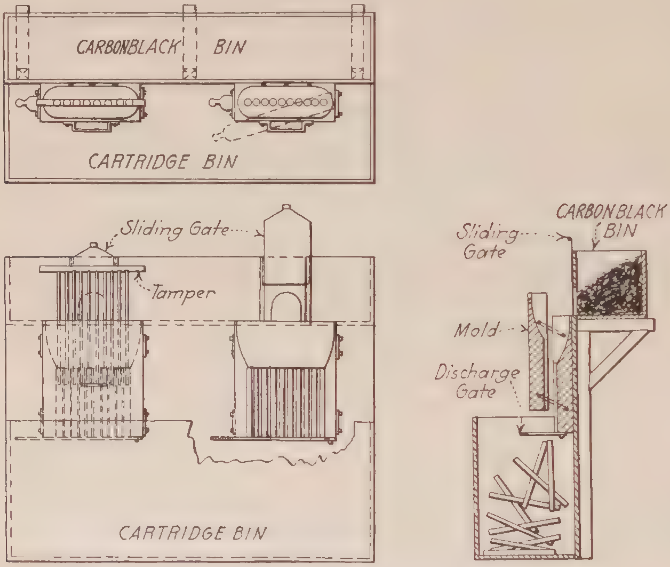

Fig. 9.—Apparatus for forming cartridges.

The proper density of the absorbent in the cartridge is of great importance. If filled too tightly, there will be an excess of absorbent and the oxygen absorption will be too low; if filled too loosely, there will be a deficiency of absorbent and the cartridge will not be sufficiently powerful. One boy can fill 1500 cartridges in an 8-hr. shift. Automatic machinery could be readily developed or adapted from other industries for doing this work; and where a large number of cartridges were used or labor was high, machine filling doubtless would prove advantageous. The simple hand method described, which was developed to meet Pachuca conditions, has worked out well and, as will be noted, is capable of producing cartridges at a low cost.

Present Cost of Cartridges

Using as an absorbent Mexican carbon black, purchased from small producers, the cost of cartridges filled by hand is as follows:

Size of cartridge	1.38 by 12 in.	1.1 by 12 in.
Weight of cartridge	0.14 lb. (63 gm.)	0.096 lb. (43.6 gm.)
Carbon black at 10 c. per lb. (22 c. per kg.)	58 gm., 1.28 c.	40 gm., 0.88 c.
Wrapper, paper	0.14 c.	0.11 c.
Wrapper, labor and starch paste	0.10 c.	0.10 c.
Filling, folding, and supervision	0.21 c.	0.21 c.
Miscellaneous supplies	0.12 c.	0.10 c.
Total present cost per cartridge U. S. Cy	1.85 c.	1.40 c.

Estimated Cost of Cartridges

With local manufacture of carbon black from Mexican fuel oil by simple native method and machine filling of cartridges, for an operation using 100 l. per hr. of liquid oxygen, the estimated cost of cartridges, in U. S. currency, is

1.38- by 12-in. cartridge	1.25 c.
1.10- by 12-in. cartridge	0.95 c.

Methods of Exploding L.O.X.

Authorities state that when L.O.X. in a confined space is ignited with a fuse alone, an explosion results which resembles that produced by black powder under the same conditions. When exploded by detonation, the effect is similar to that with dynamite. Methods of firing are described in a pamphlet issued by the Oxyliquit-Sprengluft G.m.b.H.[20] In metal mines, a powerful explosive, with the characteristics of dynamite, is required. Our experience with firing by fuse alone is limited but of the holes so tried about 75 per cent. broke well; the absorbent in the others merely burned without explosive effect. The U. S. Bureau of Mines[21] reports that an ordinary electric detonator, such as is used for dynamite, inserted in the cartridge prior to soaking has proved satisfactory in its experimental work. Abroad, cap and fuse, electric detonators, and even liquid-oxygen detonators especially developed for the purpose, have been used. At Pachuca, a number of methods of detonation, employing both fuse and electric firing, have been tried. Fuse and caps of both European and American manufacture have been used.

Electric delay-action detonators supplied by Sprengluft were used in the early work; Fig. 10 shows the detonator in detail. The fuse is cut to different lengths so as to time the shots properly. These had the advantage that the round could be exploded shortly after the men had retreated to a safe place, thereby reducing the loss of oxygen to a minimum. On the whole, they proved satisfactory but the making of the electrical connections was too complicated and slow for the native miners, who had little or no previous experience with electric blasting; also the cost was higher than with the simple cap-and-fuse method later used, to which the miners have been accustomed. The miners could have been trained to use electric detonation but this did not appear to be warranted until L.O.X. was placed on a firmer basis in other respects.

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[20] Aus der Praxis des Sprengluft verfahrens. Berlin, 1922.

[21] J. W. Paul: Personal communication.

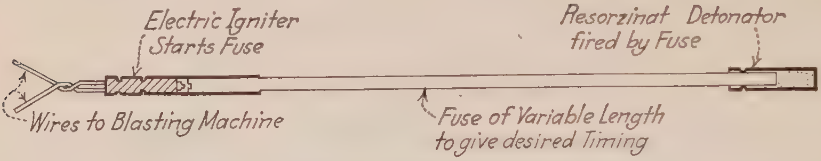
Electric Igniter Starts Fuse
Resorzinat Detonator fired by Fuse
Wires to Blasting Machine
Fuse of Variable Length to give desired Timing

FIG. 10.—ELECTRIC DELAY-ACTION DETONATOR.

At present, firing is accomplished with safety fuse and ordinary 8X blasting caps manufactured in the United States. The caps have proved quite satisfactory, very few misfires having occurred. The fuse is not entirely satisfactory on account of occasional side spitting and burning through of the covering. This defect may result in the burning of the cartridges before the cap detonates. This difficulty has, however, been overcome by smearing the fuse with moist clay. Numerous burning tests have been made on several brands of American fuse, and none of these has been found to be free from side spitting.

Although no timing tests have been made, the burning rate of fuse in holes loaded with L.O.X. appears to be slower than with dynamite. It has been found that a fuse having a tar or asphalt covering becomes brittle and easily broken after contact with liquid oxygen; therefore it is not suitable for use with L.O.X. The Sprengluft fuse has been found highly satisfactory as it does not side spit in the hole and the covering is relatively non-inflammable; it has the same outward appearance after burning as before.

S. P. Howell, of the Pittsburgh station of the U. S. Bureau of Mines, has examined the Sprengluft fuse and reports as follows:

> The fuse is very stiff and crackles when it is unwound. It is difficult to straighten out, as it will not uncoil. The average diameter is 0.180 in., and the greatest variation is 0.002 in. The weight per yard of fuse is 76.79 gr. The weight of powder per yard of fuse was obtained by unwinding 43 in. of fuse and weighing the powder. The powder in this length weighed 5.945 gr., or practically 5 gr. per yd. The powder used in the fuse is the ordinary fuse powder in use in most places.
>
> *Rate of Burning.*—29¾ in. burned in 1 min. 9⅕ sec.; therefore, 1 ft. burns in 28¼ sec. In burning this distance, fire spit out of the fuse three times, and smoke came through the covering, always just back of the fire, having no difficulty in reaching the outer air.
>
> *Outside Covering.*—The outside covering consists of eight medium-sized cotton threads, wound clockwise, with a space of 0.05 in. center to center, along the length of

the fuse. The threads cover less than one-half of the surface. This is covered with a water-soluble glue, which makes this fuse sticky to the touch. The cotton threads are strong and consist of three strands each.

Middle Covering.—This consists of six thin cotton threads 0.059 in. center to center, measuring in the length of the fuse. These threads are wound to the left with considerable space between adjacent points. This covering is also treated with the glue.

Inner Covering.—This consists of ten coarse threads, wound tight against each other and turned to the right. Seven of these threads are of hemp of natural color, and consist of one strand each. The other three are woolen, gray in color.

The Sprengluft resorzinat detonator, Fig. 11, having an aluminum shell which, it is claimed, suffers no loss in efficiency through immersion in liquid oxygen, was likewise satisfactory. Both the fuse and caps were more costly than the ordinary safety fuse and copper caps now used.

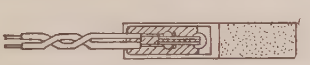

FIG. 11.—RESORZINAT DETONATOR.

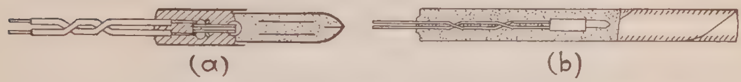
(a) (b)

FIG. 12.—(*a*) LIQUID-OXYGEN DETONATOR; (*b*) INSERTED IN CARTRIDGE.

The Sprengluft liquid-oxygen detonator, Fig. 12, although early discarded on account of giving poorer results, probably merits further investigation. It consists of a slotted steel capsule in two parts filled with a special absorbent. Into the open end is inserted the fuse or electric igniter. It is necessary to embed the detonator in one of the cartridges before soaking, as the detonator does not become effective until its absorbent is saturated with liquid oxygen. In fact the manufacturers recommend that the slots in the cap be freed from dust or other obstruction then dipped in liquid oxygen for about ½ min. before inserting in the cartridge. The oxygen finds its way to the absorbent through the slots in the capsule and, of course, the detonator becomes harmless after the oxygen in the cartridge has evaporated. With this detonator, even the hazard of an unexploded cap in a missed hole is absent.

In the early practice, when Sprengluft fuse and resorzinat detonators were used, the cap with fuse attached was embedded in the absorbent of the primer cartridge prior to soaking. This practice was possible as the Sprengluft fuse does not become unduly brittle after soaking in liquid oxygen, but the American fuse so far used becomes so brittle after soaking in oxygen that this practice is inadvisable. The present procedure is to insert the fuse and cap into all the holes before beginning to load the cartridges. This practice saves oxygen, as the time between removing the cartridges from the dipping container and firing the round is reduced thereby. In loading upper holes, the fuse at the cap end is bent back through an angle of 180°; despite this rather rough treatment there has been surprisingly little trouble traceable to this cause.

Perhaps the most serious disadvantage of firing with fuse is the greater loss of liquid oxygen through evaporation during the time necessary for the fuse to burn. With a burning rate of 40 sec. per ft. and a round requiring fuse 7 ft. (2.1 m.) in length, nearly 5 min. elapse from the time the fuse is lighted until the explosion takes place. On the whole there seems to be but little doubt that electric detonation is superior to cap and fuse by reason of the lower oxygen loss, greater safety, and the ability to load and fire more shots at a round.

Loading of L.O.X.

The necessary paraphernalia, consisting of the blank cartridges, caps, fuse, carrying containers holding the liquid oxygen, and the dipping container or containers are brought to the place where the blasting is to be done. After placing the necessary number of cartridges in the dipping container, a small amount of liquid oxygen is poured into it from one of the transport containers. The vapors of the rapidly evaporating oxygen rise from the bottom, completely enveloping the cartridges and the inner walls of the container, thereby effectively precooling the relatively hot cartridges and container. After about 2 min. the container is filled with liquid oxygen to within about 1 in. of the top. The soaking of the cartridges continues until the cartridges are thoroughly saturated. This requires about 6 min. While this is going on, the drill holes are cleaned by blowing out with air, and a fuse with cap attached is placed in each as shown in Fig. 13.

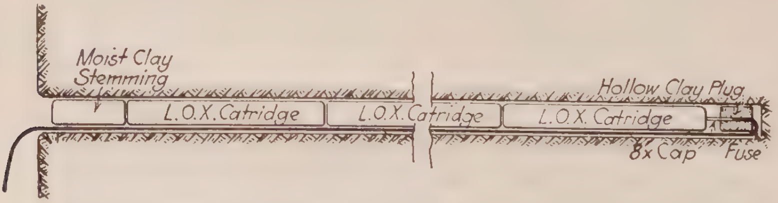

Fig. 13.—Method of inserting detonating cap in drill hole.

Following these preliminary preparations the dipping container is brought within reach of the face. If less than twelve holes are to be shot at one time, the loading crew consists of two men; one man removes the cartridges, one at a time, and passes them to the second man, who pushes them into the hole with a wooden tamping stick. When the round is more than twelve holes, the crew is increased to three men, one man passes the cartridges and two men load them into the holes. As a matter of fact, the passer does not save much time and, with higher-grade labor, might be dispensed with. The cartridges saturated with liquid oxygen are handled with the bare hands; and while there was some trouble at first through burns, caused by too intimate contact of the extremely cold cartridges

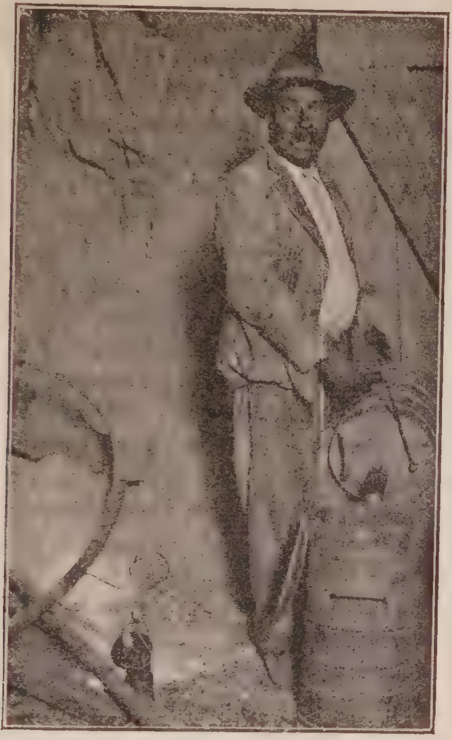

FIG. 14.—POURING LIQUID OXYGEN INTO DIPPING CONTAINER.

FIG. 15.—LOADING L.O.X. IN CROSSCUT.

with the skin, the men soon learned that by rolling the cartridge in the hands, so that it did not remain long in contact with the skin at any point, burns could be avoided. The oxygen vapor given off during loading cools the surrounding air, condensing water therefrom and forming a heavy mist, which settles down and tends to obscure the bottom of the working face. This has led to the practice of loading the bottom holes first.

The L.O.X. cartridges do not have the plasticity of dynamite, therefore considerable difficulty was at first experienced in loading them into the upper holes on account of their frequent failure to stick. The brittle cartridges, when they fell to the rocks below, broke, resulting not only in a loss of time but a loss of absorbent and oxygen. This difficulty was overcome by tying to the cartridge a pointed spur of thin sheet iron or tin plate, ¼ in. wide by 2½ in. long (6 by 63 mm.). The cartridge is inserted into the hole so that the projecting point of the spur is downward. In this position it permits pushing the cartridge into the hole with the tamping stick, but effectively prevents the cartridge from falling back. This simple expedient has proved effective and inexpensive.

Unlike other explosives, it is necessary to make provision for venting the stemming used with L.O.X. so that the gaseous oxygen, which is constantly being given off from the cartridge, may escape; otherwise, sufficient pressure will develop to blow the stemming and, at times, the cartridges out of the hole. The early practice was to tamp stiff clay stemming around a ⅛-in. brass pipe of sufficient length to extend from the top of the charge to outside the collar of the hole. After all the stemming was in position, the tube was withdrawn, leaving a vent from the cartridges to the outside air through which the gaseous oxygen could escape. This is the method that has been used most in European practice. It was found that the vent tube could be dispensed with if a more porous stemming were used; this reduced the time occupied in loading, which decreased the amount of oxygen given off by the cartridges in the interval between placing in the hole and firing them. Clay plugs or wads of moist earth so placed as to give the necessary porosity are now used for the stemming.

Number of Holes Fired at a Round

During the early investigation of L.O.X., and before there was an opportunity of making working tests at Pachuca, there was much uncertainty as to the number of shots that could be successfully fired in a single round; indeed, it appeared that this was one of the most important of the unknown factors to be determined.

The local practice with dynamite, in drifts and crosscuts has been to load and fire the cut holes first; the same practice is followed with L.O.X. Two men load and fire eight 5.5-ft. (1.68-m.) holes requiring thirty-two 1.38 by 12 in. L.O.X. cartridges in about 7 min. After waiting about 5 min. for the face to clear, the two men are able to load and fire the remaining eight or ten holes in about 8 min.; when there are more holes in the last section of the round, say twelve to sixteen, a third man is required if the loading and firing is to be completed within 10 min. On several occasions twenty-two 5-ft. (1.52-m.) holes requiring 80 cartridges have been loaded and fired at one time; employing two passers and two loaders and electric firing, the elapsed time was 9½ minutes.

The largest stope round that has been loaded and fired at one time consisted of twenty-eight holes about 4.5 ft. (1.37 m.) deep, in which seventy-five, 1.1- by 12-in. cartridges were used. With two men passing and two men loading, 8½ min. elapsed from the time that the loading began until the charges were exploded with an electric blasting machine. When safety fuse and caps are used, about 10 min. is required for the same number of holes. The usual large stope round of about twenty holes, fired with safety fuse requires on an average about 8 minutes.

It has been found that the number of holes that can be loaded and fired at one time is governed by a number of factors. The available life of the cartridges used fixes the time available for loading and firing. The room available at the working face, the convenience of the drill holes (as well as their depth), the method of firing, and the experience and skill of the loading crew have a great influence upon the number of holes that can be loaded and fired in a given time. With favorable conditions for loading, electric firing and cartridges with somewhat better than a 15-min. available life it does not seem unreasonable to expect that rounds of forty holes could satisfactorily be handled.

Use of L.O.X. in Different Operations

It has been found practical in short timbered raises, up to 100 ft. (30.5 m.) to load and fire twelve holes at one time. Little timber is used in most of the raises and this, together with the lack of room, renders the handling of the carrying and dipping containers inconvenient.

Several old stopes that are being reworked by slabbing-off and robbing of the pillars are accessible only by narrow crooked ladderways. The use of L.O.X. under these conditions has not been seriously considered. It is possible that further experience with L.O.X. will result in the development of special equipment or cartridges having a longer available life to meet such situations.

The use of L.O.X. in shaft sinking has not been attempted. The limited space, the frequent presence of considerable water, together with the present limited time available for loading and retreating, appear to offer obstacles for which there is at present no entirely satisfactory solution.

The consensus of opinion of those who have had experience with L.O.X. in Europe is that this explosive cannot be satisfactorily used in wet holes. Experience at Pachuca confirms this opinion. If there is merely water standing in the hole, this can be removed or the rapid insertion of the cartridges will freeze the water into an encircling layer of ice and the loading can be completed despite the clouds of vapor. But the few attempts made to load holes having water flowing from them, have failed on account of rapid evaporation of the oxygen with the formation of dense clouds of vapor as well as the blocking of the hole with ice. Various methods of overcoming this difficulty have been suggested; among others it has appeared not impossible that some kind of protecting envelope could be developed.

Attitude of Miners and Training Period Required

When this investigation was initiated, it was feared that the miners would be prejudiced against the use of a new explosive and, at best, that a rather long training period would be required before they could use it effectively. Fortunately, these difficulties, which would have seriously interfered with a prompt appraisal of the value of L.O.X., were only imaginary; the miners accepted the innovation enthusiastically and their training proved to be a simple matter.

Our experience may be of interest to others who may contemplate introducing L.O.X. The miners who are not concerned with the cost of the explosive used, invariably prefer L.O.X; The contractors, when it can be had at the same cost as dynamite, likewise prefer to use it. One reason for this preference, notwithstanding the extra trouble of handling the necessary paraphernalia, the necessity for haste in loading and firing, the limitation on the size of the round, and the inability to load holes in advance to be shot later at a predetermined time, is that the comparative freedom from noxious gases permits them to return to the face promptly, in most cases within 5 min. after blasting; this, they feel, is a decided advantage. Undoubtedly, the greater safety of L.O.X. has a considerable influence upon their preference. When a misfire occurs, they can remove the dead cartridge promptly and reload the hole; the danger of drilling into missed holes is practically eliminated. There can be no dangerous explosive in the waste pile. Another important point is that, in general, there is more of a shattering effect which tends to break the rock smaller, thereby reducing the work of bulldozing. The prompt acceptance of the new explosive is all the more noteworthy in a locality where the theft of dynamite reaches considerable proportions; so far as we know, there has been no stealing of L.O.X.

The time necessary to introduce L.O.X. into a working mine employing native miners and under the conditions obtaining at Pachuca is illustrated by the experience at the Rosario mine. This mine is worked through a tunnel and two interior shafts. The monthly development advance amounts to about 400 ft. (122 m.) and stoping is done on two levels. The average production per day of two shifts is 475 metric tons. The new explosive was first used on Jan. 9, 1922, and by the end of the month approximately 90 per cent. of all the blasting in the mine for both development and stoping was being done with L.O.X. It was found that the native miners, after observing the loading of two or three rounds, were able to use this explosive quite effectively.

STRENGTH AND BREAKING EFFECT OF L.O.X. AS COMPARED TO DYNAMITE

The strength of L.O.X. is dependent on a number of factors: purity of the oxygen, nature of the absorbent, time at which cartridge is fired, all exert an important influence. Failure to define clearly the precise conditions of comparison have in the past caused confusion. In some cases, it has been uncertain whether the basis of comparison was equal weights or equal volumes of L.O.X. and dynamite. Sieder[22] gives data which indicate that with favorable absorbents L.O.X. is more powerful than certain dynamites. Rice[23] states that tests made at Pittsburgh in the ballistic pendulum by the standard Bureau of Mines test indicate that L.O.X. is 4 to 12 per cent. stronger than 40 per cent. straight nitroglycerin dynamite. The cartridges used in these tests contained a mixture of absorbents now known to be less powerful than some of the absorbents that have been used in practice.

Attention has been called to the fact that L.O.X. can be less effectively tamped and stemmed than dynamite. This, together with the difficulty of firing at exactly the ideal moment for detonation of the cartridge, accounts for an actual decrease in strength, under working conditions, below that shown by laboratory tests in the Bichel gage and in the ballistic pendulum. The strength or performance of L.O.X., compared to the 40 per cent. dynamites commonly used in mining operations, as determined by use of a considerable amount of both explosives in mining operations is therefore more convincing than laboratory tests, in which no allowance can be made for these factors. The basis of the comparison here given is the previous known dynamite consumption per ton of ore stoped and per meter of development in the Rosario and Camelia mines in which L.O.X. is now largely used. The first month's work with L.O.X. is excluded for obvious reasons.

The full monthly production of the oxygen plant is equivalent to about 17,000 lb. (7700 kg.) of 40 per cent. gelatin dynamite. The use of this quantity of explosive permits of a reliable appraisal of the actual strength or breaking effect of L.O.X. under working conditions as compared to the gelatin dynamite heretofore used. It has been found that 1 lb. of 40 per cent. gelatin dynamite is equivalent to 0.87 lb. of L.O.X. consisting of 0.25 lb. carbon black absorbent and 0.62 lb. of liquid oxygen. This is the weight of liquid oxygen that is theoretically present to give complete combustion at the ideal moment for detonation of the cartridge. As the firing of the cartridges is timed to take place as near this point as possible, this weight approximates the actual weight of liquid oxygen present at the moment of explosion. However, there is a heavy loss of liquid oxygen through evaporation between the time that the oxygen is produced and the cartridges are exploded and as a consequence the actual weight of oxygen required is much greater. Most of this loss occurs in the dipping container and during the time that the cartridges are being loaded. Actually, it has been found necessary to produce 2.10 lb. of liquid oxygen against a theoretical requirement of 0.62 lb. to equal 1 lb. of 40 per cent. gelatin dynamite. As most of the loss through evaporation takes place at the working face, the oxygen is not altogether wasted, for it has some influence on the ventilation. The 0.25 lb. of carbon black absorbent is equivalent to the contents of 1.8 cartridges 1.38 by 12 in. or 2.6 cartridges 1.1 by 12 in. One carbon-black L.O.X. cartridge 1.38 by 12 in. (35 by 300 mm.) is the equivalent of one cartridge of 40 per cent. gelatin dynamite 1.25 by 8 in. (32 by 200 mm.) and one cartridge L.O.X. 1.1 by 12 in. (28 by 300 mm.) is the equivalent of one dynamite cartridge 1.00 by 8 in. (25 by 200 mm.).

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[22] Ludwig Sieder: "Oxyliquit." *Zeit. des Schiess-Sprengstoffwesen.*

[23] George S. Rice: "Development of Liquid Oxygen Explosives during the War." U. S. Bureau of Mines *Tech. Paper* 243, 14.

The approximate oxygen distribution per pound of carbon-black cartridge is as follows:

	Pounds
Theoretical amount necessary for complete combustion of absorbent and wrapper at time of explosion	2.5
Estimated evaporation loss in drill hole	1.5
Estimated evaporation loss in filling and emptying soaking containers, by spillage and during soaking	3.7
Estimated loss in carrying containers, taking as a basis 12 hr. in a 5 l.-container	0.7
Total oxygen necessary to produce for each pound of carbon-black absorbent	8.4

Ratio of liquid oxygen produced to oxygen usefully utilized as an explosive is therefore 3.36 to 1.0.

Diederichs[24] estimated that 72,000 l. of liquid oxygen would be required to replace 40,000 kg. of dynamite, in a German coal mine producing 400,000 tons of coal per annum, 1 kg. of dynamite being equivalent to 1.8 l. of liquid oxygen, or 1 lb. of dynamite equivalent to 2.0 lb. of liquid oxygen. This compares quite closely with the Pachuca figure of 1 lb. of dynamite equivalent to 2.10 lb. of liquid oxygen. It is now claimed that better results are being secured in Germany, *i.e.*, 1½ l. of liquid oxygen is the equivalent of 1 kg. of 40 per cent. dynamite, or 1.65 lb. of liquid oxygen is the equivalent of 1 lb. of 40 per cent. dynamite.

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[24] H. Diederichs: "Die Erzeugung und Verwendung flüssiger Luft zu Sprengzwecken" *Stahl und Eisen* (1915) **35,** 1180.

There is an opportunity for reducing the loss of oxygen occurring during soaking of the cartridges and this, together with other refinements, leads us to think that the best German figure is not impossible of attainment.

Present Cost of L.O.X.

The cost given herewith is computed and stated on the basis of the comparative cost of 40 per cent. gelatin dynamite. These unit costs have been derived from the present operations and are given in detail earlier in the paper. Liquid oxygen 2.79 c. per lb., carbon-black cartridges 1.38 by 12 in. 1.85 c. each, and 1.1 by 12 in. 1.40 c. each. The equivalent of 1 lb. 40 per cent. gelatin dynamite requires 2.1 lb. of liquid oxygen and 1.8 cartridges 1.38 by 12 in. and 2.6 cartridges 1.10 by 12 in. The total cost then of L.O.X. equivalent in breaking effect to 1.0 lb. of 40 per cent. gelatin dynamite would be:

	Cents U. S. Cy.
2.1 lb. liquid oxygen at 2.79 c. per lb.	5.86
1.8 cartridges at 1.85 c	3.33
Total cost, if 1.38 by 12 in. cartridges are used	9.19
2.1 lb. liquid oxygen at 2.79 c. per lb.	5.86
2.6 cartridges at 1.40 c.	3.64
Total cost, if 1.10 by 12 in. cartridges are used	9.50

Average total direct or operating cost of, say, 9.35 c. per lb. No allowance is made in this for amortization or interest on the capital expenditure for the liquid-oxygen plant and equipment. In other words, 40 per cent. gelatin dynamite would have to be delivered at the mine for 9.35 c. U. S. Cy. per lb. (20.6 c. U. S. Cy. per kg.) to equal the direct costs now being secured with L.O.X. in an experimental unit.

SUMMARY OF MAJOR ITEMS OF PRESENT COST OF L.O.X.

	U. S. Cy.	Per Cent.
Power, 306 kw.-hr. at 0.63 c.	$1.93	20.6
Labor, in oxygen plant	2.05	21.9
Caustic potash, 4.4 lb. at $0.11	0.48	5.1
Lubricants, 0.52 gal at $1.00	0.52	5.6
Carbon-black absorbent, 23.0 lb. at $0.10	2.30	24.6
Paper for cartridges, 2.0 lb. at $0.13	0.26	2.8
Total of major items	7.54	80.6
Miscellaneous labor, supplies, and repairs	1.81	19.4
Total L.O.X. equivalent to 100 lb. of 40 per cent. gelatin dynamite	9.35	100.0

ESTIMATED COST OF L.O.X. ON LARGER SCALE OPERATION

The present unit was installed primarily for large-scale experimentation and demonstration and without the expectation of securing as low costs as would be possible following this preliminary experience and with a larger oxygen unit. It is therefore of interest to include an estimate of costs with an oxygen unit having a capacity of 100 l. per hr., as this is more nearly in line with the scale of the operation if L.O.X. were generally used at all the mines of Cia de Real del Monte y Pachuca; 100 l. per hr. of liquid oxygen would be equivalent to 70,000 lb. (31,750 kg.) of 40 per cent. gelatin dynamite per month. The principal economies of the larger scale operation would result from a lower cost for liquid oxygen, a lower cost for carbon black through local manufacture from Mexican fuel oil, and a lower cost of manufacturing the cartridges through substitution of machinery for hand labor. The other items, such as equivalent oxygen and cartridges per pound of 40 per cent. gelatin dynamite would remain the same. The estimated costs of the oxygen and cartridges, which have been given in detail, are briefly summarized as follows:

Direct cost of liquid oxygen	1.86 c. per lb.
Direct cost of carbon black	6.00 c. per lb.
Direct cost of 1.38- by 12-in. cartridge	1.25 c. each
Direct cost of 1.10- by 12-in. cartridge	0.95 c.
2.1 lb. liquid oxygen at 1.86 c	3.91
1.8 cartridges at 1.25 c	2.25
Total cost if 1.38- by 12-in. cartridges are used	6.16 c.
2.1 lb. liquid oxygen at 1.86 c	3.91
2.6 cartridges at 0.95 c	2.47
Total cost if 1.10- by 12-in. cartridges are used	6.38 c.

Average total direct operating cost of say 6.25 c. per pound.

The direct cost of L.O.X. is therefore equivalent to 40 per cent. gelatin dynamite at 6.25 c. per lb. (13.8 c. per kg.) at the mine. Amortization and interest charges on the capital expenditure for the liquid-oxygen plant and equipment are not included in this estimate for the reason that we have no knowledge of what the cost of the plant would be under present conditions. Equipment for this purpose is now manufactured mostly in Germany. On account of the violent fluctuations in exchange and the ever-changing policy of venders of machinery and supplies, all purchases in that country are the subject of special negotiation and the cost in U. S. currency therefore cannot be determined until the sale has been consummated. American manufacturers have begun the manufacture of equipment for L.O.X. and it is expected that soon this will be placed upon the same basis as air compressors, rock drills, etc.

General charges are excluded from the estimate for the reason that they would vary at different properties, depending on the policy of the management and the methods of accounting used. However, this item can be added by any one interested to fit particular conditions. There would also be an item for supervision of the oxygen plant and blasting practice during the period necessary to train a local crew. Should the use of L.O.X. become at all general, competent operators would soon be developed, as is the case with the older operations of mining.

Transportation of Liquid Oxygen

The expense of transporting the containers from the oxygen plant to the working face in the mines depends, a great deal, on local conditions and to what extent this can be combined with other operations. A motor truck makes one round trip per day (24 hr.) between the oxygen plant and the Camelia mine, 1.5 mi. distant, delivering the full containers at the collar of the shaft and returning the empty containers to the plant. The full and empty containers are transported to and from the Rosario mine, which is 0.9 mi. from the oxygen plant, by the ore train. At both mines, the containers are lowered to the working levels through the shafts and carried to the working faces by the drilling crews or by the returning ore cars. The additional cost for distribution of liquid oxygen over dynamite has therefore proved to be a small item at Pachuca; in fact, so small that we have not taken it into account in the costs given. Elsewhere, conditions might be different and allowance would have to be made for this item. However, it would appear that the benefit to ventilation and the lessened costs for accident benefits would, in most cases, more than offset the additional cost of distribution.

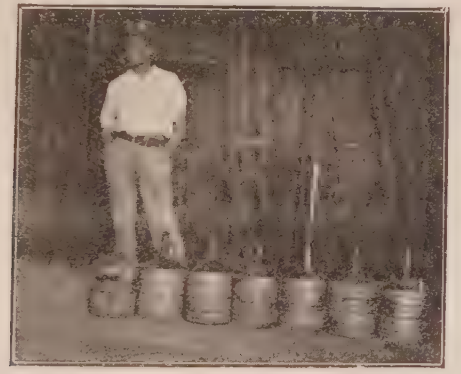


Fig. 16.—Liquid oxygen containers at Rosario shaft.

Effect upon Ventilation

The formation of an undue amount of carbon monoxide gas has been frequently urged as a serious disadvantage of L.O.X. This was doubtless true of some of the earlier operations in which enriched liquid air instead of liquid oxygen was used and before there was a clear understanding of the technique of the operation. Pachuca experience indicates that there is no difficulty in avoiding the formation of noxious gases, when L.O.X. is intelligently handled; indeed, it has been found that there are less noxious gases than with dynamite. Invariably the miners are able to return to the working face sooner than with the explosives previously used. There may be even a small enrichment of the air in oxygen at the working face where most needed, which would be a distinct benefit to ventilation. E. D. Gardner, a mining engineer of the U. S. Bureau of Mines, took a number of samples of mine air at various working faces after blasting with both dynamite and L.O.X. These samples were brought, in the special containers that the Bureau has developed for mine gases, to the Pittsburgh station where they were analyzed by Bureau chemists having special experience in the examination of mine gases. The results of these analyses are given in Table 7.

TABLE 7.—*Mine Air Analyses, Percentages*

DESCRIPTION	O_2	N_2	CO_2	CO	CH_4	H_2	NO_2
L.O.X.							
Sample taken in 410 crosscut 3½ min. after firing 54 cartridges	23.45	74.43	2.03	0.09	0.0	0.0	0.0
Sample taken in Fortuna level, only one hole out of 22 exploded on account of weak battery	20.91	79.05	0.04	0.0	0.0	0.0	0.0
Sample taken in San Francisco level immediately after firing 12 cartridges	20.93	79.03	0.04	0.0	0.0	0.0	0.0
Sample taken in dead end of 410 crosscut 4 min. after firing 12 cartridges	20.93	79.04	0.03	0.0	0.0	0.0	0.0
Sample taken in 880 crosscut 1 min. after firing 85 cartridges (21 holes)	21.12	78.52	0.36	0.0	0.0	0.0	0.0
Sample taken in Santa Sophia level 2 min. after firing 12 cartridges (5 holes)	21.10	78.54	0.36	0.0	0.0	0.0	0.0
Gelatin Dynamite							
Duplicate samples taken in E face 480 level 8 min. after firing 110 sticks of gelatin dynamite (manufacturer A)	20.81	79.07	0.12	0.0	0.0	0.0	0.0
	20.79	79.09	0.12	0.0	0.0	0.0	0.0
Sample taken in 440 H. crosscut 6½ min. after firing 60 sticks of gelatin dynamite (manufacturer B)	19.68	77.49	2.15	0.41	0.15	0.12	0.004

On account of not being able to approach the face as soon, a longer time elapsed before the samples were secured after the explosions with the dynamite than in the case of L.O.X. This would tend, if anything, to make the results more favorable to dynamite for there would be a longer period for the gases to diffuse. It may be added that these samples were taken when the technique of using L.O.X. was not as well understood as at present. It will be apparent that difficulty through the formation of noxious gases is not to be feared.

OTHER USES FOR LIQUID OXYGEN

An important economy has resulted through the use of the liquid oxygen for oxy-acetylene welding and cutting instead of purchasing compressed oxygen in cylinders for this purpose. The method of using it is very simple; sufficient liquid oxygen is poured into an air receiver to give, upon evaporation, a pressure of about 100 lb. per sq. in. (7 kg. per sq. cm.). The gas is piped from the receiver to wherever it is to be used about the machine shop. The oxygen is not as pure as is required for some classes of cutting, but there has been no difficulty in doing a wide range of work. Liquid oxygen can also be used in breathing apparatus especially designed for the purpose, but so far this use has not been investigated.

ADVANTAGES AND DISADVANTAGES OF L.O.X.

The advantages of L.O.X., which are somewhat the same as those given by various authorities, but with added weight on account of demonstration at Pachuca, are briefly as follows:

1. A substantial saving in cost over the present cost of dynamite.
2. Greater safety, although it is difficult to place a definite monetary value upon this advantage.
3. Freedom from noxious gases, when properly used, renders it possible to return to the working faces sooner than when dynamite is used.
4. Small increase in the oxygen content of the air at the working face is an advantage.
5. Although the powder magazine cannot be entirely dispensed with, there is no hazard from the storage of large quantities of high explosives.
6. Entirely eliminates thawing of powder; this is of no consequence at Pachuca but in a cold climate it would be an important advantage.
7. No high explosive in the waste pile or mill bins, for the L.O.X. cartridges become harmless soon after dipping.
8. No danger from drilling into missed holes, for the cartridges are harmless if not detonated soon after loading.
9. No loss of explosive and hazard to innocent persons through theft of explosive.
10. A feasible and economical method whereby mining companies can manufacture their own explosive.
11. The liquid oxygen can advantageously be used for welding and cutting, also in mine-rescue apparatus.

The disadvantages of L.O.X. are likewise much the same as those that have been previously cited by other authorities. These are briefly given below:

1. Requires additional capital expenditure for liquid-oxygen plant and other equipment.
2. The limitation on the number of holes that can be fired in a single round is a disadvantage in large stopes and drifts.
3. Rapid and uninterrupted loading is imperative.

4. Variations in strength and general characteristics of explosive, owing to constantly diminishing oxygen content.

5. Not as convenient to use, especially in inaccessible parts of a mine.

6. With present technique not feasible to use it in holes having running water.

7. Questionable as to the advantage of using it for shaft sinking.

8. Cannot be used in gaseous or dusty coal mines.

The disadvantages cited are not serious and are more than offsetbyl the advantages.

CONCLUSIONS

We are convinced that L.O.X. will find a considerable field of usefu application in mining, quarrying and tunneling operations in this country. Improvements will doubtless be made in the present technique and there is also promise of reduction of costs. It is known that liquid oxygen can be produced more economically in a large plant than in a small one, and it is probable that the Claude cycle would be better in this respect than the Linde. The Air Reduction Co., coöperating with the Ingersoll-Rand Co., has arranged to manufacture and sell equipment for L.O.X. The first plant, which will employ the Claude cycle and have a capacity of 75 l. of liquid oxygen per hour, is now being installed at the iron mines of the Witherbee, Sherman Co., at Mineville, N. Y. This plant will be unique in that the equipment for producing the liquid oxygen will be placed underground. This unit gives promise of producing liquid oxygen considerably more economically than the small Linde unit now at Pachuca.

There is ample opportunity for improvement in absorbents and wrappers, with a view to longer "available life." The life of the Pachuca cartridge is satisfactory for rounds of reasonable size, but a more efficient cartridge in this respect would be a great advantage, if not a necessity, for very large rounds. The finding of cheaper absorbents will have an important influence in reducing the costs of L.O.X. The overcoming of the difficulties with wet holes and the overcoming of the disadvantages of using L.O.X. in shaft sinking and inaccessible parts of the mine merit attention, and it is reasonable to look forward to progress in these directions. American manufacturers of fuse can doubtless improve their product so as to render it as satisfactory for this purpose as the fuse of German manufacture.

ACKNOWLEDGMENT

The broad gage and generous policy of the management of Compania de Real del Monte in promptly rendering available to the mining world the results of their experience with liquid-oxygen explosives is to be commended. It is a pleasure to record the strong support given the inves-

tigation at every stage and the many helpful suggestions offered by D. S. Calland, director, and J. E. Smith, assistant director.

The cordial coöperation of the U. S. Bureau of Mines has proved helpful. Those to whom we are most indebted in the Bureau organization are: H. Foster Bain, director; George S. Rice, chief mining engineer; J. W. Paul, chief coal mining engineer, E. G. Gardner, mining engineer.

During the important, but difficult, initial work, J. H. Henderson, captain of the San Juan Pachuca mine, displayed much interest and perseverance. Later, H. J. Filsinger, captain of the Camelia mine, and D. A. Smith, captain of the Rosario mine, took an important part in systematizing the practice at the mines where liquid-oxygen explosives were adopted. We also wish to mention the good work of the German engineers, Messrs. von Teichmann, Klein and Schmatz. The successful outcome of a project of this character depends, in no small degree, on the interest and effective work of the men on the job; unfortunately, space will here only permit of general acknowledgment of them.

DISCUSSION

George S. Rice, Washington, D. C.—In January, 1916, at the Pan-American Convention in Washington, I heard a chemist describe the situation in Germany in 1911, when the military men wanted to go to war with France, but the German chemists said "not yet;" they had not perfected their process for obtaining nitrates from the air.

A littler later, Secretary Lane asked the technical chiefs of the Department of the Interior in what way, if the United States entered the war, could help be given by these civil establishments. At about this time, I read a German publication giving the results of tests of liquid-oxygen explosive in three coal mines in Upper Silesia. It was called "liquid-air" explosive, but I thought "liquid-oxygen" explosive was much more appropriate and this term was adopted by the Bureau in subsequent work. The term adopted by the Pachuca officials, "L.O.X.," is short and expressive and it now seems likely that it will be sanctioned by use. There had been a number of early attempts in Germany, some as far back as 1897, to use liquid-air explosives, but the results were not satisfactory. This, as it later proved, was largely because detonation is not obtained until there is over 75 per cent. oxygen; and the best results are not had until over 95 per cent., and preferably 99 per cent., purity is obtained.

Early in 1917 experiments were begun, on the basis of meager prewar information, at the experimental mine of the Bureau, near Pittsburgh. Soon the Bureau's investigators developed a technique, so as to be able to get the equivalent of about 40 to 50 per cent. dynamite; but they were handicapped by the difficulty of getting oxygen, as the liquefying plants then in this country were not adapted to supplying it. When our country got into the war, other matters were given precedence, obliging us to suspend our work in liquid-oxygen explosives.

During the last part of the war, when shortage of shipping made it difficult to import nitrates, the War Industries Board became interested; and if the war had lasted a little longer, with the increasing difficulty of bringing nitrates from Chile, probably the liquid-oxygen explosive investigations would have been pushed, as was done in Germany as a matter of necessity. Following the war I found, from giving talks in various parts of the country, that it was difficult to interest engineers in liquid-oxygen explosives except as an academic question.

During the Armistice period, I was detailed abroad and looked into the use of liquid oxygen in Lorraine, where the Germans had introduced it in some of the French iron mines for the blasting of iron ore. Also they had erected a plant in one of the great French steel works at Longwy, to assist in destruction of the works.

In the French iron mines, they were using sawdust or wood pulp for the carbonaceous material in the cartridges, in conjunction with plain fuse. For the purpose of blasting the iron ore, this combination was very satisfactory. The use of a coarse carbonaceous material and employing fuse without detonators made a relatively slow explosive but it did the work of black powder formerly used or the lower grade of dynamite.

In a paper published at that time (Bureau of Mines *Technical Paper* 243, 1919), I stated that one of the limitations of liquid-oxygen explosives was the limited number of holes that could be fired in a round. This was not a difficulty in the chamber method of mining in Lorraine but a method had been developed in Germany by preparing the wiring and detonators or fuse in advance—the detonators at the back of the hole—by which this disadvantage has been overcome.

I do not think liquid-oxygen explosives will supplant fixed explosives, except in favorable places. In coal mining, its use would be a real menace. This conclusion was not conceded by the Germans at first, but I understand they now acknowledge it. Our tests in Pittsburgh have shown that it will act like dynamite, giving a sufficiently long flame to ignite either gas or dust; there is also danger of premature firing with coal dust present. Nevertheless, there is a large field for it in other kinds of mining and quarrying where the cost of power is relatively low.

Immediately following the war, the French mining officials expressed to me great interest in this explosive, but they did not know what the attitude of the Government might be. The Government has a monopoly of the manufacture of explosives, and recently imposed heavy taxes on liquid-oxygen explosives. The amount of the tax, the Association of Mine Operators of Alsace-Lorraine felt, was unjust. The Association therefore appointed a commission to investigate the cost of production, the safety, and other features connected with it. Seventeen iron mines in

Lorraine now use liquid-oxygen explosives and produced over 5,000,000 tons of iron ore per annum; their production with other explosives was about 2,500,000 tons.

This year the U. S. Bureau of Mines has been given an appropriation for carrying on the study of this explosive; this appropriation will be be available July 1, when we expect to resume our investigations. One of the largest opportunities for this explosive, and the one about which the technique is least developed, is in "sprung" or chambered holes, for stripping work; that is one of the lines about which we will endeavor to get information.

Charles S. Hurter, Wilmington, Del.—The diagram shows a long length of fuse in contact with the explosive; has trouble been experienced because of the charges going out of rotation or being inflamed from the side-spit of the fuse? It is practically impossible to make a fuse that will not spit at the side at times. It is my understanding that if liquid oxygen is spilled on any organic substance or waste or clothing, these immediately become, for the time being, highly explosive or highly inflammable; is that so? Is there not also some danger of the liquid oxygen coming in contact with the skin and burning the person? Apparently, there is a limit of strength; in very hard and tight blasting this could not be used, on account of practically being unable to get beyond a bulk strength of about a 25-per cent. dynamite.

The strengths of the regular dynamite advance in a kind of progressive 10-per cent. increase for every 10-per cent. addition of nitroglycerin, or its equivalent. For instance, 30-per cent. dynamite is 10 per cent. stronger than the 20-per cent.; the 40-per cent. is 10 per cent. stronger than the 30-per cent., and so on. Thus, if a 1¼- by 12-in. cartridge of liquid-oxygen explosives is equal in explosive strength to a 1¼- by 8-in. cartridge of 40-per cent. dynamite, the bulk strength of the liquid-oxygen explosives will be somewhere around, and probably not stronger than, a 25-per cent. regular dynamite.

I have not had occasion to make time studies on the loading of drift rounds, and therefore do not know the difficulty of trying to load a large number of shots to be fired in rotation; how many shots can two men load?

M. H. Kuryla.—Two men have loaded and fired a round of 20 holes; the holes were about 4½ ft. deep. The time required was 9 or 10 minutes.

At Pachuca we are not boosters for liquid-oxygen explosives. We have tried here simply to state the facts and the difficulties found in a year's experience. The German manufacture sent over a fuse which did not side-spit; a number of 50-ft. lengths tested in a dark drift showed no signs of side-spitting. But the American fuse would side-spit, and the cartridge nearest the collar of the hole would ignite and burn before the cap went off. Our present fuse has been in use for about 8 months, and we have had no trouble from side-spitting.

Regarding the danger from spilling oxygen on carbonaceous material, one of our first experiences was with a stope round. When some of the cartridges dropped out of a hole, a cigarette dropped down on one that was fresh and well saturated with liquid oxygen. The cartridge immediately flared up and burned furiously, but there was no explosion.

If liquid oxygen comes in contact with the flesh, it will cause painful burns. At first we used gloves of various kinds, but found that our hands were burned just the same. We soon learned that if, while taking the cartridge out of the liquid oxygen and loading it into a drill hole, we moved it around in our hands, just as we would a hot potato, we would not be burned. Since that time, which was about a year ago, I have not heard of a single case of burning or blistering.

Weight for weight, at the instant of explosion, the liquid-oxygen explosive is slightly stronger than 40-per cent. dynamite, but to get the same breaking effect as that of an 8-in. stick of dynamite a 12-in. liquid-oxygen cartridge must be used. In some rounds where, with 60-per cent. dynamite in a face of 20 holes, it is necessary to blast two or three times with the holes loaded pretty well up to the collar, we have not been able to use liquid oxygen. The absorbent does not take up enough oxygen to become effective in extremely hard ground of that sort.

We have done comparatively little investigating of absorbents in Mexico. It was a question of putting in a trial plant and making it pay. Sometimes it is hard to carry on experiments and make money at the same time, but our plant, as it now stands, will pay for itself in about three years.

During the first few days that we tried this explosive, after loading a hole with four or five sticks, we put in a brass tube of about ¼-in. diameter, and then tamped around that tube. Then we pulled out the tube, leaving a hole for the escaping oxygen gas; that was a great nuisance, for instance, with 15 or 18 holes. As the men became more experienced, however, they could load so much more rapidly that there was not enough time to generate any substantial pressure in the hole; at present, we use no vent, although the stemming is slightly porous.

As the paper states, we have loaded and shot 28 holes in one round. Time studies indicate that men will load liquid-oxygen explosive faster than they do dynamite, not because it is easier, but because liquid oxygen stimulates quick movement.

The first three or four days we used liquid oxygen, several of us were rendered unconscious by the carbon monoxide, generated through pure ignorance and lack of experience; since that time, we have overcome this fault.

George S. Rice.—I do not think that the limit of L.O.X. strength is reached at the equivalent of 40-per cent. dynamite; strength depends a great deal on the absorbent material. The Germans have evidently developed the technique of making lampblack of extreme fineness, which they term acetylene lampblack. Liquid-oxygen cartridges using this as an absorbent are said to have a strength equivalent to 80-per cent. dynamite. Theoretically, the finer the material the better, and evidently for rapid detonation lampblack quite free from any oily matter must be used. It seems that they heat the ordinary lampblack to drive off any oily matter.

In the report of the Association of Mine Operators of Alsace-Lorraine, the period of three years when liquid-oxygen explosives were used is compared with the three years previous, when ordinary explosives were used; it is found that the accident rate had lessened. Some of the accidents have been the result of carelessness. You can return to the face immediately after the blast, without difficulty from fumes which generally are very light; certain of the accidents resulted because the men did not accurately count the shots and went back too soon, before all the shots had gone off.

There is a great deal still to be learned concerning these explosives. When Adolph Messer, one of the three German manufacturers of liquefying machinery, was visiting us recently we arranged, in coöperation with a quarrying company, to give him an opportunity to show what could be done with some 20-ft. vertical holes; also some block holes. Some of these tests were very successful but one group of three holes failed to go off. Mr. Messer used an untried method, sand for stemming as there was no other loose material available for the purpose. The detonators, which were in wooden blocks, were lowered on top of the cartridges and then the sand was poured in; the vaporization of the liquid oxygen lifted the detonator blocks, floating them on the sand and away from the cartridges, so that there was no explosion. But I was rather glad of the occurrence for he then showed us a method of unloading that was interesting and valuable. After waiting sufficient time for the oxygen to be all vaporized he gradually forced into the hole a compressed air pipe and blew out all the lampblack and cartridge paper clear to the bottom. It is a very rapid and safe unloading method.

R. M. Raymond, New York, N. Y.—On what hardness of rock should liquid-oxygen explosives be used? It has been intimated that the explosives cannot be satisfactorily used on the very hardest rock. Can they be used on the hard magnetite ores of New Jersey, or the Adirondacks, or those hard magnetite rocks that require a hole to be completely filled with 60-per cent. dynamite to get a proper explosion?

O. P. Peterson, Hinsdale, Ill.—W. Myron Davey, who is now chief geologist of the Real Del Monte says: "The country rock is dense, uniform, andesite porphyry flows usually hundreds of feet thick, and for the most part can be considered free of bedding planes or any important planes of weakness. Usually several slips parallel the veins, but not closely enough spaced to assist materially in breaking up the rock. The quartz vein filling is often uniform and uninterrupted by joints for widths of 10, 15 or 20 ft., and the large blocks of both quartz and andesite in the stopes are of jagged, irregular outline on all sides, pointing to a lack of weakness planes of any great importance."

It is not as hard as silicified limestone of the Bingham district. I do not think anything will compare with that, which is the hardest rock I have encountered underground. It does not compare (that is, the average rock) with the thick-bedded limestone of the Bingham district, and with the fresh, unaltered monzonite of the Utah Copper Co., for it is not affected by closely spaced joint planes. It compares with a normal, fresh granite such as you might find in the Lake Superior country or in Butte. As far as I know, not many tests have been made on extremely tough ground, because ground as tough as the silicified limestone of Bingham is not usually found.

Glenn A. Keep, Elizabeth, N. J.—What is the limit of the cost for power? Is this method economical and will these explosives compare favorably with other explosives? Is the company planning to extend its activities so as to use these explosives throughout the Pachuca district, and perhaps introduce it into the metal mines in the States?

M. H. Kuryla.—With regard to influence of power cost, page 315 contains a summary of the major items that enter into the present cost of liquid oxygen; this summary shows that power represents 20 per cent. of the total cost of the explosives. There could be quite a fair increase in power cost and still leave the cost of liquid-oxygen explosives below that of dynamite.

With regard to the extension of the use of liquid oxygen in Pachuca, there are a number of factors that must be considered. First, as we use a considerable amount of power, about 18,000 hp., of which 4000 hp. is for pumping, and there is a shortage of power at the present time, it would not be feasible for us to extend the use of these explosives. Besides, we are novices in this and are investigating other processes of making liquid oxygen. Some other processes of liquefaction are much more economical of power; in fact, the power consumption would be less than half of what we are using with our old-style plant.

We have not tried these explosives in any of our mines in the United States as yet. We want to get the data of a full year's operation with our Pachuca installation before we do anything in this country.

Galen H. Clevenger (authors' reply to discussion).—On page 287, when describing the transport and dipping containers, the statement is made that all metals are porous and permit gases, including air, to pass slowly through them, thereby rendering it difficult, if not impossible, to maintain a vacuum between metallic walls unless some highly adsorptive agent for gases, such as charcoal is known to be at the temperature liquid oxygen, is present in the vacuum chamber.

The idea conveyed, although supported by the quite recent literature, is perhaps misleading in the light of recent achievements in the production of high-power vacuum tubes. A portion of the surface of these tubes is water cooled, therefore it is the present practice to build them up by fusing a glass section to a copper section, which can be water cooled. Despite the fact that charcoal or other adsorptive agent is not used in the tube, one of the well-known workers in this field has informed us that tubes having a considerable portion of their surface of copper have retained a high vacuum for two years. Much depends on the preliminary treatment the copper has received and especially the completeness with which the adsorbed gases have been removed.

While it is doubtful if the use of charcoal will ever be dispensed with in the metallic vacuum containers used in connection with L.O.X., it seems probable that through the use of copper that has been made dense and homogeneous and from which the adsorbed gases have been removed and through greater care in manufacture that eventually a container will be produced which will have a much longer life than those now in use, the average life of which is about one year.

Alfred James, London, Eng. (written discussion).—The author's method of charging the holes may not improbably lead to the solution of the present difficulty in firing large rounds of shots. The capped fuses can be placed in position in the various holes and the electric contacts

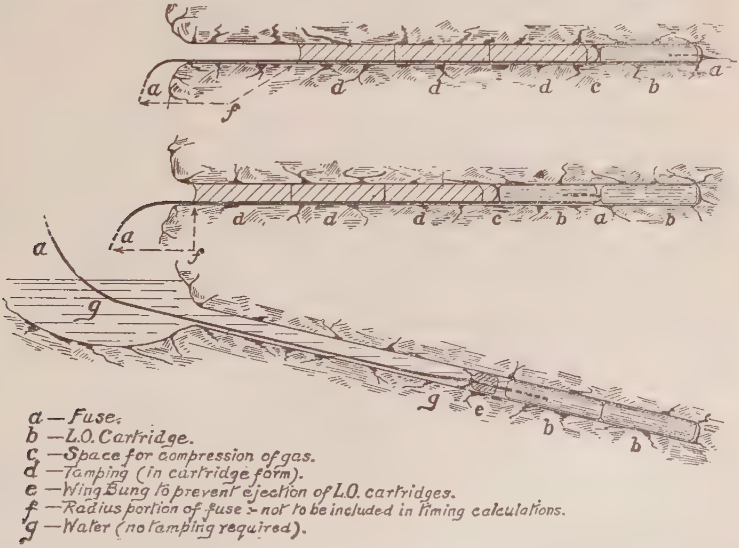

a
a
d
d
d
c
b
f
a
d
d
d
c
b
a
b
f
a
g
g
e
b
b
a—Fuse.
b—L.O. Cartridge.
c—Space for compression of gas.
d—Tamping (in cartridge form).
e—Wing Bung to prevent ejection of L.O. cartridges.
f—Radius portion of fuse :- not to be included in timing calculations.
g—Water (no tamping required).

Fig. 17.—Method of charging holes in Lorraine.

made at leisure. The cartridges and tamping can then be rammed in rapidly, with a considerable margin of time available from the adoption of electric firing in place of the 5 min. taken by fuse ignition. I am, however, much impressed by the thorough tamping usual in Lorraine. Fig. 17 shows the large amount of tamping ordinarily introduced there, there, and the convenience of its insertion in cartridge form.

The total output of iron ore in Lorraine for 1922, with liquid-oxygen explosive, was about 4,089,300 tons. For this tonnage, the tamping method here shown appears to have been responsible for two mishaps, the tamping blowing out against the right arm of the man tamping, in one case, and against the left hand of the man in the other; the wounds, however, were not serious. The growing indifference of miners to any danger from handling these supercool cartridges accounts for the two last mishaps of the year, in which the hand or thumb was burned as the cartridges were being withdrawn from the soaked vessel. These injuries also were very slight. Such accidents are avoided at de Wendel by pushing a pointed iron rod into the end of the soaked cartridge. The pierced cartridge is then lifted out, caught in a fold of the coat, and the fuse rapidly inserted into the hole made by the iron rod which is of the same diameter as the fuse.

Abstract of Report on the Use of Liquid-oxygen Explosives in the Iron Mines of Lorraine, Dec. 1, 1922, by a Technical Commission of the Mining Association of Alsace and Lorraine

Note.—The remarks in brackets are those of George S. Rice who abstracted this report.

The industrial trials of liquid-oxygen explosives began 20 years ago. Liquid oxygen was tried in the driving of the Simplon tunnel and it was patented in Germany under the name of oxyliquit; but its use was soon abandoned. The "Sprengluft" Association again took up the problem in 1910 and has carried on since that time a strong propaganda.

In France, the work of Georges Claude in the liquefaction of oxygen and [in England] that of Dewar in the absorption of gases by charcoal at low temperature and their preservation in double-walled vessels, contributed to the practical utilization.

As the nitrates from Chile had to be reserved for the needs of the army, during the war Germany began to use liquid oxygen as an industrial explosive. Since 1915, it was used in a number of mining enterprises in Silesia; and, in 1918, the French troops, upon reentering the Lorraine, found several mines provided with installations for the making of liquid-oxygen explosives; the de Wendel Co. had pushed the utilization by means of special cartridges since 1914.

Today liquid-oxygen explosives are currently used in 18 mines in Lorraine. These mines have 27 liquid-air machines with a total hourly capacity of 1005 liters. [The description of the procedure has been given in the paper by Kuryla and Clevenger.] In a description of the process and technique employed, there is the statement that liquid oxygen may be produced by machinery established either on the surface or underground. An installation underground has a lower efficiency on account of the difficulty of purifying air heavily charged with carbon dioxide; while this can be remedied by aspirating from the surface, the overload thus produced hinders the operation of the machine.

Processes.—1. In the Georges Claude, after purification, the air compressed to 40 atmospheres is allowed to expand in two stages with exterior work, in an actual compound engine, combined with a dynamo brake designed to recover the energy given off by the expansion. After liquefaction, the air passes into a rectification column which supplies liquid oxygen.

2. The Linde employing an adiabatic expansion, to which class belong the German apparatus [chiefly Linde] used in the iron mines of Lorraine. The purified air is compressed to 200 kg. This air liquefies itself after cooling and expands and runs into the separator at the outlet of which the liquid oxygen is recovered.

3. The Jaubert, derived from the preceding, which, by the action of the expansion and the arrangement of cooling by means of a circulation of nitrogen gas and liquid oxygen in a temperature exchanger, permits being obtained at will, from the same apparatus, gaseous or liquid oxygen. [This description would apparently also apply to the Messer process used extensively in Germany, there also being some plants in Lorraine.]

Carrying and Dipping Containers.—Liquid oxygen is transported into the mine and to storage places in special brass receptacles in glass or porcelain with double walls between which there is a vacuum. The miners are given soaking vessels constructed on the same principle and the cylindrical form of which permits an easy impregnation of the cartridges. [The carrying and dipping containers used in Lorraine, in 1919, were all of German manufacture; the diagram shown in the report indicates that the German type is still used.]

Cartridges.—The properties of the cartridges vary according to their constituents. One can obtain the range of effects produced by ordinary explosives. Brissant cartridges can be made, the effect of which is about two and one-half times as great as dynamite [presumably 30 or 40 per cent. strength dynamite]. They can be used under water. The duration of the impregnation of the cartridges by liquid oxygen must be sufficient for complete combustion; carbon monoxide is then not produced on explosion. The time of impregnation must be at least 10 min. The length of the life of the cartridges, figured from the time they are removed from the soaking vessel and the combustion commences to a time when combustion can no longer be completed when CO would be produced, can be varied up to 20 min. to permit firing promptly, or firing very large charges.

Formerly, it was considered necessary to have a channel along the side of the cartridge open to the air, to aid in the liberation of the vaporized oxygen. This is now considered unnecessary for the gas easily escapes through interstices of the stemming and fissures of the ore. [This presupposes the use of sufficiently coarse material in tight rock to permit the escape of the gases. Sand, for example, was found to be too fine for a large hole at the Martinsburg, W. Va. test.] It is recommended that the first stemming be not compressed and that the use of impermeable material be avoided to prevent its being thrown out of the bore hole, although the pressure that can be attained in the interior of a bore hole by the liberated gas is relatively feeble.

Criticisms.—After great enthusiasm, because of the appreciable economies attributed to liquid-oxygen explosives, criticisms are more and more frequently made and the recent decision of the Administration of Finance to impose a heavy tax on the new explosive, serves to discredit it. Its detractors became so numerous that one might sometimes ask if the agitation were not in reality a small cabal, managed in the service of certain private interests.

Liquid oxygen has been accused of presenting grave dangers from explosions and manipulation. The operators of the iron mines of Moselle were stirred by such statements and entrusted to this Commission the study of the question in all details, conducting if necessary experiments. In a footnote, reference is made to a paper by M. Vinnany criticizing liquid oxygen unfavorably. The sequel of this report shows that if certain of M. Vinnany's observations may be justified in coal mines, and especially in dusty and gaseous mines, none of them can be considered as applying to iron mines.

The Germans still use, even in firedamp coal mines, a "safety liquid-oxygen explosive" but serious accidents have compelled them gradually to abandon this procedure, which is found now but rarely in coal mines. Probably some day it will be possible to prevent the spilling of liquid oxygen on the ground. Absolute saftey cannot be assured until it will be possible to prevent contact of oxygen with the coal dust.

Safety.—Although liquid-oxygen explosives have been considered as possessing an immense advantage over other explosives in becoming inert after a certain time of waiting, the following objections have been made:

1. That they frequently burn.
2. That they are subject to delayed explosion and premature explosion.

As regards burnings, the burning takes place only when the cartridge has been in contact with the liquid oxygen for too long a time. Accidents of this kind are always much less serious than burnings of other explosives and common sense forbids the comparison to the risk of a workman with an open lamp transporting powder, with the risk of a workman carrying a can of liquid oxygen under the same conditions.

Accidents connected with storage are avoided when liquid oxygen is used as compared with the storage of powder.

Deflagrations, or explosions before the placing of ordinary explosives in the bore hole, which are generally caused by the falling of a lamp, are extremely rare with liquid oxygen. The time during which the explosion is dangerous is reduced to a minimum, as the sensitive cartridges can be exposed to the fire of the lamp only during the short trip from the dipping container to the bore hole.

Delayed explosions are less with liquid-oxygen explosives than with other explosives, as in most cases the cartridge that explodes after a time of waiting has lost much of its strength. The waiting time varies with the diameter and the amount of the charge and the degree of saturation. In principle, the [mine] regulations permit a return to work in the case of a misfire 1 hr. after the time of lighting. It is permitted to reduce this time to ½ hr. for liquid-oxygen explosives. Investigations concerning accidents of this class with liquid-oxygen explosives have not allowed us to assert that the latter time has been strictly observed; on the contrary, the time has been found to have been disregarded. Moreover, statistics show that these accidents are less numerous with liquid oxygen.

Premature explosions have occurred while the bore hole was being charged. Such accidents have nearly always been due to recharging of the blast after a blow out.

When recharging after a blown-out shot, the regulations require that a ball of wet clay be placed in the bottom of the bore hole before recharging. This regulation has not always been observed, so it may happen that a part of the old charge or scraps of the cartridge incompletely burned will set fire to the new cartridges put in. With liquid air the danger is greater, because on putting in new cartridges one introduces into the bore hole an excess of a supporter of combustion [oxygen].

Acetylene and oxygen mixed in fixed proportions form a detonating mixture. The danger is more serious in the case of liquid oxygen, as under the influence of the intense cold the acetylene may be solidified in little crystals sensitive to shock. It may have happened that some workmen added some calcium carbide to the cartridge in the hope of increasing the effect of the shot. It may also have happened that the cartridges coming in contact with the calcium carbide, which is frequently handled in the iron mines, have exploded spontaneously if the conditions of humidity necessary to the release of acetylene were present or if the carbide contained impurities—phosphide of calcium, for example—and could thus release a gas which is liable to spontaneous combustion. These, however, are only hypotheses that need more thorough study.

In the meantime, in the interests of safety, it is recommended that where acetylene is used for lighting and liquid oxygen is used as an explosive, to avoid the presence of calcium carbide in the intake of the air supply to the liquid-oxygen machines and all chances of contact between the cartridges and the carbide receptacles.

It has been alleged that liquid oxygen accidentally spilled over a fuse could increase considerably its speed of combustion. Tests of a highly conclusive kind allow us to reject such a hypotheses completely and to assert that liquid oxygen exerts no other direct action on the fuse than retarding the combustion, due to the cooling and to the frost that deposits on its surface as a result of the soaking. Nevertheless, it is important to point out that the fuse may be in an atmosphere rich in gaseous oxygen, produced by the evaporation of the impregnated cartridges. Testing, in a mine, allows us to reach the following conclusion.

In holes charged with liquid oxygen, the gaseous oxygen may promote the combustion of the outer envelope of the fuse in the places where the fuse is not completely buried in the stemming. In one particularly unfavorable case an increase in the rate of burning of about 20 per cent. was observed. It should be noted, however, that this acceleration is partly due to the pressure exerted on the tamping on the fuse; this had been determined in previous investigations in a coal mine with compressible materials. In the case of iron mines, it does not appear that the hypothesis need be considered, for the tamping was done exclusively with powdered ore, which does not permit the development of high pressure. A recent test in one of the iron mines showed that where tamping material was put in very forcibly it showed an increase in speed of burning of the fuse of 6 per cent. The conclusion is that with suitable tamping the acceleration of burning due to the influence of gaseous oxygen is practically nil.

Although the statistical reports in the last three years do not mention any premature explosions occurring after lighting the fuse, for the sake of safety the following precautions are recommended:

1. Forbid strictly the firing, by a fuse, of cartridges of liquid-oxygen explosives placed in a bore hole without tamping.

2. Do the tamping as carefully as possible so as to avoid any breach of continuity of the stemming.

3. Use fuses a little longer than for powder and projecting to a sufficient length out of the bore hole. In calculating the length of the fuse, only the part buried in the tamping and that projecting from the mouth of the borehole is to be taken into account.

It would be desirable if the manufacturers of miners' fuses would try to produce incombustible wrappings at a low price.

Spontaneous explosion of cartridges under pressure of gas in the bore hole, the question in regard to this being whether the pressure of gas in the interior of the bore hole before firing would be sufficient to produce spontaneous inflammation. Tests were made and the conclusions were:

1. When a bore hole charged with liquid oxygen is tamped, the pressure rises rapidly and within 2 to 6 min. reaches a maximum which seems

to depend chiefly on the temperature of the bore hole. This maximum is at the rate of 0.5 kg. (1.1 lb.) for a bore hole at the surrounding temperature and charged with three cartridges.

2. After a certain time, the pressure curve seems to be determined chiefly by the nature of the tamping.

3. A sudden throwing out of the tamping has not been obtained through the pressure produced in the bore hole by the injection of compressed air or oxygen. A cylinder charged to 120 kg. never produced more than 25 kg. of pressure in the bore hole, in spite of tamping compressed as tightly as possible. Cartridges subjected to the pressure of compressed oxygen up to 25 kg. in the bore hole did not explode. They

STATISTICS OF WORKMEN INJURED DURING THE YEARS 1919, 1920, 1921 IN THE IRON MINES OF MOSELLE

	1919	1920	1921
Number of mines with liquid-air installations.	17	17	17
Tonnage mined by these mines.	4,826,182	5,445,381	5,315,104
Total tonnage by all mines in the basin	7,137,206	8,061,514	7,817,126
Tonnage mined by mines using powder	2,311,024	2,616,133	2,502,822
Number of liquid-air machines	24	24	24
Total hourly capacity of all these machines, liters	715.5	728	869
Accidents with liquid oxygen and liquid-oxygen explosives:			
Cause			
Burned by liquid oxygen during transportation	3	2	5
Burned by L.O.X. in contact with flame outside of the bore hole	1	0	1
Delayed explosions or premature return to work	1	2	0
Premature explosions	4	0	7
Miscellaneous	0	0	0
Total	9	4	13[a]
Per 1000 liters of liquid oxygen used	0.024	0.008	0.023
Accidents with powder:			
Burns in the room or during transportation, by contact of the powder with a flame	16	14	4
Delayed explosions or premature return to room	11	3	0
Premature explosions	4	1	2
Miscellaneous	7	0	1
Total	58	18	7[b]
Per 1000 kg. of powder used	0.061	0.017	0.008

[a] 2 deaths, 2 serious.

[b] 1 death, 4 serious.

	1913 Various Explosives Liquid Oxygen not Used [Number per 1000 Workmen]	1921 Various Explosives Liquid Oxygen Used in 17 Mines [Number per 1000 Workmen]
Slight accidents	5.0	2.535
Serious accidents	0.48	0.340
Mortal accidents	0.48	0.50
Total	5.96	3.375

therefore assert that the hypothesis of spontaneous explosion by pressure of oxygen is untenable.

The accompanying table gives the ratio per 1000 workmen, of accidents with explosives in all the iron mines of Moselle in 1913, when liquid-oxygen explosives were not used, and in 1921, when liquid-oxygen explosives were used in 17 mines [producing two-thirds of the iron production].

As concerns the increase of accidents with liquid-oxygen explosives during 1921, it is necessary to examine into the circumstances of each accident and its severity. The records show that the accidents caused by liquid-oxygen explosives belonged for the greater part to the class involving less than 13 weeks incapacity for work.

The consumption of liquid-oxygen explosives in the iron mines of Moselle, in 1919 was 372,000 liters; in 1920, 499,000 liters; and in 1921, 558,150 liters.

Hygiene.—Liquid-oxygen explosives have been said to produce harmful gases, particularly a large proportion of carbon monoxide. This is the result of the methods used in the first application of liquid oxygen. The claim is no longer well founded with cartridges easy to saturate and with long life. Samples gathered in tests continued for more than a week in one mine have shown that only unappreciable traces of carbon monoxide, and these without effect on the human body, could be detected in certain badly ventilated workings.

Technical Advantages of Liquid-oxygen Explosives.—The possibility of obtaining a better strength efficiency than with ordinary explosives, for the use permits choosing types of cartridges suited to the physical condition of the rock.

Good visibility after the shot, as the air is charged only slightly with fumes absolutely harmless, thus decreasing the result of accidents from a fall of rock or ore when the workman returns.

An appreciable enrichment of oxygen in the mine air, resulting in an appreciable economy in the power consumed by [mechanical] ventilation.

An important increase in production for the miners by advantages of visibility and ventilation permitting earlier return when they are certain of the firing of all the blasts.

5. The practical impossibility of the workmen carrying the explosives home and selling them unlawfully.

6. The utilization of a liquid-oxygen installation for the manufacture of gaseous oxygen for welding.

7. The advantage for national defense which will permit, in case of a crisis, a decrease in the consumption of nitrate explosives of which the army will have urgent need and the lowering of exchange [through non-purchase of nitrates from abroad].

Relative Cost of Liquid-oxygen Explosives.—Part III relates to the cost of liquid-oxygen explosives in relative equivalence to other explosives used in the district, such as brown powder. The first comparison is made on the basis of practical equivalence, as result of tests in a mine over a period of several months with powder and for a similar period with liquid oxygen. It was determined that, under the given conditions, 1.11 liters of liquid oxygen measured at the time of distribution was required to produce the effect of 1 kg. of powder; the liquid oxygen was about 95 per cent. oxygen. Similar tests in other mines showed that the equivalence varied from 1.05 to 1.30; 1.20 was considered the mean coefficient. That is, 1 kg. black or brown powder equaled in practical effect 1.20 liters of oxygen, at the point of distribution to the workmen.

Losses from Evaporation during Transportation.—The hourly loss for small containers was determined as 0.5 per cent., and with the large containers 0.1 to 0.2, depending in part on the age and condition of the containers. When studying the losses in different iron mines during transportation to the point of distribution in ten mines, it was found that the average loss was 23 per cent., giving a mean coefficient of 0.77. Under the cost of production at seven mines, the consumption per liter of oxygen produced and measured at the machine is shown in the accompanying table.

Consumption per Liter of Liquid Oxygen Produced and Measured at the Machine

Mine	Kw.-hr.	Potash, Kg.	$CaCl_2$, Kg.	NH_3, Kg.	CO_2, Kg.	Oil, Kg.	Cooling Water	Labor and Repairs
A........	3.49	0.032	0.0136			0.0196		0.210
B........	2.96	0.040	0.018	0.004		0.020	600 li.	0.281
C........	3.828	0.031	0.012		0.02	0.03		0.280
D........	3.60	0.040	0.0029			0.0096	200 li.	0.274
E........	4.19	0.050	0.003			0.011		0.227
F........	2.60	0.042	0.0016			0.015		0.196
G........	2.32	0.0429	0.0296			0.0198	300 (?)	0.350

The favorable figures for mines B and F correspond with installations that permit obtaining either oxygen or gaseous oxygen and can be operated continuously; this is an advantage that cuts the cost of starting. Many machines have run 2½ hr. before they are sufficiently cold to give results.

Estimate of Cost of Producing Liquid Oxygen.—The average cost of power to manufacture locally is 0.34 franc per kw.-hr. The average price of current at high tension, during 1921, was 0.278 franc per kw.-hr., to which the cost of transforming must be added; 0.30 franc is considered to be the average for Lorraine.

The cost of the materials used is: potash 2.34 francs per kg.; calcium chloride 0.80 francs per kg.; ammonia 6.50 francs per kg.; oil 1.74 francs per kg.; cooling water 0.30 francs per kg. cubic meter. Using these cost figures the total cost of manufacture of liquid oxygen per liter at the machine varies at the different plants from 1.10 francs to 1.60 francs; the commission estimates 1.45 francs as a weighted average. To this manufacturing cost must be added the amortization and machine installation. Liquid-oxygen machines are offered for sale at the following prices, June 19, 1922.

Installation with an hourly production of

5 liters	75,000 francs
12 liters	105,000 francs
25 liters	150,000 francs
50 liters	235,000 francs

These figures include necessary apparatus for compression, purification, drying, liquefaction, as well as the purification of the liquid air obtained, but they do not include motors. The commission allows for motors: for 40-hp., 10,000 francs; 75-hp., 15,000 francs; and for 150-hp., 20,000 francs. The commission takes as a basis of comparison the machine of 25 liters capacity, as this seems to meet the needs in the majority of cases [the extraction of 1000 tons per day or 300,000 tons per year].

The cost of installation, including buildings, is estimated at 50,000 francs. It is necessary to have carrying and dipping containers; for each room producing 25 tons a day there is required a 3-liter dipping container and a 5-liter transport container. The number of containers should be increased by 24 per cent. to take care of repairs. The cost of a 5-liter transport container is 150 francs and a 3-liter dipping container is 100 francs. Total cost necessary for the operation: amortization charges (*a*) buildings, 50,000 francs, 10 years, (*b*) mechanical parts, 165,000 francs, 10 years, (*c*) containers, 15,000 francs, 3 years. On the basis of annual production of 80,000 liters of liquid oxygen measured at the machine, amortization cost per liter of liquid oxygen was 0.483 franc. Therefore the cost of industrial production per liter of liquid oxygen at the machine is 1.45 ÷ 0.48 = 1.93 francs.

The cost of production of liquid oxygen at the point of distribution 1.93 ÷ 77 = 2.505 francs. To obtain an effect equivalent to that produced by one kilo of powder it is necessary to pay 2.505 francs times 1.20 plus the cost of the fraction of a cartridge corresponding to 1.20 liters of liquid oxygen.

Determination of Absorbent Power of Cartridges.—The absorbent power of a cartridge varies according to the composition and the diameter of the cartridge. Figures from 13 mines show an absorbtive power for one liter of liquid oxygen varying from 0.230 to 0.392 kg.; averaging the 13 mines gives a figure of 0.292 kg. Therefore to absorb 1.20 liters of liquid oxygen (equivalent to 1 kg. of powder) will require 0.350 kg. of absorbent. The Weber No. 3 cartridge, currently used in the district, weighs 0.125 kg. and costs 0.20 franc, taxes not included. The cost of the fraction of cartridge absorbing 1.20 liters of liquid oxygen is 350 × 0.20 ÷ 125 = 0.56 franc.

Cost of production of a quantity of liquid oxygen explosive equivalent to 1 kg. of powder: 1.20 liters of oxygen at distributing point at 2.555 francs per liter, total 3.00 francs; 0.350 kg. of cartridge totals 0.56 franc making a grand total of 3.56 francs.

Comparison of Costs, Taxes not Considered, of Liquid Oxygen and Powder.—A kilogram brown powder costs the operator: 4.50 francs plus 0.15 franc for handling, guarding, and amortization of magazines, total 4.65 including 0.60 francs tax. If the tax is not considered in either case there is an advantage of about 0.50 francs in favor of liquid-oxygen explosives.

Conclusions.—L.O.X. presents incontestably great advantages of safety, hygiene and utility over other explosives.

The general use of this explosive is desirable for it would lower the cost of production in many mines.

It would permit greater economy of nitrates, which we are obliged to import, and, in case of a crisis, would be of great value in the national defense.

Taxes.—The tax imposed on the cartridges is calculated on the weight of the absorbent material; namely, 7 francs per kg. on soot, 4.25 francs per kg. on wood pulp, paper or aluminum. For the common cartridge of 125 gm. the tax is 0.46 franc. The tax carried by a quantity of L.O.X. equivalent to 1 kg. of powder is 350 × 0.46 ÷ 125 = 1.28 francs. The maximum revenue to the state for 1 kg. of powder is 0.60 franc.

1. L.O.X. pays, for the same result, a tax twice as high as powder, and which is 43 per cent. of its cost of production at the distributing point.
2. The imposing of this tax has raised the cost of production of liquid oxygen from 3.56 francs to 4.84 francs and makes it more costly than powder.

3. To obtain in the iron mines the equal cost of production for L.O.X. and powder, it will be necessary that the cost per kilowatt-hour should not be greater than 0.26 franc. The average power consumption for for L.O.X. equivalent to 1 kg. of powder is 5.14 kw.-hr.

4. As a result, the margin of economy obtained by the use of liquid oxygen is no longer sufficient to interest iron mines in new installations.

Then follows an argument that under the method of paying the workmen it would be necessary to raise the price of liquid-oxygen explosives which would reduce the workmen's pay and that this is a contingency practically impossible.

Appendix I.—Following the report proper there is an extended appendix describing tests made on the greater combustion of fuse in the presence of liquid-oxygen explosives; on the measurement of the quantity of oxygen given off by a cartridge impregnated by liquid oxygen and placed in a bore hole, both with and without tamping; the burning of fuse in an atmosphere of oxygen gas, regarding which the conclusion was reached that in the tamped portion the burning of the fuse and current of the oxygen gas proceeded normally as if the current of oxygen did not exist.

Appendix II.—Tests were carried out to obtain the pressure of oxygen within a charged bore hole. There were two tests: (1) the pressure obtained in a bore hole, loaded with liquid-oxygen cartridges and caused solely by the evaporation of the oxygen; and (2) the pressure to which tamping can be subjected and its permeability. Under the first test, it was observed that the pressure rose rapidly with the first tamping, reaching in less than 1 min. a little higher than 150 cm. of water.

In the matter of resistance to pressure of ordinary tamping: There was a pressure of 5 kg. behind the tamping. Conclusions are that: (1) when one tamps a bore hole charged with liquid oxygen, the pressure rises rapidly and reaches at the end of 1 min. a maximum which depends chiefly on the temperature of the bore hole. This maximum will be of the order of 150 cm. of water for a bore hole taken at the surrounding temperature and charged with four cartridges. (2) That at the end of a certain time, about 5 min., the pressure curve seems determined chiefly by the nature of the tamping. (3) The maximum resistance of the tamping is far from being reached in the cases studied. (4) In view of the small difference between the pressure obtained and the atmospheric pressure, there does not seem any possibility of spontaneous combustion within a bore hole.

Appendix III.—Experiments were conducted to determine the sensitiveness to shock of solidified acetylene. The tests were not elaborate but confirmed what was already known about the properties of acetylene and dangers from its storage and its transportation.

Appendix IV.—The second series of tests described in this appendix were conducted in a Nord mine, on pressures which could be obtained by a bore hole charged with liquid-oxygen explosives. A pressure of 0.50 kg. was obtained. The stemming material was powdered ore and clay well tamped.

Appendix V.—This appendix describes the accidents with different kinds of explosives used in Lorraine. Only those relating to liquid oxygen will here be mentioned. [The numbers used refer to official notifications of accidents.]

Appendix VI.—This appendix includes the decree of April 12, 1921 fixing the tax applicable to liquid-oxygen explosives.

ACCIDENTS FROM USE OF LIQUID-OXYGEN EXPLOSIVES

1919

1. Burns by liquid oxygen during transportation.

No 11.—During the distribution of liquid oxygen the receptacle fell on the leg of B; result, water on right knee.

No. 12.—F, with several workmen, was carrying containers holding about 15 kg. of liquid oxygen when some liquid spilled on the hand of F and caused burn; one month rest.

2. Burns by explosion of liquid oxygen in contact with a flame outside of bore hole.

No. 37.—After having charged several bore holes, M had thrown half of a cartridge of liquid oxygen on the ground. When the blast had gone off, M stopped to observe the effect of the explosion and placed his lamp on the ground. His lamp tipped over and set fire to the cartridge. M was wounded by the explosion, right hand being badly lacerated and right eye injured.

3. Delayed explosions on premature return to room.

No. 36.—S and helper were at work in a gallery parallel to the principal gallery. Having lit two blasts they retired. After ½ hr. waiting and having heard one blast, they returned when suddenly a second hole went off. S was slightly injured on his left hand and G was burned about the face.

4. Premature explosions.

No. 23.—A bore hole 2 meters in depth, which G was loading the second time with liquid-oxygen cartridges, suddenly exploded. The cartridges probably took fire from the contact with a piece of paper lit by the explosion. The flame shot out from the bore hole and burned the workman; first degree burns on legs and abdomen.

No. 24.—M was engaged in loading a bore hole with liquid oxygen; the cartridge, pressed too hard, exploded throwing grains of sand in the eyes of M.

No. 45 and No. 46.—H was charging a bore hole with liquid oxygen, aided by helper F, when from some unexplainable cause the blast detonated and wounded both of them, causing fracture of the right arm and wounds to head and chest of H and severe wounds to head and chest of F.

1920

1. Burns by liquid oxygen during transportation.

No. 3.—On taking hold of a container of liquid oxygen to pour it over some cartridges P turned receptacle upside down, causing second degree burns on left hand.

No. 5.—F, while dipping some cartridges into liquid oxygen, burned his fingers.

2. Burns by explosion of liquid-oxygen explosives in contact with flame outside of bore hole.

None.

3. Delayed explosions on premature return to the room.

No. 1.—A miner S lit two trimming shots charged with liquid oxygen. After the explosion of the first, the other hanging fire, the miner entered the room contrary to rules. The shot went off and tore off his lower left arm; 6 weeks rest.

No. 2.—B, after waiting 40 min., returned to a blast which had missed. It blew out upon tamping and slightly injured B on the body. Slight injuries to the body and right knee; 8 to 10 days rest.

1921

1. Burns by liquid oxygen during transportation.

No. 7.—W, while transporting liquid oxygen, was burned on the index finger of right hand.

No. 8.—B, while lifting off a liquid-oxygen container, the liquid oxygen splashed out of can and burned two fingers of right hand.

No. 9.—K, while unloading a wagon of liquid-oxygen containers, was burned slightly on fingers of right hand by jet of liquid projected by the shock to the containers.

No. 10.—M, while charging a blast, let several drops of liquid oxygen fall on his right hand.

No. 13.—C, while withdrawing a liquid-oxygen cartridge from an impregnation vessel, the cartridge broke burning him on left hand; 15 days rest.

2. Burns by explosion of liquid-oxygen explosives in contact with a flame outside of bore hole.

No. 6.—T was preparing a blast with liquid oxygen without having his lamp sufficiently far off. An explosion took place, the flame penetrating his left ear, tearing his eardrum.

3. Premature explosions.

No. 1.—P was charging a blast after having inserted the first cartridge. The blast blew out and powdered stemming hit him on the body and eyes causing serious injury to eyes.

No. 2 and No. 3.—A miner B had charged a blast in the roof with two cartridges of liquid oxygen. After lighting them G and B retired to a crosscut. When the blast did not go off after 1 hr., they returned to the room. B remarked that the blast had blown out and commenced to reload it, standing on a ladder. G was passing the cartridges. After insertion of the fourth cartridge a blast occurred killing B and causing slight injury on the head to G.

No. 4 and No. 5.—A fatal accident was caused by the premature explosion of a bore hole charged with liquid oxygen while the miner was engaged in reloading it; T was killed and N burned and seriously wounded.

No. 11.—K was charging a bore hole with cartridges. He introduced one and the second stuck in the middle. He drew this out with a scraper and tried to enlarge the hole with a drill. The cartridge in the back exploded and a piece of the ore struck K causing wound to body, forearms, and eyes; 10 days rest.

Installation of Fire-fighting Equipment in Mines

By Benjamin F. Tillson, Franklin, N. J.

(New York Meeting, February, 1923)

Although portable fire extinguishers are valuable for fires in an incipient stage, some medium that will dissipate a large amount of heat is needed to fight a fire in mine timbers, because of the size to which the fire will probably have grown before it is discovered. Sealing off, or otherwise smothering such a fire with gases that will not support combustion, is often ineffective for the residual heat will cause reignition when the area is opened and fresh air reaches the hot charred timber.

The most effective ways of fighting mine fires, therefore, are to drown and cool the hot zone with water, and to "load out" the partly burned material. Either of these methods requires a direct attack of water upon the inflamed material; therefore, every mine containing a large quantity of timber or other combustible should be as completely equipped with a fire-hydrant system as it is with compressed-air mains. Mine managers may object to the expenditure, especially if they have not suffered from mine fires. The cost of such preventative measures, however, is only a small portion of the possible losses, and the water pipes can be used for other purposes, which will pay the carrying charges. They can supply water to the rock drills, which will encourage more wet drilling with hollow-drill steel, for the trouble and expense of packing water to individual tanks at the drills will be removed.

The carbon-tetrachloride hand extinguishers seem to have no place in mining except for fires that start in electrical equipment. They give off noxious gas, which may prove poisonous in confined quarters; and as it is heavier than air it is of no value for smothering fires overhead or in side timbers and would be dissipated by a good ventilating draft.

The old soda-acid extinguisher or the more modern "foam" types are preferable. The former probably does not rely for its efficacy on the carbon dioxide generated but on the smothering and cooling effect of the water and the trajectory of stream that may be obtained because of the gas pressure developed. The production of a foam having a tough film to smother a fire, even on overhead surfaces, with the added advantage of holding a greater quantity of cooling water against the burning surfaces, would seem to be the best for fighting incipient mine-timber fires.

Fire Patrols

Fire patrols should examine a mine between the regular working shifts; one round should be made immediately after the last working shift and a second one within 4 hr. Time clocks should be provided to assure proper watchfulness. It is possible to get time clocks for servicing 100 or more stations. The time clocks at the Franklin mine are for 64 stations, which cover as large a territory as one man can inspect.

In a large mine, there should be several mine telephone stations on each fire-patrol route. The telephones at shaft stations lose their value when access to the shaft is cut off; besides, the interior telephones save time in reporting a mine fire and in the promptness with which help may be given the patrolman. They will also enable him to keep in touch with the hoisting engineer or some other centralized party, who would know, by the patrolman's failure to report, that some accident had befallen him and could direct the other patrols or outside parties to render assistance.

Water Systems for Fire-fighting

Some mines have avoided the expense of carrying special fire-hydrant mains throughout the underground workings by arranging to turn water into the compressed-air lines in case of fire. There are several drawbacks to this plan. The compressed-air mains are usually smaller than is desirable for fire mains and the air-pipe joints and fittings may be too weak to stand the hydraulic pressures that may be desirable. The deprivation of compressed air may be a serious loss, for even with rescue crews carrying self-contained breathing apparatus, fresh-air bases must be established close to the fire and compressed-air portable blowers would be relied upon, as electric power is not usually available, except on the relatively few electric-haulage levels. Even though portable blower equipment is not available, if the air mains are intact, compressed air may be used to establish the circulation.

Some mines have provided, on each level, several large water tank cars, which may be moved near the fire and effective streams forced through hoses attached to the tanks by means of compressed air. This makes available only relatively small amounts of water for fire fighting and requires vigilant inspection to prevent the tank water from being depleted by drilling and other needs.

In view of the foregoing considerations a complete system of water piping was installed in the Franklin mine. This system was connected with the mine pump discharge column pipe, which discharges into the bottom of a 25,000-gal. tank at the surface, so that water will always be available even though the mine pumps are not running. As insurance against an accident in the main shaft destroying the fire-fighting water supply, a second pipe is carried up a timber shaft and manway remote from the main shaft, where it can be joined to the town water supply fire-hydrant system through the medium of a water tank.

Connection of Non-potable and Potable Water Supplies

The connection, even as an emergency measure, of a non-potable water system with a drinking water supply should be given far more serious consideration than it is generally accorded. The result of such connections may be an epidemic of typhoid fever or other water-born infectious or contagious disease. Such cross connections are recommended by fire-insurance companies and fire-protection associations. In general, there are two fire-protection reasons for the cross-connection of a town fire and drinking water supply system with a non-potable industrial-plant water system used for plant-fire protection, cooling water for condensers, boiler-feed water, and water for the ore-concentrating processes. The first may be to insure added fire protection of the plant by having a second source of supply for the sprinkler and fire-hydrant systems; the second, may be to assure the town dwellings and stores against the catastrophe of a general conflagration in case of a shortage of water supply in the town standpipe or damage to the water main feeding or leaving the standpipe, because it would be possible to supply non-potable plant water to the town system by means of the plant fire pumps. However, it would be far better to suffer property loss by fire than to run the chance of infecting the public water supply. No matter what precautions might be taken for the disinfection and sterilization of the town water pipes subsequent to the passage of non-potable water, it is impossible to prevent some of the people drinking some of the non-potable water during its use for fire fighting.

We would condemn the following methods of making cross-connections between non-potable and potable water systems, though they have been considered acceptable practice.

1. The use of one or more check valves, or gate valves, or combinations of the two in a cross-connecting pipe.

2. The use of a tee with an open outlet in the connecting pipe with a gate valve on each side of the open tee. Provision may be made to close the open outlet of the tee by a plug or blank flange, which can be applied when it is desired to use the connection, or a gate valve (normally open) may be mounted on the outlet of the tee.

3. A section of pipe may be removed between two gate valves in the connection and a section of pipe, fitted with flanges or union couplings,

of proper length to span this gap, may be kept ready to be installed when an emergency demands completion of such connection.

By all of these methods the potable supply may be polluted when a connection is made. It is true that the conditions of the relative pressures between the two sources of water supply may indicate that the flow will be only from the potable to the non-potable system, but no confidence should be placed in this. Some years later, changes may be made in the nature or operation of the plant pumping equipment that will reverse the conditions, but it is unlikely that the effect of such changes on water pollution will be given any thought.

A potable water supply, however, may be added to a non-potable water system, without danger of polluting the potable system by placing a water tank between the two systems. The inlet of the potable-water supply to this tank should be above the maximum high-water level of the tank. Although this high-water level might seemingly be controlled by the location of overflow orifices or pipes, these might become choked by floating refuse or ice, so the inlet pipe should be placed above the upper rim of the tank; even then a float valve may be placed to control the inflow of water. The interposition of an air space between the two sources of water will positively prevent the pollution of the potable-water supply by such a cross-connection.

Underground Fire-hydrant Pipe Lines

In planning the size and installation of underground fire-hydrant pipe lines, the following factors must be considered.

1. Maximum volume of water effective from a single hose-line nozzle.
2. Minimum headroom clearances and the trajectory desired of the fire streams.
3. Maximum water pressures available.
4. Number of streams it is desirable to play upon a fire at one level.
5. Greatest length of hose that should be relied on from the pipe line to the fire.
6. How much transportation of the hose, especially up confined ladderways, will be required.
7. Will advantage be taken of the light weight and accessibility of the hose to rely upon the fire patrolman promptly attacking the fire while awaiting help.

Underground fire-fighting measures are widely different from those of surface fires. The exposed areas of inflammable materials in a mine are not so great, the material is more concentrated, its height is not so great, nor is it exposed to the actions of winds. On the other hand, the hot gases are likely to come into the faces of the fire fighters and the confinement of the workings will require the nozzle men to approach closer to the fire than is customary at surface fires.

A standard surface fire stream is based on a discharge of 250 gal. of water per min., which will be passed by a 1⅛-in. nozzle at a pressure of about 45 lb. per sq. in. at the nozzle. The smallest pump approved by the National Board of Fire Underwriters must discharge 500 gal. per min. to serve two of such streams. It seems unnecessary to handle such volumes of water for the direct attack of mine fires and, in fact, it would seem unwise to have such quantities draining through filled ground, with danger of undermining the timbers supported on the same, and unnecessarily taxing the mine drainage system. At the same time, it seems desirable to have flatter and longer horizontal trajectories than are obtainable from the comparatively low pressures used for surface fire streams; flatter trajectories are easily obtained with the high static pressures obtainable from mine-pump columns, because of the depth of the mine workings below the surface. With the smaller volumes of water discharge, smaller nozzles may be used for any given range of pressures, above which the water is ejected too much in the form of a spray instead of a fairly solid stream and will not therefore carry well through the air.

Some tests were made along these lines, about ten years ago, under the direction of George S. Rice of the U. S. Bureau of Mines, and the results were published in Technical Paper No. 24, "Mine Fires." However, these tests were made under pressures much lower than we consider desirable and at which we have carried out a few experiments. Probably the chief reason why other tests were not made by the Bureau of Mines, which was coöperating with the Illinois Mining Commission and the National Fire Underwriters' Association, is its conclusion "that one man cannot readily handle such a hose (1½-in. hose with ½ to ¾ in. cone nozzle) in a narrow place, such as a mine entry or heading, when the water pressure exceeds 40 lb. per sq. in. and only with difficulty when the pressure is 50 lb. or over." The author takes exception to this statement, which was repeated last fall in the report of the Standards Committee of Mine Fire Fighting Equipment of the American Mining Congress. Our experience at Franklin mine, under underground operating conditions, is that with either a ½ or a ¾-in. nozzle and with a 1½-in. hose, one man can readily control the nozzle reaction at pressures ranging from 150 to 186 lb. per sq. in. (the latter being the highest available under the conditions of our tests). Possibly, the above adverse opinions were based upon handling say 1⅛-in. nozzles, the larger discharge of which greatly increases the nozzle reaction. We would, therefore, strongly urge a renewed study of this subect by the Bureau of Mines and feel confident that its data and opinions will be revised.

We have adopted as a normal condition, a ½-in. smooth-bore conical nozzle 12 in. long, which will pass about 90 gal. of water per min. at 150 lb. per sq. in., which we consider a proper working pressure. We have adopted a 1½-in. rubber-lined, rubber-covered, fire hose with the intent that the maximum length of hose to serve a nozzle will be 200 ft. long, as it will then pass 90 gal. per min. with a friction pressure loss of about 56 lb. per sq. in. Under these operating conditions, it would be possible to use a "siamese" connection on the end of 150 ft. of 1½-in. hose and connect two nozzles each with a 50-ft. length of hose. This would give a pressure of 90 lb. per sq. in. at each nozzle and pass 70 gal. per min. from each, or a total of 140 gal. per min. from one hydrant connection if the initial pressure be 200 lb. per sq. in. One of these nozzles might be employed to form a rosette spray to blanket back the hot gases and form a fire-insulating shield to permit the second nozzle crew more closely to approach the flames. If the head room in the drift were 6 ft., each of the above streams might be expected to strike a point in the roof about 55 ft. from the nozzle or to wet a point on the floor 100 to 110 ft. away.

One of the factors to be decided is the method for determining the hydrant pressures on the various levels and the maximum static pressures that will be permitted on the water-pipe lines. A reduction in these static pressures may be obtained: (1) By breaking bulk through the installation of water tanks on different levels; (2) by installing pressure reducing valves for the supply to each level; (3) by installing individually proportioned throttling orifices for the fire-hydrant outlets on each level. The last method was chosen as the cheapest to install and to maintain. The first method requires one or more water tanks to serve each level (depending on its length). These tanks must be located several levels above the one to be served to give sufficient water head, and each must be of sufficient depth to supply the velocity head loss at the inlet from the tank to the pipe at maximum water demand. Each tank must be equipped with float valves to control its water supply, but these valves are not easily kept in good operating condition. There is furthermore considerable duplication of overlapping vertical stand-pipe feed lines from the tanks to feed each of the different levels. Pressure-reducing valves are rather costly, are not satisfactory for controlling static pressures, and are expensive to maintain when used with the dirty mine water.

As there do not appear to be data in engineering literature covering the design of such pressure-reducing orifices, the author has plotted curves showing the pressure-drop discharge ratings of a number of sizes of orifices adapted to such an installation; see Fig. 1. The discharge of a ½-in. hose nozzle is practically the same as a ⅝-in. plate orifice. There is also included a chart showing the pressure losses from the friction of different quantities of water flowing through 1½-, 2-, and 2½-in. rubber-lined hose and through wrought-iron pipes from 1½ to 4 in.; see Fig. 2.

Another chart shows the total pressure needed at the hydrant outlet for a ½-in. nozzle combined with 100 ft. and 200 ft., respectively, of 1½-in. rubber-lined hose, and also the pressure needed when two ½-in.

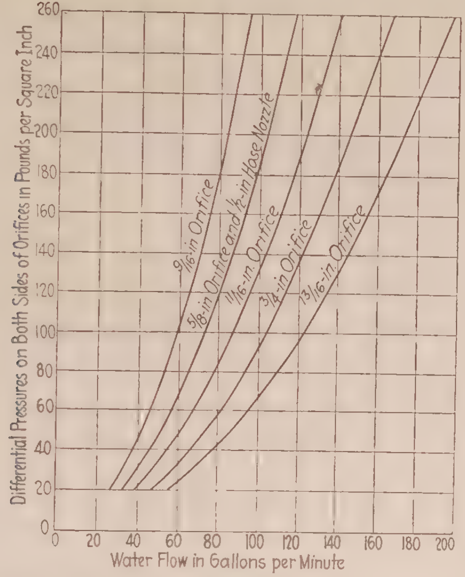


FIG. 1.—DISCHARGE THROUGH ORIFICE AT VARIOUS PRESSURE DROPS.

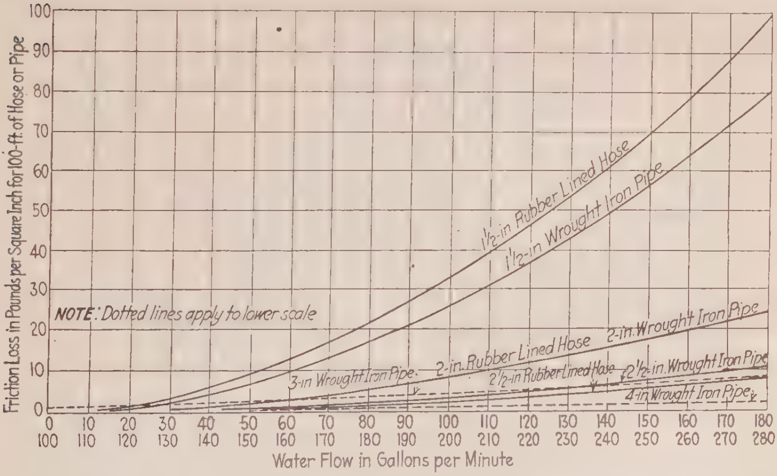


FIG. 2.—FRICTION LOSS IN DISCHARGE OF WATER THROUGH HOSE AND PIPE.

nozzles (each with 50 ft. of 1½-in. hose) are "siamese" connected to a single 1½-in. hose line 150 ft. long attached to one hydrant outlet; see Fig. 3.

A fourth chart shows the approximate maximum distances that may be reached by fire streams from ½-in. nozzles operating at various pressures, when the nozzle is elevated about 12 in. above the floor of a drift,

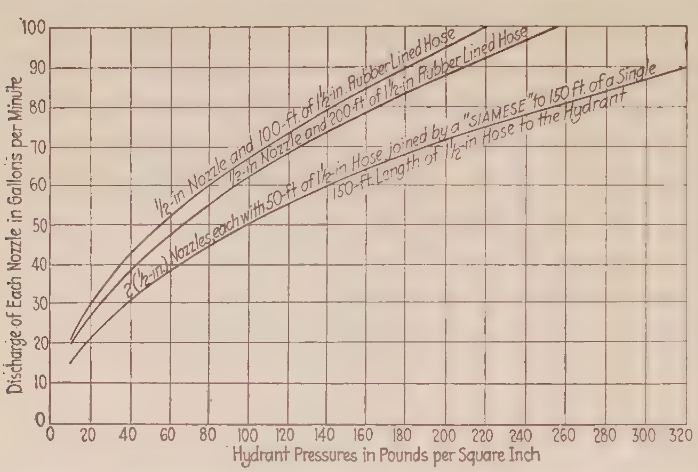


FIG. 3.—PRESSURE REQUIRED AT OUTLET SIDE OF HYDRANT ORIFICE TO ALLOW FOR HOSE AND NOZZLE FRICTION AND DISCHARGE OF STREAMS.

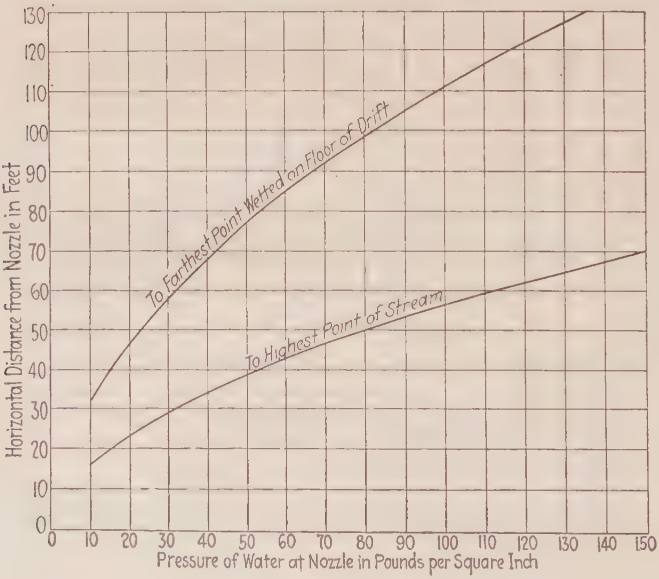
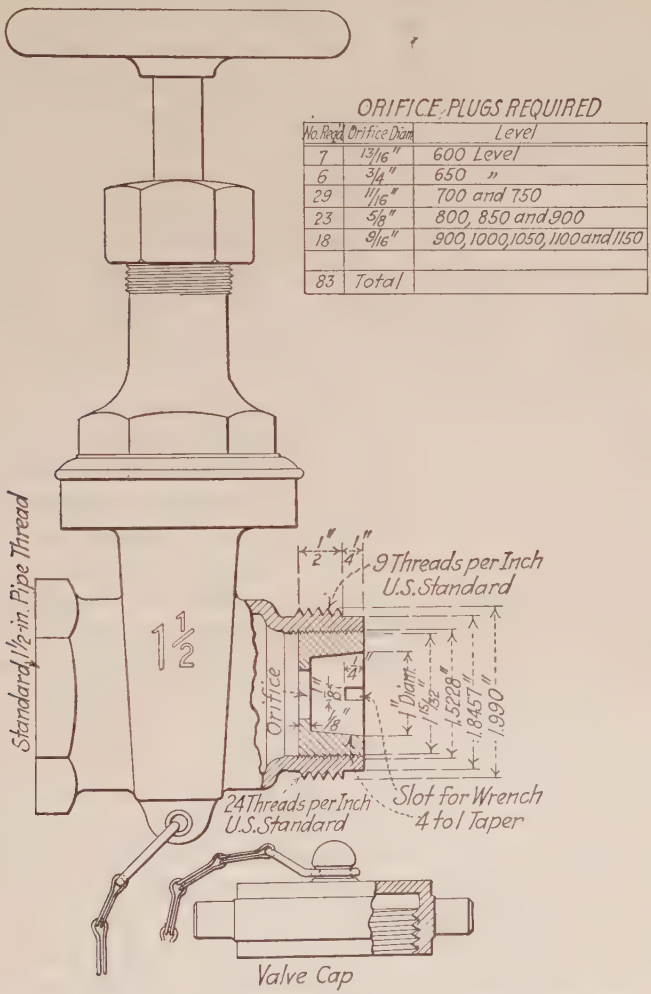


FIG. 4.—TRAJECTORY OF HOSE STREAMS FROM ½-IN. NOZZLE ELEVATED 12 IN. ABOVE GROUND, WITH MAXIMUM HEIGHT OF STREAM LIMITED TO 6 FT. AS HEAD ROOM CLEARANCE ABOVE FLOOR OF DRIFT.

which is only 6 ft. high in the clear; see Fig. 4. This chart should be considered only a rough approximation and tests should be made to revise it.

As the maximum static water pressure in the mine was about 500 lb. per sq. in. and 3- and 4-in. lap-welded, wrought-steel, standard-weight pipes are rated to stand 1000 lb. pressure per sq. in., and under tests have shown bursting pressures of 5000 to 6000 lb. per sq. in., it seemed advisable to carry full pressures on the water lines. Taking advantage of the considerable pressure drops then available for friction losses, smaller pipes could be used, in spite of the fact that over 3000 ft. of pipe was required on a level. It was not necessary, therefore, to use 4-in. pipe on levels more than 420 ft. below the surface, and some of the piping on these levels was of 3-in. size.



ORIFICE PLUGS REQUIRED

No. Req'd	Orifice Diam	Level
7	13/16"	600 Level
6	3/4"	650 "
29	11/16"	700 and 750
23	5/8"	800, 850 and 900
18	9/16"	900, 1000, 1050, 1100 and 1150
83	Total	

FIG. 5.—VALVER AND ORIFICE PLUGS FOR MINE FIRE PROTECTION.

In general, screwed joints were arranged for, with some flange joints for an easy installation of bends. Malleable-iron, standard, refining, oil fittings are used from the surface to the 400-ft. level and extra heavy malleable-iron fittings for the lower levels. These fittings assured ample strength for the mechanical strains, which may be added to the water pressures. The former are rated for 200 lb. per sq. in. water pressures and the latter for 1000 lb. per sq. in. The hose outlets are 4 by 4 by 2-in. and 3 by 3 by 2-in. tees, with bushings to 1½ in., to which are attached, by screwed nipples, standard, brass, 1½-in. hose, gate valves with male hose coupling threads on one end, and equipped with brass caps attached by chains to protect the hose coupling threads when not in use. These valves are rated at 125 lb. steam pressure and 200 lb. water pressure, but our tests showed that they would stand 800 lb. per sq. in. hydraulic pressure without deformation or leakage. The hose-coupling end was internally threaded with 24 threads per inch, as shown in Fig. 5, to permit screwing in the orifice plug used to control the outlet hydrant pressures. This threading also tested satisfactorily under an 800 lb. per sq. in. pressure without leakage. The orifice plugs were thus installed on the outlet side of these valves, to facilitate their removal for cleaning out pieces of bark or wood fiber that might lodge behind and constrict them; this being easily and quickly done after closure of the hose gate valve. As open nozzles only are permitted on the hose lines, the maximum pressures on the fire hoses will be the dynamic pressures permitted by the orifice dimensions. As there is, however, a remote possibility that kinking of the hose might place it under higher static pressure (possibly 500 lb. per sq. in.), the fire hose must stand 750 lb. per sq. in. bursting pressure for 5 min. and some of the hose is made to stand 1000 lb. per square inch.

We have been working for over 6 months with a number of hose companies on tests of specially constructed 1½-in. rubber-lined, rubber-covered, wrapped, duck-fabric fire hoses of different plies to secure great strength and durability together with maximum flexibility and portability with minimum weight, and have finally obtained hose that will meet the specifications here given.

Although it may be possible to develop a four-ply hose that will approach the performance shown by the actual bursting test described in this paper, the five-ply hose described is unusually strong for its weight and represents the best product we have tested. Its weight, uncoupled, is about 39½ lb. per 50-ft. length, and the pair of expansion ring type brass couplings add about 3 lb. more per 50 ft. length. When empty, it will bend around about an 8 in. radius arc without buckling. It was made under a guarantee to stand 800 lb. per sq. in. test pressure for 5 min. without bursting. The inside diameter is 1½ in. and the outside 2⅙ in. The five plies of fabric are cumulatively about 0.155

in. thick in a wall of the hose, so that the rubber lining and rubber cover are each about 1/16 in. thick. The rubber in the lining and cover is a high-grade compound with good density and prepared to resist penetration of fungus growths through it to the fabric. It should have a small amount of antiseptic, as creosote, incorporated in it to insure this mildew-resisting property and should be designed to give a long life, say 20 years, with relatively small depreciation.

We expect a hose suitable for 750 lb. per sq. in. hydrostatic pressure when new to be in condition to stand at least 500 lb. pressure after a life of 15 years, in the mine, since the exclusion of sunlight, the fairly uniform temperature of about 55° F., and the uniform humidity underground are excellent conditions for its storage.

The fabric used in this hose is a high-quality tire duck weighing 17½ oz. per sq. yd., made of Egyptian quality cotton fiber having a length of 1⅜ in., which is about the maximum obtainable in a staple fabric. As the Scott machine used in testing the fabric had a capacity of about 400 lb., the sample used was a strip 1 in. wide and 3 in. long, clamped in the jaws of the machine and the jaws were separated at the rate of 20 in. per min. The duck, tested in this way, shows 305 lb. as the strength of the warp and 335 lb. for the filler.

The question of the proper standard method of testing fire hose under pressure for acceptance tests does not seem to have received the attention it should. Underwriters' approved hose is commonly marked "Tested at 300 lb. pressure" and during such tests the hose is under pressure (varying from zero to 300 lb.) for a total time usually less than 20 sec. Hose that stands this test may burst if the same pressure be maintained for a longer period. The manufacturers arbitrarily assume that hoses are suitable for a working pressure of about one-fifth of their instantaneous bursting pressures. Even when tested at certain pressures for 5 or 10 min., the hose is under stress too short a time for the test to be of practical value, because in service the periods are longer. The author urges research in regard to the cumulative effect of such strains on the bursting strength of hose and a study of the lasting injury that may be done to the hose by longer testing periods under varying pressures, and suggests that, until such knowledge is available, one out of every five lengths of hose be taken at random and submitted to cumulative bursting pressures, holding each pressure stage for 5 min. and arranging for the hose to be under substantial pressures for 40 to 45 min. in a manner somewhat similar to the performance test on the 1½-in. hose to which reference has been made. As in this instance, the test can be made on a 3-ft. length, preferably cut from the end of the standard 50-ft. length, the acceptance of the entire order of hose should be contingent on a perfect showing of all the 3-ft. samples.

Bursting Tests on a Five-ply, Wrapped, Duck-fabric, Rubber-lined, Rubber-covered, 1½-in. Water Hose

Original External Diameter, $2\frac{1}{16}$ in.

Pressure, Lb. per Sq. in.	Time Maintained, Minutes	External Diameter, Inches
100	None	2⅛
200	None	2¼
300	None	$2\frac{5}{16}$
400	5	$2\frac{11}{32}$
500	5	2⅜
600	5	2⅜
700	5	$2\frac{13}{32}$
750	5	$2\frac{13}{32}$
None		$2\frac{3}{16}$
800	5	$2\frac{13}{32}$
880	5½	$2\frac{7}{16}$
970[a]	5	$2\frac{7}{16}$

[a] At this pressure the hose was bent in an arc of about 15 in. radius with no noticeable affect. On raising the pressure to 1025 lb. per sq. in., the hose burst, but not in a portion under tension during the bending test.

This hose will be carried in 50-ft. lengths, stretched out in V boxes or launders suspended from the backs of the drifts, at a slight gradient to insure natural drainage, and will have water passed through it once every 6 mo. to wash out the sulfur "bloom" and stretch the rubber so as to keep it "lively." We are led to expect, under these conditions and the darkness and humidity in the mine, a life of about 15 yr. without a depreciation in strength below a 500 lb. per sq. in. bursting pressure.

The hose valves are, in general, spaced from 150 to 250 ft. apart, except where the width of the orebody makes it advisable to place them 50 to 60 ft. apart. The intent is to make the most remote points, even in top slices 50 ft. above a level, easily accessible with 200 ft. of hose. As the levels on which the water lines and hydrants are located are about 50 ft. apart in elevation, approximately 27,000 ft. of 3-in. pipe and 4000 ft. of 4-in. pipe are required for this fire-protection water system, with about 160 hose valve outlets. Over 40,000 ft. of mine workings on the levels, without considering the length of lateral drifts and parallel workings, are thus given fire protection. Cast-iron pipe nozzles were adopted, rather than brass, because of their lower cost and because they offered no temptation for theft and sale to junk dealers. Thirty-five such nozzles and about 3200 ft. of 1½-in. hose seem sufficient to supply complete fire protection. In general, the fire lines on the levels have three or four vertically connecting pipe lines in ladderways; hence, for any fire location the water is supplied in circuit from both sides of the fire area, thus normally reducing the friction losses by splitting the flow in any single pipe line on a level and also insuring a supply, even though a fall of ground should damage a fire line at any point on a level. Enough gate valves are inserted in these mains to permit their necessary segregation under such emergency conditions. Depending on the lengths of levels, from 100 to 300 ft. of hose in 50-ft. lengths is allotted to a level, and, by transportation up or down 50 ft. of ladderways, the total quantity of hose on three levels (from300 to 800 ft. hose) can readily be made available on any one level, as ten levels each have 100 ft. of hose, five levels 200 ft. each, and four levels 300 ft. each.

The new (1922) National Fire Protection Association standard threads for 1½-in. hose couplings have been adopted. They are recommended to the mining profession, as they have also been adopted by the National Fire Protection Association, the National Board of Fire Underwriters, and the Associated Factory Mutual Fire Insurance Companies. These male coupling threads have the following specifications: 9 threads per inch with a root diameter of 1.8376 in., a pitch diameter of 1.9138 in., and an outside diameter of 1.9900 in. The threads in the female coupling have a root diameter of 2.0150 in., a pitch diameter of 1.9388 in., and an internal diameter of 1.8626 in.

The mine should be equipped with several "siamese" connections, and with transformer couplings to permit, under emergency conditions, the coupling of the 1½-in. mine hose with the 2½-in. or other sizes of fire hoses used by the town fire departments in the communities adjacent to the mine.

To abridge this paper, no detailed reference has been made to the design and installation of fire-draft control doors, which are an essential part of a mine's equipment in fighting underground fires by insuring safe working conditions in the mine shafts and other traveling roads and exits. Nor is reference made to the necessary portable mine-ventilating equipment for the establishment of fresh-air bases close to the seat of the fire; and, finally, discussion has been omitted respecting the self-contained oxygen-breathing apparatus and squads of well-trained, seasoned, mine-rescue crews so necessary for the direct attack of mine fires, because of the non-respirable atmosphere usually present during such a catastrophe.

DISCUSSION

George S. Rice.—When the Bureau of Mines Technical Paper No. 24 was prepared, I found no literature that dealt with the use of hose underground.

The Cherry mine fire, in the fall of 1909, brought up the question of fire fighting directly with hose and a number of incidents there caused the Illinois Mining Commission to try to standardize fire-fighting devices and methods. (This Commission was appointed by the State, following the Cherry mine disaster, to prepare a mine fire code. Doctor Holmes, first director of the Bureau of Mines, was a member, but as he often had to be away on other matters, I substituted for him. A most important member of the Commission was Prof. H. H. Stoek, representing the Mining Department of the University of Illinois.) This Commission formulated the first state code of specific mine-fire regulations. This code was very complete and in most respects has stood the test of time. In fact, I think that the only thing that has been made void related to an automatic fire alarm. Electric fire signals were tried but, after several instances where a short-circuit rang the bells throughout the mine causing men to stampede back to the bottom, it was decided that this signal system was in itself a danger. Telephones are now required for notification of fire, and the system must extend to the inner partings or sidings.

One of the difficulties in the use of hose which arose in the Cherry mine fire was the size and number of threads of the hose connections. The mine was not equipped with much hose and the disaster, in which 260 men lost their lives, attracted great attention. It was only about 100 miles from Chicago, so the city fire department offered its services and took down hose. It could not connect up with anything that was on the ground, however; that showed me the desirability that fire hose at mines should have universal pipe threads.

The Commission decided that there should be fire protection in shafts and shaft bottoms or landings by means of water lines and hose. It visited the Underwriters' laboratory in Chicago, which is supposed to have the last word on fire protection, and some tests were arranged by the management to enable the Commission to determine proper pressure and size of hose for mine fires. A frame was set up, of the height of an average mine entry, under which the trajectory of the stream would come and be tangent to, as it would under a mine roof or cross timber, because the critical thing found usually in mine-fire fighting with a hose is the ability to reach the upper part of the timbers. The fire, in its incipient stages, is apt to be in the timbers overhead. Measurements were then made of how far the stream was thrown, holding the hose nozzle close to the floor and pointed upward, so the stream would just graze the bottom of the frame.

Hose of different sizes was tried, also different sizes of nozzles, and the individual members of the Commission tried handling and moving the hose and nozzle when the water was turned on, and under various pressures. It was decided by the Commission that, for emergency purposes, the hose should be of a size and the maximum pressure such that it could always be handled by one man; that the time to put out a mine fire was in the first few minutes, and therefore everything should be ready quickly. With these premises, it was decided that the usual standard 2-in. hose used for surface-fire fighting was too large and 1½-in. hose was the right size.

The Commission also considered that, in the excitement at the time of a fire alarm, the hose might not be laid out in the admirable way in which the Chairman proposes to lay it out, but it would be in a box or some such arrangement. In the excitement of the moment, the full water pressure might be turned on the hose before it was straightened out. Hence the water pressure should be limited automatically in the water line or hose connection, so that one man could straighten out and handle a hose with the full pressure on.

It was finally decided, along the lines which I accepted for the paper (Technical paper No. 24, Bureau of Mines), that 40 lb. and a maximum of 50 lb. was as high as it was desirable to use. Several years later, I had similar tests conducted at the Experimental Mine, and more fully studied the stream trajectories and volumes of water discharged, up to those produced by 40 lb. per sq. in., which was as high as our facilities permitted. So in the matter of pressures we accepted the previous findings of the Illinois Commission. Nothing, so far as I know, has been done in this general matter since that time, until Mr. Tillson began his work.

Last fall, following the Argonaut mine fire, the Governor of California appointed a committee to ascertain the facts and make recommendations. Simultaneously the State Industrial Accident Commission, which has charge of inspection and has the power to make regulating measures, began a series of conferences to determine the adequacy of its existing regulations and requested the Director of the Bureau of Mines to send a special representative. B. O. Pickard, the Bureau's representative in California and adjacent states, and I presented the Bureau of Mines' views on the subject.

The American Mining Congress, then in session at Cleveland, had also been requested to consider the matter, and its Committee prepared a valuable report, under the leadership of Charles A. Mitke, which was subsequently presented by Edwin Higgins to the conference in San Francisco. Many experienced mining men who were in attendance at the conference added their views regarding the deficiencies and needs of the existing California regulations.

A committee was appointed to draft a tentative code of mine-fire orders or regulations. After publicity to these had been given, and two months for deliberation, a further conference was called, when a new committee was appointed to redraft the code, which had been acceptable in principle.

The report of this Committee, with some revisions by a reassembled conference, was submitted to the State Industrial Accident Commission on Feb. 5, 1923, and the finally revised code has been accepted and will be issued by the State Industrial Accident Commission under the head of Fire Control Safety Orders.

In California, the State Industrial Accident Commission has the power to issue orders; therefore, if certain orders are insufficient or are causing undue hardship, the Commission may revise them from time to time. Those of you who have served or assisted in the formation of proposed legislation for the different state legislatures know how difficult it is to build up good codes, under the pressure of various partisans, no matter how laudable may be their intention.

The Bureau of Mines does not render an opinion until the Director has signed the recommendation but a representative may give an informal opinion as an individual. Before going into the first conference, I set down the things which I thought, from frequent discussion with experienced bureau engineers, especially Mr. Paul, Mr. Pickard and Mr. Harrington, would represent the most important things to be considered in this matter, with such modifications as not to make too serious a burden upon the operators, under the conditions in California metal mines. I might summarize these as follows:

First, in deep mines, there should be two shafts, preferably continuous, extending from the surface to the lowest level on which stoping is done.

Second, both shafts should have hoisting equipment. Those of you who have mined out of deep mines realize that to climb 1000 or 2000 ft. to get out of a mine is no small task; it was estimated before the Commission that, in the Argonaut mine, it would have taken a strong man from 5 to 12 hr., to climb out from the 4000-ft. level, through the offsetted escape way.

Third, there should be a positive ventilating current; in other words, actuated by a fan. On the question of reversibility of the ventilation, the Bureau is in favor of the principal fan, to be placed on the surface, being of the reversible type. There is always danger at the moment of stress of doing the wrong thing, yet there are cases where men have been saved by reversal and cases where they might have been. There could not have been a better illustration of this than in the Cherry mine, which had two shafts. The fire started in the upper bed in the downcast shaft, when a car loaded with loose hay was pushed under a torch lamp dripping with oil. To switch the burning car, the men at the station propped open a door nearby, permitting the fire to sweep rapidly over into the upcast hoisting shaft, setting both shafts on fire and making escape from the mine impossible. The fan was not reversible but it might have been stopped and the heat of the fire, if the connecting door had been closed, might have reversed the air current.

Fourth, a downcast shaft and landings should be fireproofed or else constantly sprinkled. You doubtless realize the tremendous difficulty there is in applying these propositions in the case of the California mines; but we were stating our ideals for safety. In some of the deep mines, the ground is moving; therefore, it was contended that gunite, for example, would not stay on; on the other hand, it was contended that many of the mines did not have water supply enough to sprinkle continuously.

Fifth, emergency fire doors and bulkheads should be placed adjacent to the downcast shaft landings. If a fire occurs in an upper level, these will prevent the fire and smoke entering the downcast shaft and being whipped around through the mine. By closing the emergency doors the fire will be confined in the level in which it originated, at least long enough to allow the men in the lower levels to escape.

Sixth, the buildings near the shaft should be fireproofed.

Seventh, fire lines and hose, and a sufficient supply of water for a considerable period of fire fighting, say several hours at least, should be provided.

Eighth, outline maps should be posted at the top of the shaft and at various other places. These maps need not have property lines, nor anything relative to the shape of the orebody, but should be sufficiently near in proportion that there can be no mistake, on these the various exits should be marked and the fire-fighting arrangements indicated.

Ninth, fire-fighting crews should be organized and apparatus provided. We did not dwell on the apparatus question because that had been already well taken care of in the existing code, which the operators themselves were largely responsible for, under the guidance of the Commission. Organization for prevention or control of threatening disasters is everything, and the organization should exist before the accident occurs. This is especially true for a fire-fighting organization, so that the crew may be drilled and plans made in order that in an emergency each man will do his part almost automatically.

Tenth, places of refuge should be designated. This term is used in contradistinction to "refuge chambers," which conveys the idea of great expense. The idea was that there should be places of refuge, used possibly in connection with emergency or fire-doors, but the men should know there would be some place where they would first be sought. As those who have had experience following disasters know, we have searched mines (particularly coal mines) without having any knowledge of where, if there were live men, they may be found. That was the case at Cherry mine. After we got into that mine and searched one group of entries after another, turning successively the full air current into them to sweep out the gases, it was not until the very last entry was approached that twenty men who had shut themselves in six days before could be reached.

Eleventh, a telephone system, with separate lines down each shaft, should be installed. That plan is very important. It is not a great expense to have those double lines, but the cutting of connections in the shaft which has the only telephone circuit cuts off all the phones. That was the case in the Argonaut mine, where all the signals went off in one shaft.

That comprises the principal suggestions that the representative of the Bureau advanced to the first conference.

George W. Booth,* New York, N. Y.—When determining the type of hand extinguisher to use, the fact must be borne in mind that the stream from the foam-type of extinguisher will not penetrate the crevices nor a mass of material like excelsior.

Is not a 1⅛-in. nozzle, with a pressure of 40 lb. per sq. in., the largest that can be handled by one man. A few years ago some tests were made in Boston to determine the nozzle reaction with different streams; most of the tests were made with 1⅛-in. nozzles, and larger. The formula for the nozzle reaction that was developed from those tests is

$$R = 1.5d^2p$$

in which d = diameter of nozzle;
p = pressure, in pounds per square inch.

By this formula, the reaction of a ½-in. nozzle, with a pressure of 150 lb. would be about 60 lb.; or about the same as the reaction of a 1-in. nozzle with a pressure of 40 lb. These tests were carefully made, a spring balance being used to determine the reaction, and the results can be considered quite reliable.

Mr. Tillson is right in his contention that tests of hose should involve bursting pressure for a longer period than is now customary.

In regard to connecting supply sources of potable and non-potable water, I would like to quote from a letter I wrote to the *Engineering News Record* about two years ago, in reply to an editorial in that journal.

> Insurance standards require for complete reliability two independent sources of supply, and the plant managements or municipal authorities may, and often do, use one source which is unsafe or questionable from a sanitary standpoint for the reason that it is a cheaper or easier one. The engineers of the National Board of Fire Underwriters do not, and we believe other engineers should not, favor such connections. It is not possible without charge of discrimination to refuse credit for them as emergency sources.

We do not believe we can accept the moral responsibility involved in recommending that connection be made between these two sources of supply. If the local health authorities or the state health authorities are cognizant of that connection and are willing to accept the responsibility for it, we cannot refuse to give credit for that as a source of supply, but we do not believe it is our business to accept any part of the responsibility for making that connection in the first place.

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* Chief engineer, The National Board of Fire Underwriters.

George S. Rice.—Why do not the Fire Underwriters and the Fire Protective Association advance as their standard the iron-pipe thread dimensions?

George W. Booth.—The iron-pipe thread is too fine to make a good quick connection. These connections are emergency connections, and must be made in a hurry, so they should not be hindered by too fine a thread. For the 2½-in. size there is a national standard thread; that is 7½ threads to the inch, and 3⁄16-in. outside diameter.

George S. Rice.—In Upper Silesia, the fire-fighting crews had an attachment, or a clamp, that could be attached to a pipe of any size. The clamp was put on the underside.

George W. Booth.—That idea of adapters is used pretty generally in lieu of the general standard; 2½-in. hose is the most commonly used in fire-department practice. The standard coupling has been adopted by the International Association of Fire Engineers and various waterworks associations and fire-protective associations. It was adopted about 20 years ago. I doubt if there are any possibilities of its being changed, especially to a finer thread; most people say that the thread is too fine now and that a six-thread would be better for ordinary practice.

There is on the market a set of tools for adapting off-size couplings to the standard. With it, any seven- or eight-thread coupling that varies in diameter by as much as 3⁄32 in. can be transformed into a standard coupling. It is estimated that about 15 per cent. of the couplings are this national standard. Most of the remainder can be made to conform to that standard by the use of these tools. That is being done in about fourteen states; about five hundred cities and towns during the last few years, through the use of these tools, have standardized their apparatus so that one town can help another in cases of emergency.

H. A. Burnham,* Boston, Mass. (written discussion).—The note of caution sounded in reference to the cross-connections between potable and non-potable water supplies is opportune. In the development of private fire protection by means of automatic sprinklers, fire pumps, and tanks, the elements of possible danger from such cross-connections have been the subject of careful coöperative study, so that with the advance in recent years in the science of sanitary engineering, as applied to water works, there has been evolved a thoroughly practical and highly efficient safeguard for use wherever the conditions of the water supplies and the fire protection demands make such cross connections necessary or advisable. A full description of this safeguard has been printed.[1]

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* Engineer and special inspector, Associated Factory Mutual Fire Insurance Co.

Briefly, this equipment consists of a pair of check valves specially designed to avoid the faults that the early installations disclosed, and arranged for complete accessibility so that they may have as good supervision, maintenance, and inspection as any other part of a water-works system. At present, about 700 have been installed in about 170 towns and cities in the United States and Canada, and their record is most satisfactory. In view of this special development, universal prohibition of automatic sprinkler connections to public supplies where there is also a non-potable secondary supply would be a deterent to the legitimate extension of automatic sprinkler protection. The second and third methods of making cross-connections are, of course, dangerous under any conditions as they provide intentionally for full flow and complete filling of the public mains with non-potable water through manually operated valves.

R. DAWSON HALL, New York, N. Y.—Carbon tetrachloride may be used advantageously not only in extinguishing electric arcs but also in fighting fire in timbers adjacent to charged wires, because it is the only kind of stream along which the current will not flow back and injure the man who is directing it.

GEORGE S. RICE.—When the use of this extinguisher was first suggested to the Bureau of Mines, we were warned that the gases therefrom, if breathed, might be toxic. Accordingly, tests were made by the Bureau of Mines, and it was found that in the confinement of a mine room and absence of positive ventilation, dangerously toxic gases were produced.

It has been quite generally agreed that carbon tetrachloride should not be used in a mine unless there is a good current of air; even in the case of electric stations undeground some provision for ventilation should be made if it is to be used, or else it should be thrown in through a small opening, so as not to harm the person using it.

B. F. TILLSON.—What is the effect of these portable extinguishers on a blaze overhead?

GEORGE S. RICE.—We did not consider that, but it seemed to work very well. In the case of electricity, you can break a short circuiting arc without danger to the operator as the gas and liquid are non-conductors, which is the advantage for that particular purpose. The general opinion is that it should be used sparingly.

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[1] Development of the F. M. Double Check Valves as a Safeguard for Public Water Supplies." *Jnl.* Am. Water Works Assn. (May, 1921).

R. DAWSON HALL.—Carbon tetrachloride can best be handled where there is a good current of air to protect the man who is using the projector. Better results may be obtained where there is no current in the place into which the gun is fired, as was the case in Kansas where a fire was extinguished by this method. The man using the gun, however, should be protected by an air current. There should be no men in the workings ventilated by the air split which has passed the place where the tetrachloride gun is being used.

GEORGE S. RICE.—There are two main gases produced under some conditions by the use of carbon tetrachloride: phosgene and chlorine; carbon monoxide may also be produced. It is possible to provide a very simple protection for the phosgene and chlorine (that is, like army masks for the men to wear while exposed) but it requires a special mask or self contained breathing apparatus for carbon monoxide. If the ventilation is poor, even if you take care of the gases given off by the extinguishing fluid you are also liable to get carbon monoxide from the fire itself through insufficient combustion.

H. G. DAVIS, Kingston, Pa.—The mines in the Wyoming valley of Pennsylvania were well equipped with telephones so that a general alarm can be sent to every telephone station in case of necessity and the men called to places of safety.

The collieries in that section are not all connected to the water supply of any town or borough, but every mine office has a connection that will fit the fire hose of the borough or town.

In the case of a mine in Nanticoke that was 2700 ft. deep and 1465 ft. to the cross vein, fire lines were laid down the shaft but we never measured the pressure; we regulate the pressure by the flow. It is necessary to have a good pressure, a small hose, and a nozzle 3 or 4 ft. long with a ¾-in. orifice when fighting fires on the gangway, roads, and crossways; the timber fire situation is different and requires greater force.

Loading Ore Underground with Scrapers at the Utah-Apex Mine*

By S. P. Holt, Bingham Canyon, Utah

(Canadian Meeting, August, 1923)

The chief use of scrapers at this mine has been on lead ore of milling grade, clean and dry, in stulled stopes, 50–100 ft. long, 5–12 ft. high, and pitching 20–30°. Scrapers have also been used to fill these stopes with waste.

The first scraper used (Fig. 1) is modified from a Cleveland-Cliffs design, being only 30 in. wide. It is light, weighing but 95 lb., handles well, and digs itself in; but it spills part of its load on a long haul. It could be improved by lengthening the side plates, but because of the double bent angle irons, it is difficult to make at mine shops and has been discarded. Fig. 2 shows where the light angle irons have failed, spoiling

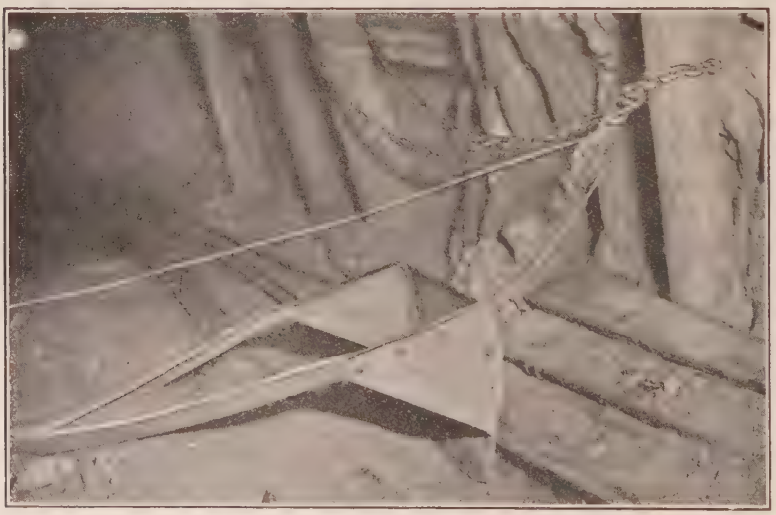

Fig. 1.—Modified Cleveland-Cliffs scraper first used.

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* Published by permission of V. S. Rood, general manager, Utah-Apex Mining Co.

the angle of the digging edge. The next scraper (Fig. 3) was designed at the mine. It gave good service but was too small, its capacity being 5–5½ cu. ft., and too light, weighing 75 lb. It was also difficult to make at the mine shops and so was discarded. The present standard type (Fig. 4) is a modified Quincy, narrower (30 in.) and a little deeper than usual. This scraper will carry 8 to 9 cu. ft., digs down, and holds its load well. All joints are bolted, for dismounting to take into a narrow place or up a raise. The original design called for a piece of 30-lb. rail on the back of the scraper. This was omitted to save weight, but will be put on the next scrapers built, to serve as a step or seat for the operator. The next

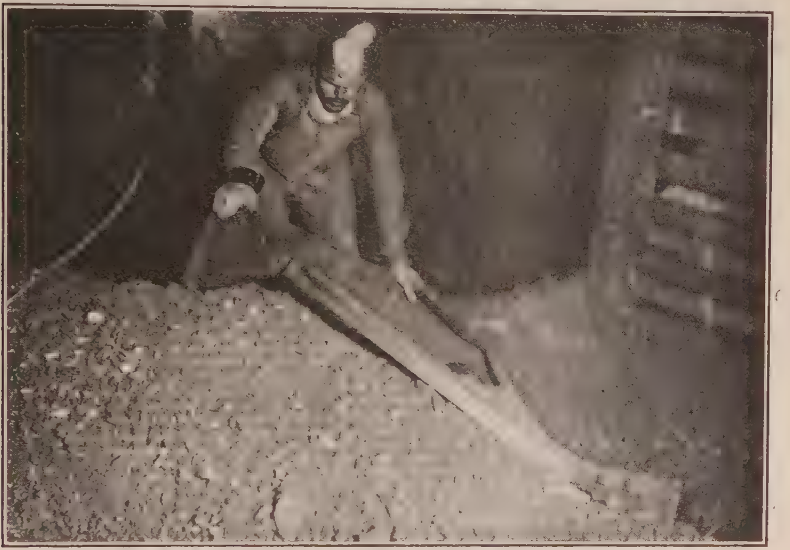

FIG. 2.—FAILURE OF LIGHT ANGLE IRONS HAS SPOILED ANGLE OF DIGGING EDGE.

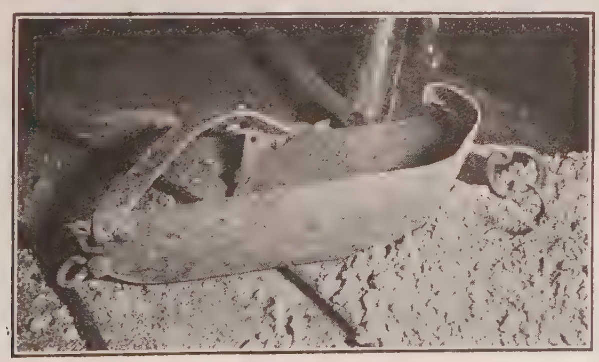

FIG. 3.—SCRAPER DESIGNED AT MINE.

scrapers of this type will be 36 in. wide. The narrow width was selected because of lack of power in the available hoists; the use of more powerful hoists permits the adoption of even wider scrapers.

FIG. 4.—MODIFIED QUINCY SCRAPER.

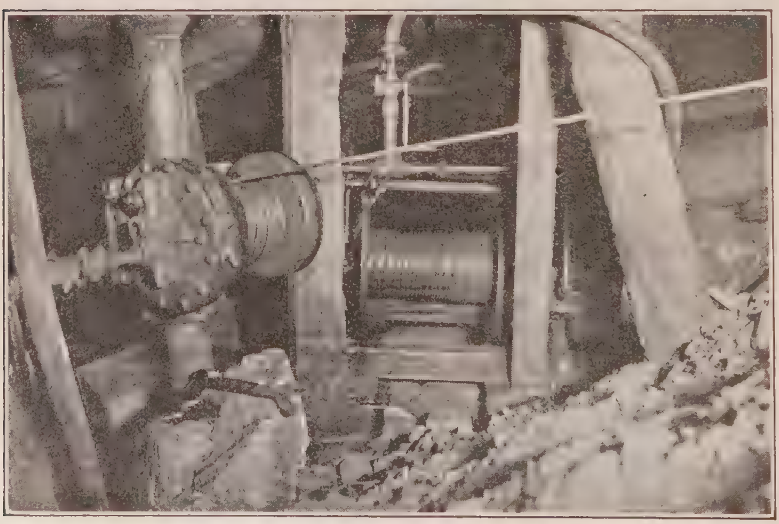

FIG. 5.—INSTALLATION OF ANACONDA AND LITTLE TUGGER HOISTS.

The first scraper hoist used was a 1-H Little Tugger, fitted at the mine with a drum divided in the middle for two ropes. As the two sections of the drum could not be operated separately, it was necessary to keep slack in the tail rope, which slack had to be rewound on every trip, and sometimes tangled. Without readjusting the length of the ropes, hauling could be done only from one distance. The rope capacity of the divided drum was small, and the hoist was only 2½ hp. The next installation, consisting of two Tuggers mounted on a drill column, overcame all disadvantages of the first hoist, except lack of power for heavy duty. Then a small Anaconda-type hoist was tried for pulling, with a Little Tugger for returning the empty scraper (Fig. 5). This arrangement was fairly successful, but required complicated pipework, and was inconvenient to install or move. The rope speed of the Anaconda, pulling down a loaded scraper on a 25° slope, at 70-lb. air pressure, was 80 to 90 ft. per min., with the hoist throttled, while the Tugger, with open throttle, returned the empty scraper at a speed of 75 to 80 ft. per min. A double-drum Sullivan hoist has been used successfully to scrape waste for filling stopes. Fig. 6 shows an unusually heavy and rigid blocking of this hoist, which has been very satisfactory.

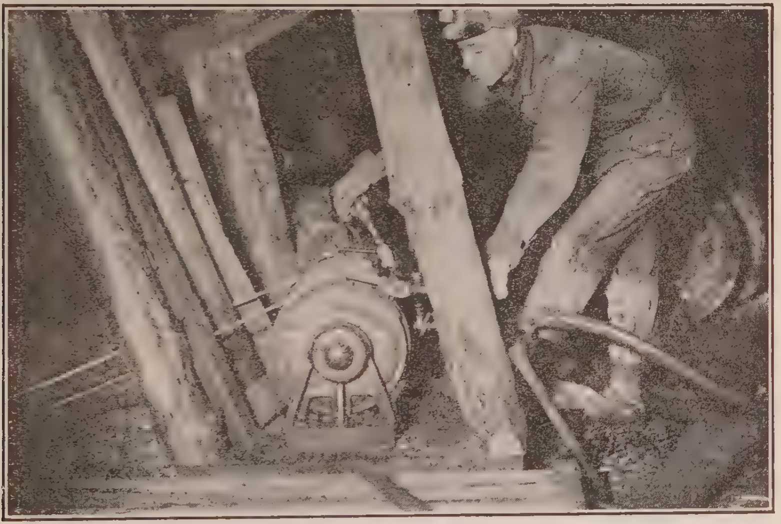

FIG. 6.—DOUBLE-DRUM SULLIVAN HOIST.

The latest type of scraper hoist used at the mine was designed by Superintendent Norden. It is an Anaconda timber hoist, with the gear and pinion moved to the center of the shaft, and two loose drums with cone clutches instead of one. It is 12 hp., weighs 1560 lb., has a rope speed in excess of 100 ft. per min. at 70-lb. air pressure, for both pulling and return of scraper. The rope speed seems to have an upper limit between 90 and 100 ft. per min. because of jerking and bouncing of the scraper at too high a speed. This hoist is simple and rugged, easy to repair, and is operated satisfactorily by common labor, due in large part to the friction clutches, which make it less liable to accident. Figs. 7 and 8 show a rigid, semipermanent mounting; a turntable mounting is now being tried, and a truck mounting is being considered. The weight might seem great for a scraper hoist, but so far has not been found to be so, and the hoist has been entirely satisfactory.

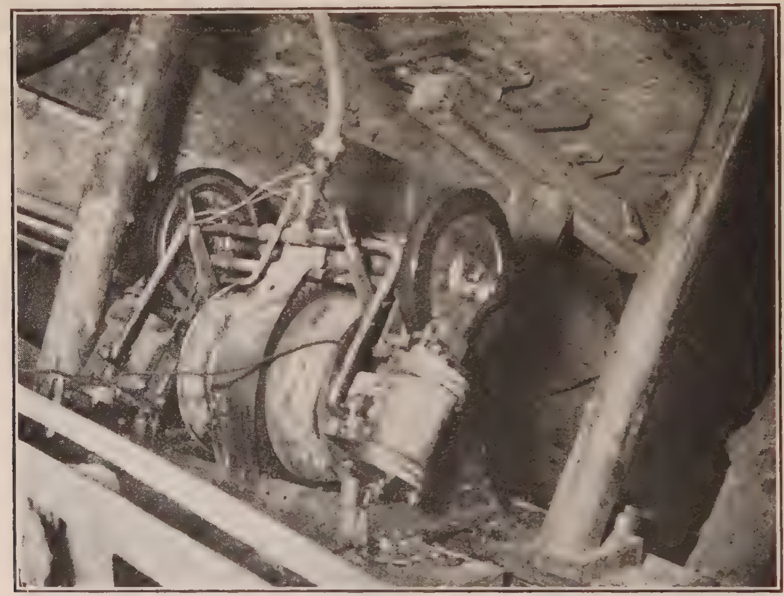

Fig. 7.—Latest type of scraper hoist.

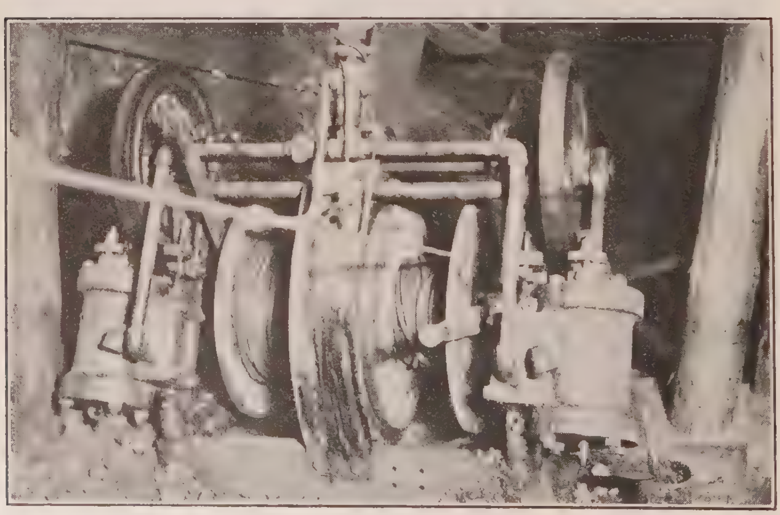

Fig. 8.

Scrapers are installed in the stopes as regularly mined, requiring no special raises or drifts, and no special timbering except that the chutes differ slightly from the standard type. Fig. 9 shows the fan-shaped chute top used, and Fig. 10 the bottom of the same chute, with scraper delivering its load directly into a car. A deeper chute, to hold 6 or 8 carloads, gives greater efficiency, and is used in other stopes.

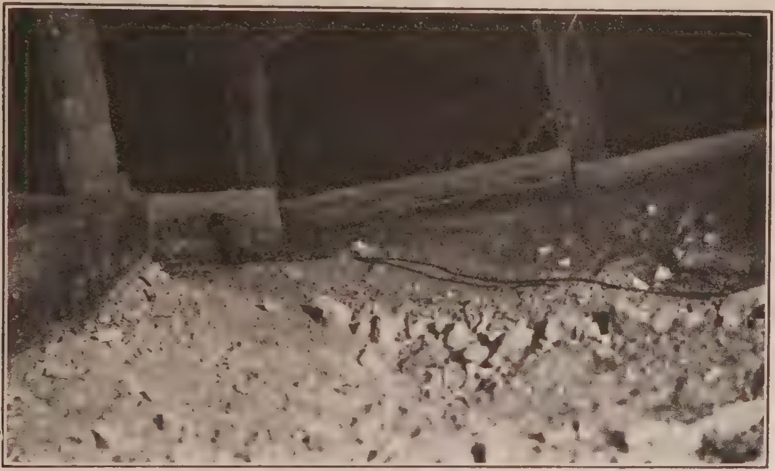

Fig. 9.—Fan-shaped chute top.

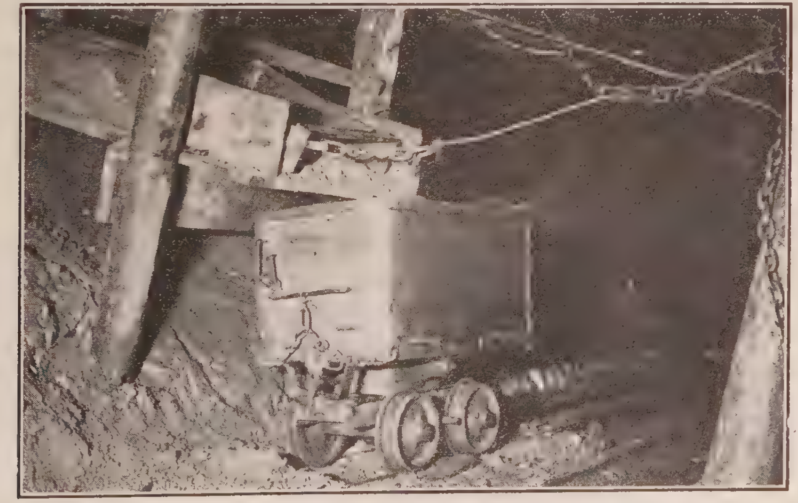

Fig. 10.—Bottom of chute.

Fig. 1 shows the chain rigging for tailrope snatchblock, where it is desired to pick up loads from varying positions, here waste dumped from a string of cars along a track. The same rigging is used in the top of a wide stope, the chain being stretched between two stulls or two drill columns. Drill columns are preferred to stulls for the double or the single tailrope rigging, as they can be more quickly set or moved.

Fig. 10 shows the snatch-block rigging at a chute, the block being attached to a plug in the roof. This is satisfactory at a chute, but dangerous in the top of a stope, where the roof may be loosened by blasting. Much trouble has been had with blocks. The usual cast pulley with light frame is cut quickly by the rope. At present, pulleys turned from steel shafting, bushed with brass, and in close-fitting casings of heavy steel plate, are being used with good results. Experiments are about to be made with pulleys of manganese steel.

All scrapers are operated by two men, both paid mucker's wages. Sometimes they take turns at the heavier work of handling the scraper, but one of a pair usually proves more adept at operating the hoist, and the other is content to let him do it. Where the chute is flat, the hoist operator trams each car when filled, but where a large chute is used it is filled, then tapped and the cars trammed by both men, which gives higher tonnage. Often the scraper operator rides down to the chute on the filled scraper, thereby obtaining a slightly better loading, but he has to walk back up the stope, and nearly the same tonnage is secured from the faster working of a scraper traveling alone with a smaller load. Higher traveling speeds could be secured, with consequent greater tonnage, if some care were used to set the supporting stulls in the stope so that the scraper would not bump them, losing its load or being stopped. This matter is receiving attention in the laying out of new stopes.

No conclusive figures can be given as to costs and saving by scraper operation. A definite saving of 10 per cent. is known, for scraper stopes are contracted at that saving over stopes where the ore is shoveled. The saving due to the fact that scrapers will allow a stope to be worked lower, and farther from a sublevel, with fewer tracks and chutes, cannot well be estimated.

After a period of experimenting with different scrapers and hoists, this company is standardizing its scraper equipment, with a tendency to use larger scrapers and faster and more powerful hoists, large chutes, and laying out stopes especially for convenient and rapid operation of scrapers. The operation of scrapers by unskilled labor has proved satisfactory, and costs are lower than for hand shoveling. Better cost figures are hoped for from the use of the heavier equipment.

Emergency Power for Mines

By Graham Bright, East Pittsburgh, Pa.

(Canadian Meeting, August, 1923)

Before the arrival of central-station power, all coal and metal mines generated their own power and, in many cases, these isolated power plants gave a fair continuity of service. In coal mines that produce considerable gas, it is essential that power be available at all times for operating the ventilating fan and the hoist, if the mine is of the shaft type. In the case of an isolated power plant, if trouble should develop with the fan the men can be hoisted quickly when the mine is of the shaft type. On the other hand, if trouble develops with the hoist, the fan can be kept in operation, and there will be no necessity of haste in getting the men from the mine.

Now that central-station power is being used so extensively for mine service, the continuity of service with gaseous mines is very important. Inherently, the central station is much more reliable than any isolated plant, but trouble may occur on the transmission lines which may prevent the transmission of power for a considerable time.

Some mines have kept their steam equipment intact and ready for operation in case the purchased power should fail. This is expensive procedure, for the boiler plant must be fired at all times so that the emergency power will be quickly available.

In a number of instances, gasoline-engine driven generators have been installed to furnish emergency power for the service hoist. As most service hoists are of the alternating-current type, the generator of the gasoline-engine driven set is usually of the a.c. type and is arranged to drive the service hoist motor at full speed, or an extra motor that can be thrown into gear on either the main or service hoist and the hoist operated at a greatly reduced speed. This arrangement is used because it is difficult and expensive to obtain and install large gasoline-engine driven units.

The author has recently developed a plan by which the main hoist and the fan can both be operated at reduced speeds, in case of loss of power from a central-station power system. In this plan a gasoline-engine driven generator operates at a reduced frequency, if alternating current, and a reduced voltage if on a direct-current system. With alternating current, the engine speed and generator are arranged to give a frequency of approximately one-half the normal frequency, but this can be varied to meet the particular conditions at any mine. In case the generator operates at 30 cycles, the voltage will be 1100 if the hoist and fan are operating on 2200 volts, 60 cycles. The motors would then operate at one-half speed with the same control and would give a fair power factor and efficiency, for the voltage has been reduced in the same ratio as the frequency. In operating the main hoist, in many cases, the load of men will be about one-half of the load of coal or ore. As, in addition, the speed is reduced one-half the power required will be about one-fourth of that necessary when hoisting coal or ore.

When the fan is operating at one-half speed, it will supply sufficient air to keep the mine clear for a short time and will require from one-fifth to one-sixth of the power necessary to operate at full speed. By adjusting the engine speed and voltage, the fan speed can be increased to 60 or 70 per cent. of the full speed, if this is necessary.

Let us assume that the main hoist will require a maximum of 600 hp. and the fan 150 hp. during normal operation. Assuming that the load of men will weigh one-half as much as the load of coal or ore, the power required for the hoist at one-half speed will be approximately 150 hp. and the power for the fan 30 hp., making a total of 180 hp. This power could be readily supplied by the gasoline-engine driven unit having an engine of approximately 250 hp. and a generator of about 150 kva. capacity. Should it be desired to operate the service hoist, a reduction of about 25 per cent. would possibly be more desirable than 50 per cent., in which case both fan and hoist would be operated at 75 per cent. of full speed.

The operation of this system would be very simple, in that no change would be required in the control system of either the hoist or the fan motor. In case of failure of power, the gasoline-engine driven set can be started by means of a push button; and when a double-throw switch is thrown, power at reduced frequency is available for either the fan or the hoist. If the fan has automatic control, the fan may be placed in operation, within 30 to 60 sec. after failure of power.

An outfit similar to that described will shortly be installed at one of the mines of the Y. & O. Coal Co., near Pittsburgh. There are two mines fairly close together, each equipped with a 400-hp., 2200-volt, three-phase, 60-cycle hoist motor and a 150-hp., 2200-volt, three-phase, 60-cycle fan motor. Fig. 1 shows the normal cycle when hoisting both coal and slate. Fig. 2 shows the hoisting cycle when handling men and using power from the gasoline-engine driven generator at 30 cycles. This cycle indicates that the maximum power required from the genera-

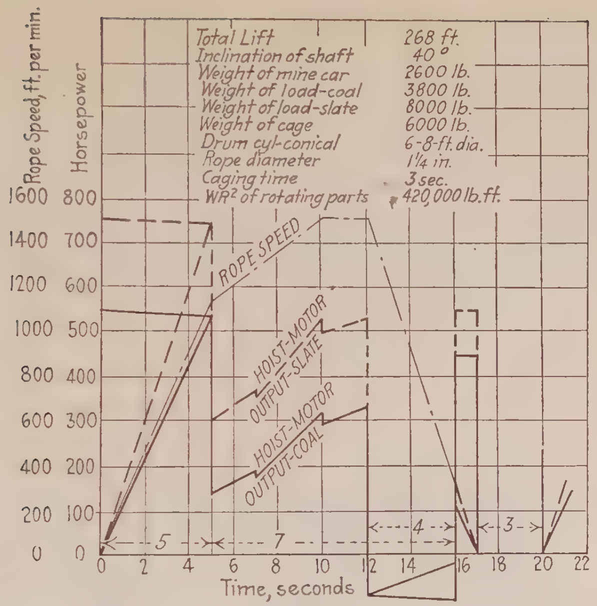

FIG. 1.—APPROXIMATE HOIST CYCLE OF Y. & O. COAL CO. MANIFOLD NO. 2, 60-CYCLE POWER.

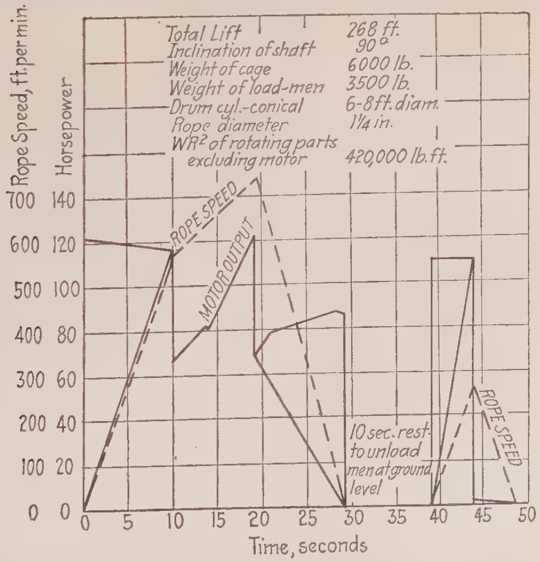

FIG. 2.—APPROXIMATE LOAD CYCLE OF Y. & O. COAL CO. MANIFOLD NO. 2, HOISTING MEN, 30-CYCLE POWER.

tor for hoisting will be about 140 hp. Both hoists could be operated from an engine-driven set of sufficient capacity to operate one hoist if it is arranged that both hoists are not operated at the same time. This could be taken care of by a signal system, which would permit one hoist being operated while the other hoist was loading and unloading. As each fan would require about 25 hp. at one-half speed, one gasoline-engine driven generator could supply power for both fans and both hoists.

The capacity of the set being installed is 250 hp. for the gasoline engine and 175-kva., 1100-volt, three-phase, 30-cycle, generator operating at 900 r.p.m. This generator will be equipped with a voltage regulator, which is essential when using an alternating-current generator with this type of drive. Fig. 3 illustrates a gasoline-engine driven generator of the direct-current type. The speed of 900 r.p.m. is a little low for a high-grade gasoline engine, but was selected, largely, because a two-pole generator was not available. With a two-pole generator, the speed could be 1800 r.p.m. This speed may be somewhat high but, if it could be obtained, would greatly reduce the cost of the gasoline engine. The two-pole generator, however, does not lend itself to increase in speed over 25 cycles, which may in many cases be desirable. With a speed of 900 r.p.m. the gasoline engine is capable of operating up to 1200 r.p.m. if conditions at the mine warrant this increase in speed and frequency.

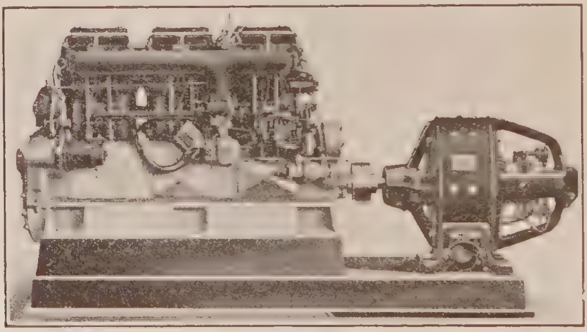

FIG. 3.

If direct current is used at the mine with field control on the hoist motor, a higher-speed engine can be selected, which will reduce the cost and weight of the gasoline engine. This is important, as the engine is much more expensive than the generator. By means of field control, heavy torques can be obtained for starting with little power required from the engine itself. The engine can be worked up to its full capacity by selecting the proper voltage for the generator. If necessary, the generator can be equipped with a differential compound field, which will automatically prevent overloading the engine. The control of this equipment would be simple and would not require any change in the control equipment of the main hoist or fan.

Using a gasoline-engine driven generator at reduced frequency on reduced voltage is much less expensive than retaining the old steam equipment and keeping it ready for emergency use at all times, or plans using gasoline-engine driven generators at full frequency and full voltage. By selecting a high-grade engine and generator, the equipment should be ready for instant service and should satisfactorily take care of the supply of emergency power in case of failure of power from an outside source.

DISCUSSION

Howard N. Eavenson, Pittsburgh, Pa. (written discussion).—One of the principal objections to the use of purchased power by shaft mines has been, and still is, the cost of installing an emergency power supply to enable the fan and man hoist to be operated in case of a shutdown to the central station. The design and operation of these large plants has now reached a stage where the likelihood of a shutdown of any customer for any appreciable length of time (more than 10 or 15 min.) is almost entirely confined to transmission-line trouble and even such occurrences are rare.

For all shaft plants, particularly gaseous ones, a shutdown of any length, however short, involving the stoppage of the fan, is a serious matter and even where the danger of an explosion is not involved, the mental hazard to the employees is serious and involves the likelihood of loss of the day's run, and, if often repeated, the trouble of keeping men at the plant.

The expense of keeping a steam plant ready to start at a minute's notice is practically as much, at a coal plant, as keeping it in operation, and a gas, or oil, engine driven plant is the only satisfactory solution of the problem. Such an arrangement as the author has proposed will result in furnishing all of the power needed during short stoppages, for maintaining ventilation and removing men from the mine, if advisable, at an expense that can easily be borne by even small mines, and if the unit operates satisfactorily, should furnish the best method of auxiliary power supply that has yet been proposed.

Some Problems in Ground Movement and Subsidence

By George S. Rice,* Washington, D. C.

(New York Meeting, February, 1923)

Those who for the first time see, at a mine, a great hole caused by subsidence; or, going underground, see an extensive fall of roof or hanging wall are apt to regard such an occurrence as an accident and not a normal condition of exhaustive mining. Moreover, the public is inclined to believe that a cave or subsidence of the surface is the result of carelessness in mining or the rapacity of the mine operators in "robbing pillars," a term too loosely used in descriptions of coal mining. This belief is fostered by the common-law requirement that an owner of the surface or of surface improvements is entitled to full support of the surface. Court decisions have upheld this law even when that owner has sold or leased the underlying mineral, unless a clause in the sale or lease contract exempts the mining company from paying the owner of the surface for damages that may result from mining. Another reason for this belief is the earlier practice (which is still extensively followed in this country) of leaving all coal pillars unmined, except as they have been thinned by so-called robbing; by not extracting the pillars, the surface may be supported by them for years, perhaps for generations.

Inasmuch as the practice of leaving permanently in the mines from one-third to one-half of the mineral is wrong, with respect to conservation of a national resource, it behooves the mining engineers to assemble data and disseminate information on the subject, pointing out that subsidence must follow complete extraction, unless expensive methods of back-filling are employed, such as are used in Germany and France. These methods inevitably increase the cost of mining and the price of the mineral produced. It is popular to speak of "waste in mining" without a realization that much so-called wasting is unavoidable unless subsidence is accepted and provision made therefor where the surface is valuable, or else the only present known alternative of hydraulically placing sand or crushed stone filling in the excavations, at an increased cost of the mineral product to the consumer.

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* Chief Mining Engineer, U. S. Bureau of Mines, and Sec'y. of Committee.

The term "waste in mining" was first applied to attract attention to avoidable waste, but the term has been over-used. The writer has urged, in technical papers of the U. S. Bureau of Mines and, recently, in data assembled for the U. S. Coal Commission, that instead of "waste in mining" the term "losses in mining, avoidable and unavoidable," be used. It seems advisable to avoid such terms as "pillar robbing," and to use instead the terms "pillar extraction," or "withdrawal," also in other ways to clarify matters in published statements on the subject so that the essential relation of mining to ground movement and subsidence may be generally understood by the owners of valuable minerals, by the owners of surface underlain by mineral deposits, and by the courts.

In certain damage-from-subsidence cases, the testimony given by various prominent mining engineers and economic geologists has differed widely, not only with respect to the description of the causes and the details of a particular subsidence, but as regards the theories on the general subject. The result has been, in certain instances, that the decisions rendered, though probably legally correct from the evidence introduced, have been unfortunate in effect.

In view of such diversity of opinion among engineers and geologists of wide reputation, the lack of a common understanding by practical mining men is not surprising. The writer has repeatedly heard coal-mine managers assert that where a coal bed was mined at great depth there would be no subsidence of the surface, and make such assertions without any qualification as to the percentage of bed extracted, the method of mining or other conditions. One of the writer's friends, the owner of a mine working a coal bed by the longwall method, held such a belief and when selling the surface made no provision in the contract for the inevitable ground movement from mining. The subsidence was small and may or may not have caused the damage to the overlying property he had sold, and on which valuable improvements were placed, but the burden of proof was put upon him; he lost, and was compelled to pay damages and costs that crippled him financially. Ridiculous, as well as just, claims for damages are sometimes brought. A Belgian mine operator told the writer of a claim that the alleged earth movement had interfered with the setting of the hens on the farm above the mine.

In this country, the number of leases and purchases of mineral rights that have been completed, without any provision for exemption from surface damage, is so large as to be surprising. These, in time, will lead to the assessment of heavy costs on the mine operator or to a great wastage of the nation's coal reserves from the unwise leaving in of large pillars to support the surface. In certain coal fields, as in central and southern Illinois where the underclay is soft, this means that 50 per cent. of the coal must be left, unless back-filling methods are employed; and such methods entail an additional mining cost that probably will much exceed the cost of repairing the surface, except where city improvements are involved. In many cases no serious or permanent damage would be done by systematic and thorough extraction of the coal, starting under the lowest part of the surface and carrying the line of extraction toward the higher ground. Hydraulic sand filling is probably not practicable in workings so level as those of Illinois, also with an underclay of a kind that the water used might soften.

Coal-mine Subsidence Localities

When the average citizen of this country speaks of caves or subsidence from mining, he apparently thinks that such troubles occur only in the anthracite district of Pennsylvania and in the city of Scranton in particular. This belief, perhaps, is quite natural, as the population of that district is large and prosperous, the total thickness of the many coal beds in intensive mining is great, and above the mines are large cities. The location of these cities is unfortunate, for many of them could just as well have been located outside the narrow basins had the inevitable consequences of mining been fully realized several generations ago. The conditions in the anthracite region have led to greater legal complications from mine-damage suits than in any other mining district of this country, and probably in the world.

Damage to valuable surface lands from subsidence caused by coal mining has also occurred in the Pittsburgh district of Pennsylvania and in the Illinois-Indiana field. In the latter, not only have town and city improvements been damaged but farms on the more or less flat-lying prairies, where irregular subsidence has ruined tile drains, thus tending to make some of the richest farming lands in the country worthless swampy places.

In mountain and hill districts, as in the Appalachians and Rocky Mountains, surface caves from mining are frequently observed, but the monetary damage is small or negligible. This refers to underground mining, as the damage from stripping methods is obvious and is taken into account before operations are begun.

Except in the anthracite district of Pennsylvania, the total vertical subsidence of the surface, where subsidence occurs, is usually small, except from workings at shallow depth; it depends both on the thickness of the coal beds and the method of mining, and usually ranges from one-half to three-fourths of the thickness of the coal excavated.

Metal-mine Subsidence Cases

Monetary damages from surface subsidence or caving by underground metal mining are usually small, as compared with coal mining. Generally, metal mines are not in rich farming districts and a mining town can usually be placed away from the mine. Sometimes shafts and mine structures are seriously damaged. In mining steeply dipping veins, the amount of the vertical displacement is usually small. Within the mine, air blasts and rock movements may cause serious damage; and in mining below bodies of water a cave, as at the Treadwell mine, Alaska, may be disastrous to the mine.

Where the veins are numerous and mineralization as extensive as at Butte, where it is estimated by Daniel Harrington in an unpublished report that a hundred million tons of ore and waste have been extracted from an area of 4000 acres, making an excavation which if averaged would be 18 ft. thick, there is necessarily an extensive though slow settlement. According to W. H. Weed,[1] the Butte post-office up to 1912 had sunk an average of 10⅝ in. Whether there was continued settlement is not reported, but in 1922 Harrington comments that so even has been the settlement of this building that no appreciable cracking had occurred. However, in other places at Butte, subsidence has been greater and more abrupt and some of the shafts have settled. The extensive slime filling of old stopes now being carried on to extinguish a slow-burning fire in the stopes, and to permit recovery of the pillars, will do much to prevent subsidence over the filled area.

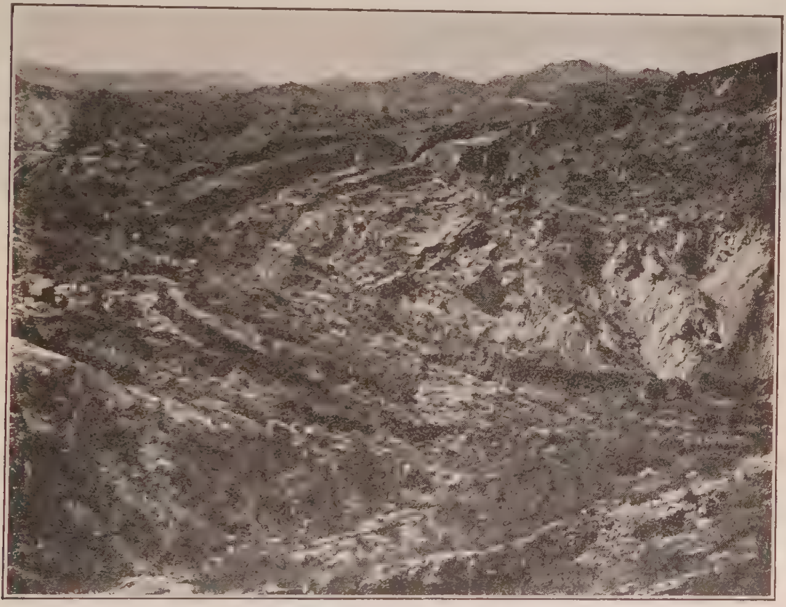

FIG. 1.—CENTRAL PART OF MIAMI CAVED GROUND; RECENT SLIPS IN CENTRAL BACKGROUND SHOWN BY WHITE BANDS IN RAVINE-LIKE DEPRESSION.

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[1] U. S. Geol. Survey *Prof. Paper* No. 74 (1912) 50.

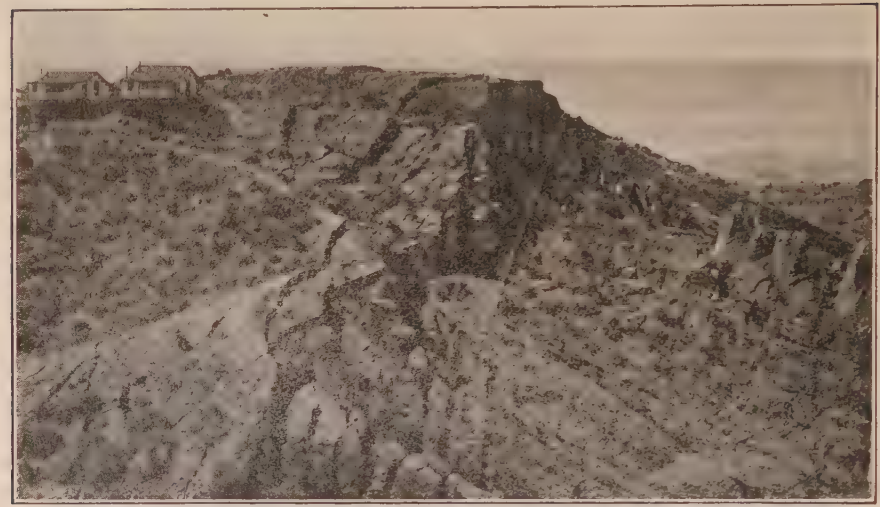

Fig. 2.—Miami caved ground showing verticality of breaks on reaching surface in conglomerate; also showing cracks in advance of subsidence.

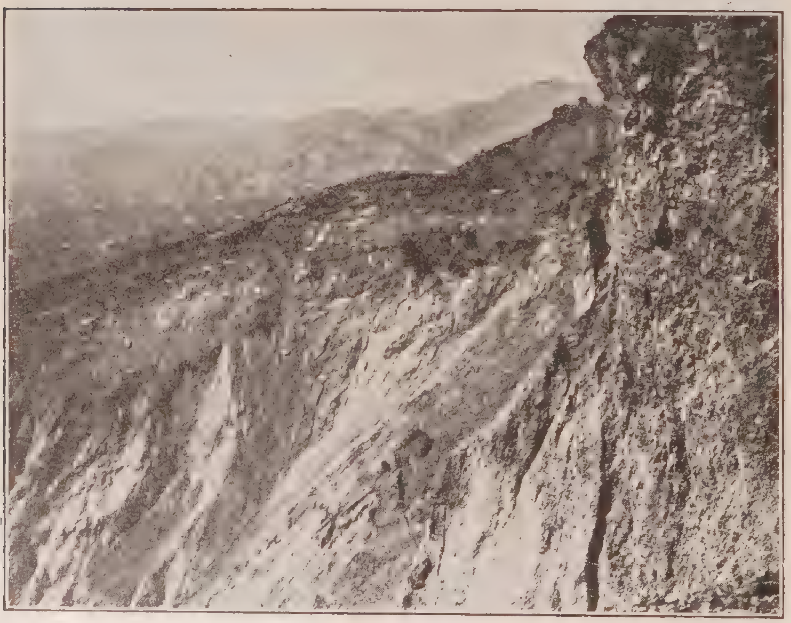

Fig. 3.—Vertical break through conglomerate at edge Miami-Inspiration caved ground.

Sinking of the surface to an enormous extent follows the mining of porphyry-copper deposits and of the Lake Superior iron-ore lenses by caving systems. Wide pits several hundred feet deep have been formed, as on the Michigan iron ranges and at the Miami and Inspiration copper mines in Arizona, and also at certain British iron mines in Cumberland.

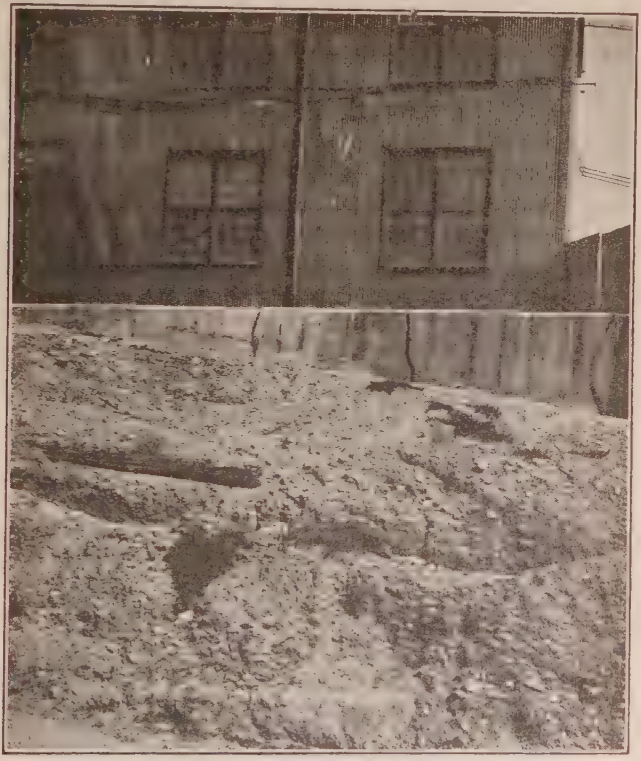

Fig. 4.—Subsidence draw, resulting in tension cracks in a Miami foundation without vertical displacement.

Unfortunately, only rarely is information available, or at least made public, regarding the ground movement at different periods of time and with relation to the extraction of the orebody.

Another phase of subsidence relates primarily to engineering works, tunnels, and excavations made in water-saturated sands, clays, and silt, where it is difficult or impossible to prevent some of the material running into the excavations. This subject has been comprehensively discussed by H. G. Moulton.[2]

Slides of Rock and Earth

The relation of rock slides near mining excavations to subsidence is not evident at first, but a relation exists although it is not simple. That mining may bring about rock slides in unstable mountain masses was demonstrated by the great Turtle Mountain slide at Frank, Alberta, in 1903, which swept away one portion of the town of Frank, killed 70 persons, and buried the tracks of the Canadian Pacific under 150 ft. of debris. This was one of the greatest rock slides in historic times, 35,000,000 cu. yd. of massive limestone breaking away and sliding on a rock joint plane of 30° to 35° from the horizontal, starting from the top of the mountain 3000 ft. (vertically measured) above the valley. A Canadian Government Commission, on which the writer served, decided that the slide started by the wall movement in a coal mine in a more or less vertical bed at the foot of the mountain. The pillars had been drawn off and the walls gave way.[3]

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[2] H. G. Moulton: Earth and Rock Pressures. *Trans.* (1920) **63**, 327.

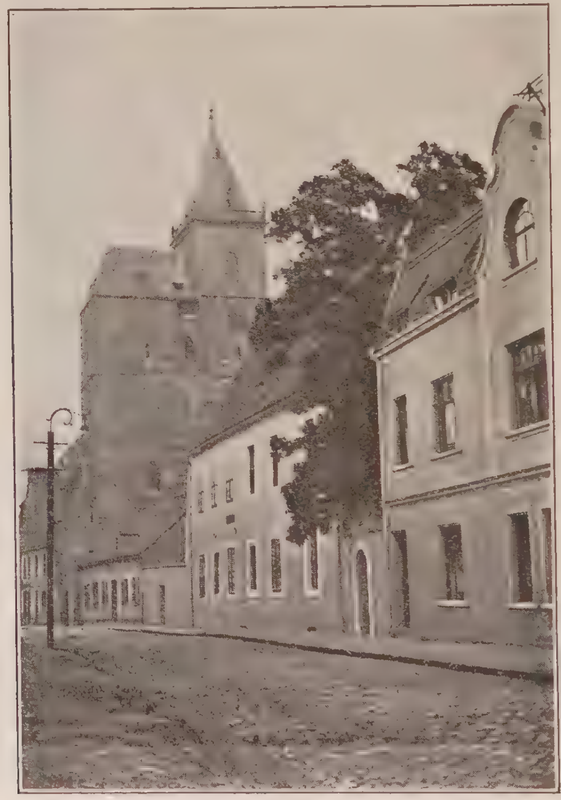

FIG. 5.—A 400-YR. OLD CHURCH THAT WAS LOWERED 21 FT. BY POTASH SALT MINING AT STASSFURT, GERMANY, WITHOUT CAUSING COLLAPSE OF CHURCH.

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[3] Commission was appointed, in 1911, to investigate the danger of a second landslide. See Canadian Dept. of Mines, *Memoir* 27; Report of the Commission appointed to investigate Turtle Mtn., Frank, Alberta.

In this case manifestly the angle of slide of the unstable rock was the most important factor. With loose material, that angle is only a few degrees steeper than the so-called angle of repose. Although the latter term properly refers only to an angle with the horizontal plane, at which loose material will stand without sliding, it has been applied to the stability of rocks or materials in places adjacent to a mining excavation, unaffected by ground movement.

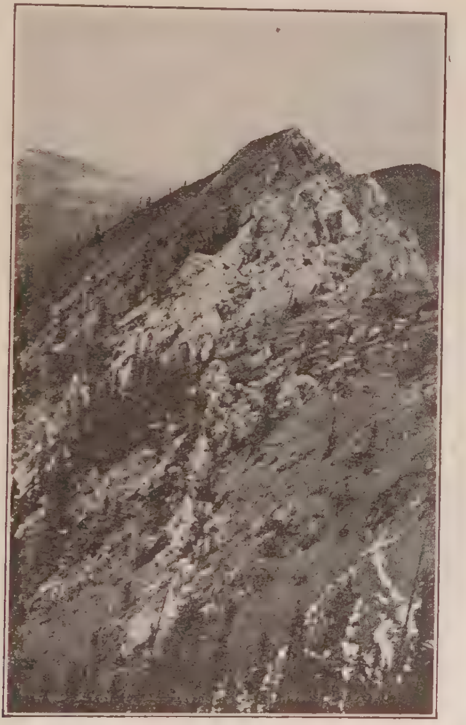

FIG. 6.—TURTLE MOUNTAIN SLIDE, ALBERTA, LOOKING NORTH TO NORTH PEAK ALONG A MAIN FISSURE WHICH IS EXPOSED FOR 1500 FT.

As stated by H. G. Moulton,[4] the term angle of repose is not really applicable to rock formations or earth formations under load. The author of this paper believes it may be convenient to coin a new term "plane of stability" to apply when stability is meant; this term properly applies to the permanent slope taken by loose material and therefore would have nothing to do with the stability of rocks in place. The plane of stability would be a curved plane extending from the edge of the mine excavation to the most advanced surface break. The draw line would be a chord of this curve. The curve is generally vertical at the surface but there is not sufficient data to know whether it is a mathematical curve in homogenous rock.

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[4] *Loc. cit.*

In water-bearing ground, or in structurally open and weak rock, as at Culebra Cut in the Panama Canal, hydraulic pressure doubtless plays an important part both by lubricating the rock joints and, if the head is considerable, by tending to burst the successive layers upward. In some slides at Culebra Cut, rocks moved when the apparent head was only one-tenth of the horizontal distance. In such instances, which D. F. MacDonald[5] terms "structural break and deformation slides," the rocks forming the bottom of the canal heaved as a shale floor may do in a coal mine where the pillars are overloaded. Considerations of this sort may have a most important bearing on the layout of large stripping operations, such as open cuts of great height at porphyry-copper mines, where the general slope of the cuts approaches the plane of stability of the rocks and a determining factor is the strength of the rocks to resist deformation.

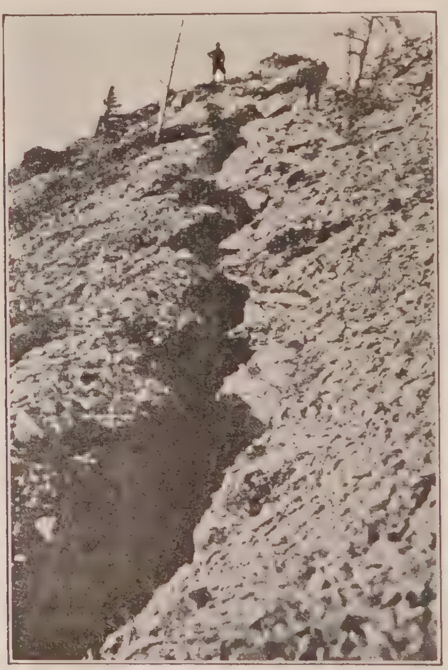

FIG. 7.—TURTLE MOUNTAIN SLIDE; A CREVICE RESULTING FROM "PULL" DURING 1903 SLIDE.

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[5] Some Engineering Problems of the Panama Canal, etc. Bureau of Mines *Bull.* No. 86.

Ground Movement

Without some ground movement, there could be no subsidence nor slides from mining excavations. The smaller movements of the immediate hanging wall or roof, in mining operations, are evident in every mine where falls occur or timbering is needed, and in deep mines by the flow of the rock gradually closing openings. Greater movements are manifest in orebodies mined by top slicing or an equivalent method, and in coal mines worked by longwall and pillar withdrawal. The successive changes between these first movements underground and the surface subsidence are surrounded with uncertainty, even in more or less horizontal homogeneous material. The variables of dip of strata, relative strength, presence of faults, method of mining employed, character and depth of overburden, and character of underlying stratum, add complexities.

Ground movements have many effects that concern mining in the most serious way both from a safety and an economic standpoint:

1. Simple falls of rock, or of ore.

2. "Bumps" so-called in coal mining, where the roof and floor materials are stronger than the coal, and the coal pillars, overloaded, suddenly and violently spall off. Bumps of this origin occur in the mines with deep cover, for example, at Sunnyside, Utah, and at Black Diamond and Carbonado, Washington. Another form of bump is that experienced in the Ten-Yard seam near Birmingham, England, and in the Crows Nest Mines.[6] In these bumps, it is conjectured that the gradual settlement of the immediate roof causes rigid rock strata above, comparable to beams or flat plates, under heavy load, to break successively when the subsidence leaves a wide enough space unsupported. Breaking of such a stratum causes a hammerlike blow on the lower rocks, which is transmitted as a shock wave to the immediate roof of the mine; as this roof is elastic, it does not give way, but loose material is thrown down, timber smashed, and wind blasts produced. Firedamp outbursts may be simultaneously caused, but are incidental.

3. Squeezes in coal mines where the pillars rest on soft clay; the clay flows and squeezes up the floor of the workings. This may cause subsidence, but it is slight, as the pillars remain in place.

4. Flow of rock, even of the granitic type, in mines 3000 to 5000 ft. deep, as in the Lake Superior district, slowly closing shafts or other excavations.

5. Extensive falls of hanging walls in metal mines, and rock readjustments in deep mines, which set up violent air blasts. Such falls sometimes occur in thick coal beds, with high open chambers, as in India.

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[6] Report by George S. Rice, Department of Mines of British Columbia, *Bull.* No. 2 (1918).

6. Subsidence and rupture of upper beds, this being irregular in effect where room-and-pillar methods are employed. If the beds are very close together, it may make the mining of the upper bed impracticable because of the difficulty and cost. Where the mining method is longwall, as practiced in Europe, or the workings are sand filled, the permanent damage is small.

7. In those metal mines working veins that are thick, dip steeply, have relatively weak walls, if the method of mining is sublevel caving, the walls may slump in with the ore and so lower its value. This must evidently be remedied, if it can be, by selecting an appropriate system of mining.

Mechanics of Ground Movement

No attempt is made in this paper to cover the numerous theories that have been advanced, from time to time, as to the mechanics of ground movement and subsidence from mining and the attempts to express the relation in formulas. This has been well done, up to 1916, by L. E. Young and H. H. Stoek,[7] whose bulletins were prepared under the coöperation of the University of Illinois, the Illinois Geological Survey, and the U. S. Bureau of Mines.

Nearly a century has passed since a commission was appointed (1825) to investigate the question of surface cracks in the vicinity of the coal mines at Liege, Belgium. That commission said that there was no danger from subsidence if the workings were deeper than 300 ft. In 1838, Gounot while studying subsidence in the same locality brought out his theory of the law of the normal, *i.e.*, planes normal to the dip of the bed limited the fracturing from an excavation. A similar theory was proposed about the same time by Trollez of France. Later, Rucloux and Durmond, of Belgium, Callon, Culomb, and Goupilliere, of France, Schulz, von Sparre, von Dechen and Hausse, of Germany, and Jicinsky and Rziha, of Austria, each developed theories. Schulz criticized the law of the normal; he considered that the plane of fracture varied with the material and in shale was vertical.

Although Fayol seems to be generally credited with developing the theory of the dome, Rziha apparently first proposed it (1881–2) on theoretical grounds only. He also believed that the stratification of the beds did not have much effect on the angle of break. He described a "falling space," which approximated a spheroid; the rocks within this space dropped when the pull of gravity exceeded cohesion. Surrounding the falling space, Rziha thought there was a "friability" or " tearing space." He also thought that when mining is carried on at great depth, the increase in volume of the rocks, when broken, may prevent disturbance of the surface, and proposed the forumla:

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[7] L. E. Young and H. H. Stoek: Subsidence Resulting from Mining. Univ. of Illinois *Bull.* 91 (1916).

L. E. Young: Surface Subsidence in Illinois, Resulting from Coal Mining. Ill. Geol. Surv. *Bull.* 17 (1916).

$$h = \frac{M}{a}$$

in which h = harmless depth;
a = coefficient of increase of volume;
M = vertical thickness of seam or of excavation.

Fayol, in France, made extensive laboratory tests as well as mine observations and summarized, in 1885, the contradictory opinions advanced up to that time. These are quoted in full, as they still seem to represent the wide diversity of views:

1. Upon the extension of the movement upwards.
   (a) The movement is transmitted to the surface whatever may be the depth of the workings.
   (b) The surface is not affected when the workings exceed a certain depth.
2. Upon the amplitude of the movements.
   (a) Subsidence extends to the surface without sensible diminution.
   (b) Movements become more and more feeble as they extend upwards.
3. Upon the relative positions of the surface subsidence and of the mining excavation.
   (a) Subsidence always takes place vertically above the workings.
   (b) Subsidence is limited to an area bounded by lines drawn from the perimeter of the workings and perpendicular to the beds.
   (c) Subsidence cannot be referred to the excavation either by vertical lines or lines normal to the beds, but only by lines drawn at an angle of 45° to the horizon, by the angle of repose of the ground, or by some other similar angle.
4. Upon the influence of gobbing.
   (a) The use of packing protects the surface effectually.
   (b) Packing simply reduces the effect of subsidence.
   (c) Subsidence is greater with stowing than without it.

Fayol's conclusions were, in effect, that the movements of the ground are limited by a dome that has for its base the excavation and that the amplitude of the movements diminishes as they extend farther from the center of the area, this also being true of the vertical effects. He also concludes that if the workings are of great depth there will be no subsidence of the surface—this because of the increase in bulk of the rock as broken.

Fayol's crushing and compression tests are of such interest that a rearranged and condensed statement[8] is shown herewith:

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[8] Abstracted and rearranged from tables given in Univ. of Ill.

VOLUMES OF DIFFERENT MATERIALS, WHEN CRUSHED AND COMPRESSED, VOLUME OF ORIGINAL MATERIAL BEING UNITY AND MATERIAL BEING CRUSHED TO GRANULAR SIZE (20 MM.)

	INCREASE IN VOLUME	VOLUME WHEN COMPRESSED BY PRESSURE OF 1422 LB. PER SQ. IN.[a]	VOLUME WHEN COMPRESSED BY PRESSURE OF 2844 LB. PER SQ. IN.[b]
Clay	2.16	1.00	0.90
Shale	2.29	1.28	1.16
Sandstone	2.14	1.36	1.25
Coal	2.02	1.30	1.25

[a] Corresponds to vertical rock pressure at depth of 1638 ft.

[b] Corresponds to vertical rock pressure at depth of 3276 ft.

U. S. Bureau of Mines' tests on broken anthracite mine rock (as naturally broken) when compressed in a steel cylinder by a pressure of 833 lb. per sq. in. showed a shrinkage in volume of 26.2 per cent. This pressure was considered to be equivalent to rock pressure at a depth of 860 ft. from the surface.

Since Fayol's work, many other engineers, in all lands, have investigated subsidence and most of them have put forward theories and empiric formulas regarding the angle of break, the angle of repose, the vertical subsidence to be expected, and the extent of the subsidence with reference to the mining excavation. The writer has found that the assumptions that must be made for coefficients in all such formulas that he has tried to apply are of such magnitude as to destroy the practical value of the formulas, except for identical conditions in the same district. The possible combination of the factors involved is infinite. The factors include the various strengths of rocks and earth; their relative dryness; their dip; the shape of the excavations, whether packed or not; and finally (the most important factor, although largely overlooked) the method of mining.

Diametrically opposed views are still held:

1. Surface subsidence always extends beyond the area of excavation. This view is supported by specific reports of British authorities on the subject as well as by many instances in this country.

2. Subsidence does not extend beyond the area of excavation. The mine subsidence committee of the Mining and Geological Institute of India has recently reported, after carrying on accurate observations at twenty-four collieries, that "Where no packing is done and pillars are taken out completely, the area of subsidence is less than the area of excavation; and in the case of seams dipping at less than 1 in 5 where no packing is done and pillars are taken out completely, there is no draw." In other words, the break does not extend beyond the vertical plane extending from the edge of the excavation. This view is supported by

specific data from mines in Pennsylvania and West Virginia that H. N. Eavenson has submitted from time to time.

VIEWS OF THE AUTHOR

The writer has had experience in the operation of mines Ladd and Cardiff in the longwall field of northern Illinois and has made observations that support the British view. That is, there is always a draw in advancing longwall, which system is the one generally practiced in Great Britain. As has happened in Great Britain, cracks opened in brick buildings in advance of the longwall face but these cracks largely closed when subsidence was complete a few years later. During the past few

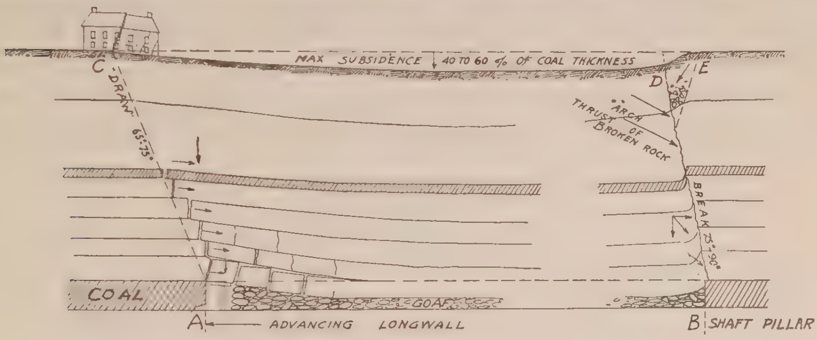
MAX SUBSIDENCE 40 TO 60 % OF COAL THICKNESS
C
DRAW 65°-75°
D
E
THRUST OF ARCH OF BROKEN ROCK
BREAK 75°-90°
COAL
GOAF
A
ADVANCING LONGWALL
B
SHAFT PILLAR

FIG. 8.—SUGGESTED MECHANISM OF LONGWALL SUBSIDENCE.

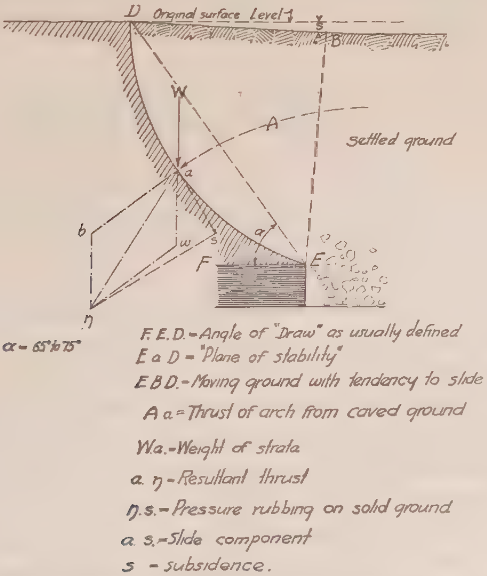
D Original surface Level
s
B
W
A
settled ground
a
a
b
w
s
F
E
η
α = 65° to 75°
F. E. D. - Angle of "Draw" as usually defined
E a D - "Plane of stability"
E B D. - Moving ground with tendency to slide
A a - Thrust of arch from caved ground
W a. - Weight of strata
a. η - Resultant thrust
η. s. - Pressure rubbing on solid ground
a. s. - Slide component
s - subsidence.

FIG. 9.—THEORY OF STRESSES IN SUBSIDENCE WHERE USING LONGWALL METHOD.

years, the survey monuments placed at other mines in the Illinois longwall district (Spring Valley) and the time studies made by the coöperative investigation[9] have established beyond question the fact of draw. On the other hand, the writer has had experience in mines using the room-and-pillar system and drawing pillars within panels, where the surface subsidence was well within vertical planes bounding the area within which the pillars had been extracted, a fact supporting the views of the India subsidence committee. L. E. Young reports similar findings in his Illinois investigations in 1916.

In the investigations of ground movement, the strength of rock layers acting as beams has been much discussed and also the arching stresses; both are undoubtedly important and the relative importance of each is difficult to determine. Where, however, the immediate rock stratum over the excavation, acting as a beam or flat slab, has broken (and as each successive rock layer breaks) the beam stresses are transferred to an arch spanning the excavation and resting on the solid strata on either side. There seems to be little question as to the actuality of the doming effect, but when the dome reaches the surface its shape may have been modified greatly by the dip and strength of the rocks and by faults. As to the effect of relative strength of the rocks, it has been repeatedly reported by investigators, and instances have come under the writer's observation, that shale, for example, breaks more or less vertically without regard to the dip of its bedding planes.

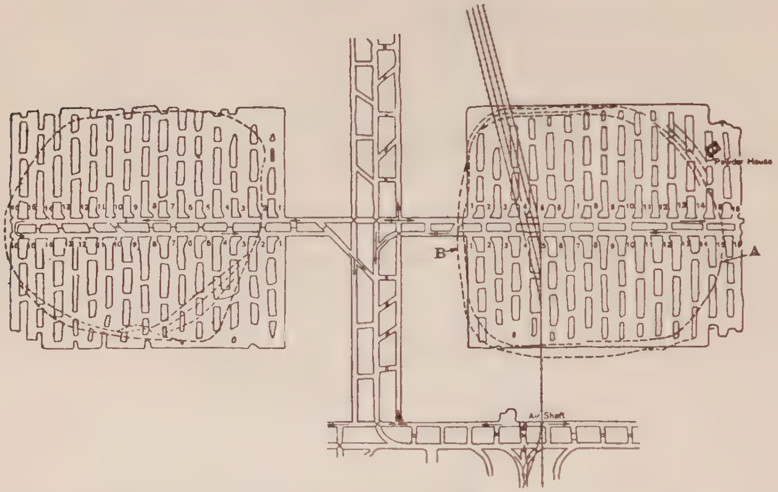

FIG. 10.—PLAN OF TWO PANELS OF A FRANKLIN COUNTY MINE 550 FT. DEEP; SUBSIDENCE CAUSED SURFACE CRACKS AS INDICATED; OVER THE NORTH PANEL, AT *A* THE CRACK WAS 8 TO 10 IN. WIDE, AT *B* WERE TWO SMALLER CRACKS.[a]

[a] From Ill.Geol. Surv. *Bull.* 17.

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[9] U. S. Bureau of Mines, University of Illinois, and Illinois Geological Survey.

When the excavation is in a bedded deposit or in a coal seam of considerable thickness, not back-filled or tightly packed, the overlying rock breaks when its span over the excavation becomes too great for its strength as a beam and the fragments, in falling, lie at various angles, thus increasing the volume. A rock layer over an excavation acts at either side as a beam fixed at one end projecting from the edge of the solid; therefore when it breaks near either support, the fracture plane is usually inclined inward toward the excavation. The layers above necessarily break in the same way; thus a flat dome is formed which, with the successive falls, becomes higher. All who have climbed up on the edges of caved ground in a mine must have observed the saucer-like shape of the initial breaks, where the ground is at all uniform.

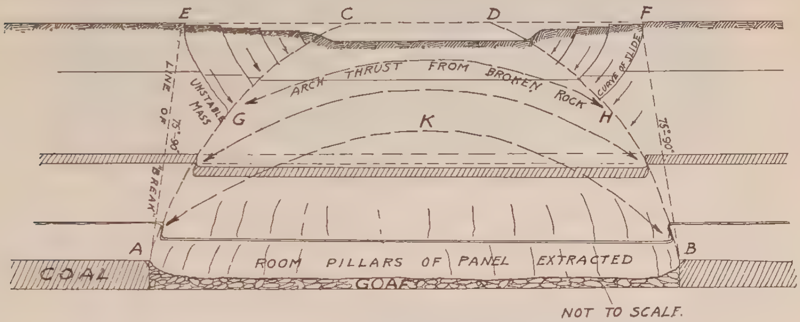

FIG. 11.—SUGGESTED MECHANISM OF ROOM-AND-PILLAR CAVING.

As the successive breaks occur the space between the broken material and the dome lessens; finally, if the depth below surface of the original excavation is not too great, in relation to its height and width, the dome breaks through to the surface. Meantime, as the rock fragments fall from the top and sides of the dome, a more or less conical pile of broken rock builds up. The falling fragments tend to roll down and wedge against the sides of the dome. When the pile has thus built up, say, to a height greater than the width of the excavation, the writer believes, there will be an arch thrust developed through the mass of more or less compacted broken rock which has a great effect on further spalling off of the sides of the dome or planes of break. This effect does not seem to have received enough attention in mining literature on subsidence.

Civil engineers, however, have considered it in connection with tunneling and trenching operations in earth and sand, as described in the discussion by J. C. Meem, of H. G. Moulton's paper[10] already referred to. Mr. Meem's experiments with dry sand in a cylinder are fully confirmed by the action of dry granular material stored in high bins. It is well understood by bin-construction engineers that the weight on the bottom of a bin is independent of the height of the column of material filled above a certain moderate height, as compared to the diameter of the bin. The weight of the material above this height is carried by the walls of the bin.

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[10] *Loc. cit.*

Every mining engineer knows that filling a stope in a vein with either ore or waste prevents the walls from caving; this is undoubtedly due to the arching effect produced in the loose rock. Furthermore, after the first load has been thrown on the stulls under a shrinkage stope, additional filling of dry material does not increase the load borne by the timbers. Also, in tunneling under caved ground, it is well known that although forepoling and heavy timber may be needed, the timber does not have to bear the full dead load of the loose material above it but only of that part under the natural arch formed in the broken material. This arching effect of the compacted rock fragments supports the sides of the dome and undoubtedly is the reason, where the mine excavation does not have too great a diameter, why there is no draw effect when the room-and-pillar or limited panel method of mining is employed.

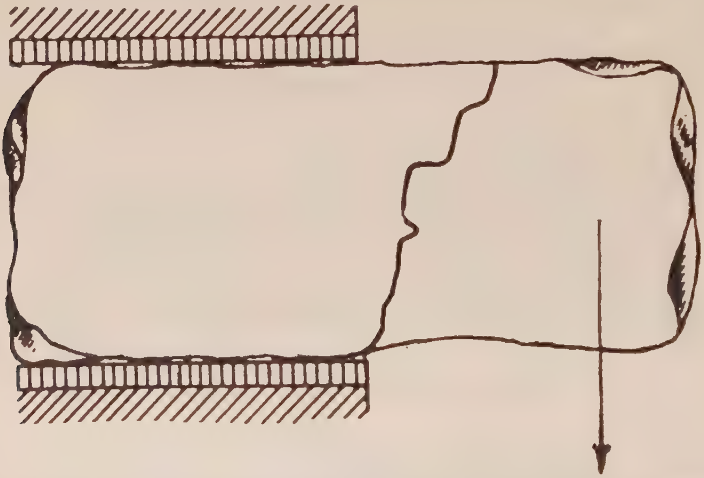

FIG. 12.—STONE LAYER ACTING AS A FIXED BEAM SHOWING HOW DOMAL EFFECT MAY BE STARTED IN HORIZONTAL STRATA AT EDGE OF EXCAVATED GROUND.[b]

[b] From Univ. of Illinois *Bull.* 91.

If the excavation in a bedded deposit continues on one or both sides, as in longwall or in a general retreating system, the stability of the buttresses of the arches is continually being impaired by undermining and the zone of fracturing extends beyond the vertical; this movement is probably assisted by the shifting arch stresses until we have the effects of draw always observed over longwall workings. The draw usually precedes at an angle with the horizontal, of 65° to 75°, but the actual plane of the break curves outwardly from the excavation and is affected by various conditions of the ground. In homogeneous rock, the plane of break is perhaps the resultant of the vertical weight component and an arch thrust from the broken rock. This thrust would add to the loading on the rocks in place, giving with the downward movement a disrupting effect. In a very thick orebody, say of porphyry or iron ore, lying at considerable depth and mined by a caving system, after the first general break to the surface a similar arching effect of the broken capping would probably prevent the side walls from falling until the descent of the capping by further caving had exposed them to a considerable height; then

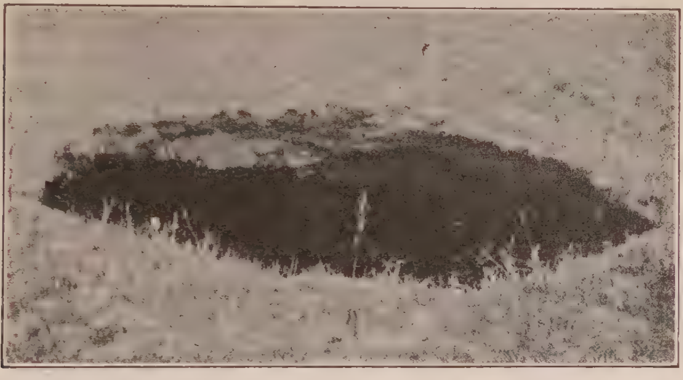

Fig. 13.—A subsidence dome breaking through to the surface.[a]

[a] From Ill. Geol. Surv. *Bull.* 17

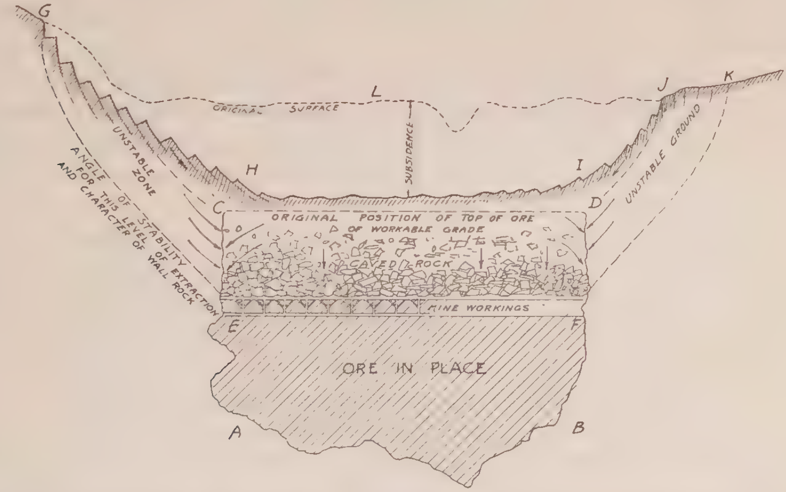
G
L
J
K
ORIGINAL SURFACE
SUBSIDENCE
UNSTABLE ZONE
ANGLE OF STABILITY FOR THIS LEVEL OF EXTRACTION AND CHARACTER OF WALL ROCK
H
I
UNSTABLE GROUND
C
D
ORIGINAL POSITION OF TOP OF ORE OF WORKABLE GRADE
CAVED ROCK
MINE WORKINGS
E
F
ORE IN PLACE
A
B

Fig. 14.—Suggested mechanism of subsidence in block caving in a large orebody.

normal rock slides would result, as is plainly indicated in the Miami-Inspiration cave.

The foregoing three types of ground movement in connection with subsidence take up only a few of the problems of this intricate subject, but they touch on the most disputed ones.

SUMMARY

The author's views on the general subject of subsidence are:

1. Mining by the room-and-pillar system, with the pillars left in, may not cause subsidence unless the pillars are very thin or the bed very shallow. Conversely, the dome of breaking will not reach the surface unless the deposit is very thick, in comparison to its depth below surface, as the broken rocks will wedge and their increase of volume by breaking will fill the dome.

2. When the pillars are partly or wholly extracted in a panel of moderate size, subsidence of the surface is inevitable after the lapse of some indefinite period of time.

When the deposit is of moderate thickness, in relation to its depth from the surface, and the lateral width of the deposit or panel is also of moderate size and the area excavated is surrounded by a barren pillar or with solid mineral, the surface subsidence will rarely be more extensive than the area of the deposit or panel; in other words, there will be no draw, because the arching thrusts of the broken rocks will buttress the side walls.

The vertical subsidence under the conditions just cited will be nearly as great as the thickness of excavation less the amount of gobbed material and mineral left in place. This assumes that the rock strata, after the first falls, will tend to come down as a whole and thus not swell the volume materially.

3. Mineral deposits mined by the longwall, or an equivalent, method will inevitably cause subsidence of the surface, independent of depth and will have a draw; that is, the subsidence will be in advance of the mine workings, and extend over the solid mineral to a line subtending an angle of 65° to 75°, from the horizon of the excavation. The angle will, of course, be modified by the dip of the strata and by faults, etc.

In longwall mining, the amount of subsidence ranges from one-half to two-thirds of the thickness of the bed mined, varying with the amount of packing done. As longwall is usually practiced only when the roof is more or less shaly, the packs are compressed so much that old ground when reopened is usually as tight and hard as the original ground.

Where hydraulic sand filling is used, as in Germany, the subsidence is negligible.

The writer's so-called plane of stability differs from the so-called angle of repose of loose material, inasmuch as the former is applied to rocks or dry earth in place. It is a curved plane extending from the edge of the excavation upward to the surface, where it meets the theoretical line of draw, therefore the line of draw is a chord of the curved plane. Below this curved plane, the rocks or ground are not affected by the ground movement. The position of this curved plane, with reference to a straight plane, will depend on the relations of the arch thrust of the broken ground, the sliding movement of the ground beyond the vertical, and the resistance of the rocks to rupture under the combined stresses. In this respect the author's ideas closely follow the views expressed by H. G. Moulton[11] three years ago.

CONCLUSION

In conclusion, the author thinks that, although it is most desirable to have formulas for determining just what the subsidence will be with certain mining conditions and methods, the empiric formulas so far developed have been based on such local conditions that to consider them of general applicability is, perhaps, unwise. Therefore the mining men should assemble specific detailed data on the problems of subsidence and ground movement. Although their data probably will not lead to the development of a formula of universal applicability, they will enable formulas to be established that will fit different classes of work.

Heretofore the assembling of precise data has been sporadic, the time now seems appropriate for the Ground Movement and Subsidence Committee to define what data should be recorded and to suggest a systematic, thorough, simple manner of record so that we may ultimately have a valuable series of accurate observations that will be of general use. One of the difficulties is that observations must be made over long periods of time. It is idle to put in monuments for observing subsidence and ground movement if, after a year or two, the observations are to be dropped. It is part of our function as a committee to show that the value of such information is not merely academic but has a monetary and a national value in its relation to the prevention of loss of mineral resources.

If the subject meets with the approval of the Institute, members might be asked to induce mining companies with which they have relations to coöperate with the Institute in making observations that may be listed in a schedule appropriate to the particular class of mineral deposit and method of mining, and to submit annually to this or another committee the records and results for that year.

(*Discussion of this paper begins on page* 414.)

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[11] *Loc. cit.*

Subsidence at Miami, Arizona

By J. Parke Channing, New York City

(New York Meeting, February, 1923)

The Miami orebody occurs in an altered Pinal schist. It is popularly known as one of the "porphyry" deposits but, as at Inspiration and Ray, the ore is an altered mineralized Pinal schist. The orebody mined is the zone of secondary enrichment.

Speaking in general terms, the area of the orebody at Miami may be considered as 1000 ft. long and 800 ft. wide. The vertical thickness varies from 200 to 400 ft., so that it is an extremely irregular mass.

The study of the ground movement and resulting subsidence at Miami is complicated by the fact that the formation is not homogeneous throughout the mining area, the orebody terminating on its eastern extremity at a fault plane adjoining the Gila conglomerate. This conglomerate is a recent sedimentary deposit somewhat analogous to glacial drift and consists of boulders of granite, schist, and other material cemented with clay, and at times is quite hard. The fault dips eastward at an angle of 45°.

The original method of mining was top slicing, the slices being 10 ft. thick. At first, there was a good deal of experimenting with mining methods. Some undercut rooms were tried, and the stopes were filled with broken ore with the idea of removing intervening pillars by top slicing. The adoption of the top-slicing method in itself involved considerable experimenting, including some experiments with a longwall system. The method finally adopted involved the operation of top slicing areas 250 ft. square, each with a central supply raise and ore-gathering raises, each block being worked as a unit.

Mining operations in the Captain orebody, which connects with the boundary of the Inspiration property, were started by the room-and-pillar system, the rooms extending across the orebody, with widths of 10 to 15 ft., with 15- to 10-ft. pillars between. The intent was to operate the rooms as shrinkage stopes, permitting the pillars to crush when the ore was drawn from the rooms. It was found that the shrinkage stoping was unnecessary, so a system of block caving was adopted. This system was so successful in the Captain orebody that, about three years ago, the method of mining in the orebody was changed from top slicing to block caving, the method now in use. The block caving system, as practiced at Miami, has been described by one of the engineers of the company, and also, in a general way, by the writer.[1]

As far as ground movement is concerned, the only difference in the operations is that instead of removing the ore in 10-ft. slices, it is being removed in 70- or 75-ft. slices. There is no evidence that the change in the mining method has changed the effect on the surface. T. K. Scott, one of the engineers of Miami Copper Co., has made the following notes on some of the important details of subsidence occurring in connection with the company's mining operations.

The ground around the drawing area starts to break in a series of small cracks, which vary in size from well-defined fissures, near the stoping area, to small cracks from 50 to 450 ft. from the mining zone. As operations advance, the cracks become more clearly defined, gradually developing into a series of step blocks from 2 or 3 ft. to 25–30 ft. across the top. This step blocking continues until it reaches the first development of the main escarpment for the lift working. A general block subsidence then occurs, the settling being bounded by the escarpment. The smaller blocks gradually open until a general sliding occurs, which rapidly disintegrates the ground in the main moving mass, and the ground eventually assumes an angle of repose, about the same as of broken rock.

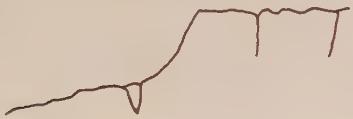

FIG. 1.—SCHIST ESCARPMENT.

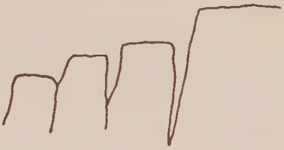

FIG. 2.—CONGLOMERATE ESCARPMENT.

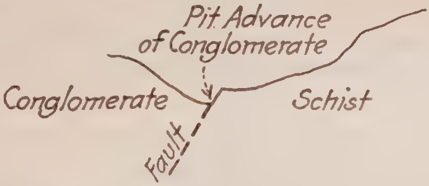

FIG. 3.—SUBSIDENCE ALONG CONGLOMERATE-SCHIST FAULT.

The schist escarpments, Fig. 1, are different from the conglomerate, Fig. 2, in that they do not shear so perpendicularly; there is usually a small angle to the wall. The conglomerate usually shears off decidedly perpendicularly. The schist is broken into a fine state much more rapidly than the conglomerate.

The southern end of the 545-ft. level was first observed for data concerning surface subsidence over an unstoped area. After operations had been started, a series of surface cracks were observed. These were followed by pipes to the surface over the area being mined and these pipes increased in size and number until they joined, forming a large pit.

Where the conglomerate-schist fault is located, the conglomerate has a tendency to slide down the hanging wall slightly in advance of the schist subsidence, at the same point—possibly 2–8 ft., Fig. 3.

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[1] Caving Systems of Mining, *Mining and Metallurgy* (1922) No. 181, 7.

In the conglomerate, as a whole, after the escarpment has been formed the general tendency is to maintain its general block movement and step-down form until it is almost halfway down the pit, when it starts to slide and break. This is noticeable in the main pit where many small blocks jut up against the skyline around the rim, Fig. 4.

The locality around the old garages and tanks was observed for data. Cracks around the garage opened about Sept. 20, 1920, minor ones having been noted earlier. The small fissures continued to widen until February, 1921, when the first subsidence was noted. The east lines on the 545-ft. level were mining near this area in October, 1920; drawing on the main E 700 line started February, 1921. Drawing the N 400 panel on the 620-ft. level was started about October, 1921. On Jan. 1, 1923, the flattest angle was 50°; this was on the 620-ft. level sill to a point 480 ft. horizontally to a crack noted on surface. The steepest angle was 82° 45′; this was on the 620-ft. level sill to a point 60 ft. horizontally to a crack noted on surface. The flattest angle recorded, N 250 section, Mar. 14, 1921, was 42° 30′. Well-defined cracks are noted in order that no error may occur in taking natural disintegration fissures that occur on surface.

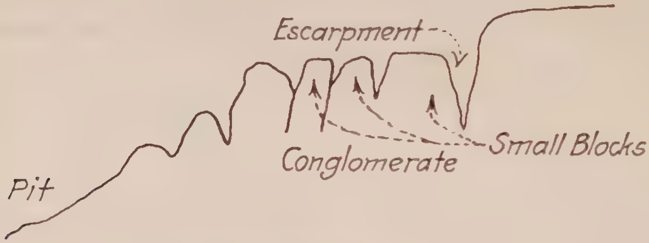

Fig. 4.

The time that elapsed between actual drawing operations and the date when the first hole or pipe appeared on surface was:

545-ft. level, southern end, drawing commenced		Apr. 3, 1919
Real line extraction July 15, 1919; 89 days, or 3 months	First hole to surface	July 12, 1919
Time		101 days or 3 months and 11 days
Tonnage extracted for 89 days		117,617 tons

Mr. Scott has made some rather elaborate calculations in regard to the volume of ore taken out to the volume of the subsiding area. Generally speaking, the area of the subsiding surface is about the same as that of the area of the horizontal projection of the largest dimensions of the underlying orebody. Because of the swell of the broken rock, the volume of subsidence is less. With the particular orebody under discussion approximately 1 cu. ft. of volume of subsidence results from 1½ cu. ft. of excavation. F. W. Maclennan, general manager of the Miami Copper Co., states that, generally speaking, the cracks resulting from the withdrawal of the ore and the corresponding subsidence are at an average vertical angle of 70°, but that one crack was recorded with an angle as flat as 42°.

The writer does not believe that the data observed in connection with the mining operations at Miami are sufficient to furnish a basis for valuable deductions regarding the effect of caving operations on the overlying surface, except perhaps that to be on the safe side in laying out a mining plant or other surface improvements a 45° angle should be assumed as the probable limit of breakage and no surfacei mprovements should, be considered outside the danger zone if they lie within a plane or angle extending upward from the nearest boundary of the underlying ore at an angle of 45° from the vertical.

When locating the No. 4 shaft, the location of the ore was fairly well known and allowance was made for the possibility of cracking or settlement within a 45° slope. The shaft is still intact, but the ore has been found at greater depths along the fault than was first expected, and the power house at the rear of No. 4 shaft has commenced to settle. In the meantime, a new shaft, No. 5, has been sunk farther east, at a greater distance from the orebody.

The location of a shaft involves considerations other than those involved in considering the possible damage from mining operations at some time in the future. It may not always pay to look too far ahead, for if the shaft is located too far from the mining operations, the cost of crosscutting, tramming, and maintenance may be greater than the saving resulting from prolonging the life of the shaft. This was worked out empirically a number of years ago in some of the Lake Superior iron districts. The calculations of the cost of the No. 4 shaft of the Miami Copper Co., as balanced against costs of tramming, drifting, maintenance, etc., confirm the wisdom of the original location of the shaft. The writer cites this as an illustration of the economic problems involved in the consideration of ground movement and subsidence: the possibility that it is at times better to accept probable loss of a shaft as a result of ground movement or subsidence from mining operations in the distant future than to incur increased operating, maintenance, or interest expenditures for the sake of prolonging the life of the shaft.

(*Discussion of this paper begins on page* 414.)

Mining an Upper Bituminous Seam after a Lower Seam has been Extracted

By Howard N. Eavenson, C. E., Pittsburgh, Pa.

(New York Meeting, February, 1923)

In many of the bituminous-coal districts of this country, more than one seam of workable coal exists, and in most cases the lower seam is the more attractive, owing to either its greater thickness or its superior quality. Apprehension that the mining of a seam will destroy the availability of all overlying seams has hindered the development of many fields and has led to the unprofitable working of certain seams in order to save them from an expected total loss.

The U. S. Coal & Coke Co., Gary, W. Va., was the first operator in the Pocahontas field to mine the No. 4 seam extensively, which occurs about 80 ft. above No. 3, the seam that was worked first. This was due to the fact that over large areas the upper seam was the only one of workable thickness, at that time. In two places, both seams were workable in the same area, the No. 4 being thinner and containing more impurities than No. 3 seam. In mining these areas, an attempt was made to keep the workings in the upper seam in advance of those in the lower one, and no pillar robbing was done in No. 3 seam until after pillars in that part of the upper seam directly over it had been mined. This resulted in an excessive development in the lower seam, with large areas of finished rooms and standing pillars. The top fell badly, and after a couple of years the falls became so heavy that pillar drawing was unduly expensive. To overcome this obstacle and allow No. 3 seam to be worked in some territory where No. 4 was thick, but too dirty for present use, a study was made of operating methods applied to superimposed seams in Pennsylvania and Maryland. This was started more than seven years ago by the writer; his data, supplemented by more recent information gathered by his associates, and by two cases mentioned in recent foreign publications, form the basis of this paper.

Central Pennsylvania

In central Pennsylvania, the lower coal measures contain several seams of nearly the same thickness but of varying quality, and as mining has been conducted here for many years, numerous instances can be found where an upper seam has been mined after a lower one. A general section of the measures in Cambria and Somerset counties is shown in Fig. 1. The coal seams vary from 3 to 5 ft. thick and occur at intervals of 40 to 80 ft.; the intervening rocks are usually shales, slates, or sandstones. By correspondence with engineers, mine inspectors, and operators, it was learned that, in many places, an upper seam was being worked with no trouble after a lower seam had been mined. In four mines visited, a careful investigation in the upper mine failed to reveal any traces of damage caused by the removal of a 4-ft. seam 80 ft. below: and in one mine a heading over a robbed area was of double-track width, without any posts, and under good roof. In this mine, it was reported that in driving the main heading in the upper seam, over a robbed area in the lower, a settlement of about 8 in. had been noticed, although the coal was in good condition. Rooms are driven 23 to 26 ft. wide with three rows of posts. It is usually necessary to drill holes to the lower seam to drain the water from the upper seam.

In another mine in this locality, recently examined, the *E* seam is being worked about 119 ft. above an area where the *C′* seam was removed some years ago. Above the *E* seam are 8 or 9 ft. of slates and shales under a 28-ft. layer of sandstone; the intervening strata are slates, shales, sandstones, and the *D* seam, the latter being about 32 in. thick and lying 25 ft. above the *C′* seam. Mining was attempted in the *D* seam, but was abandoned on account of the damage done by mining the lower seam. In the *E* seam, little settlement has occurred, but in certain places lying directly over robbed areas in the *C′* seam, heavy falls are experienced, which frequently extend up to the sandstone; this does not happen where the *C′* seam has not been robbed. Cracks came up to this seam, draining the water off, but the overlying sandstone seems to be undamaged. Mining is considerably more expensive, due to the amount of dead work and timbering required, and the coal produced is of very fine size.

One case was found where three seams in the same area were leased to three different operators, without any stipulation as to sequence of working, except that protection should be left in the lower seams under the main headings in the upper ones, if any settling was observed.

SOUTHERN PENNSYLVANIA

In southern Somerset county, four mines are operating in the Redstone seam, immediately over the Pittsburgh coal. The latter seam averages about 8 ft. thick; the Redstone coal varies from 4 ft. to 4 ft. 10 in., and is overlain by a fireclay 8 to 11 in. thick, bone and impure coal, 14 to 20 in. thick, and a slate roof. Between the two seams the strata are mostly slate, which breaks readily into small pieces, but the actual bottom of the

Redstone seam is a hard limestone from 2 to 6 in. thick. According to careful leveling, the distance from the top of the Pittsburgh to the bottom of the Redstone coal varies from 35 to 19 ft., the average being

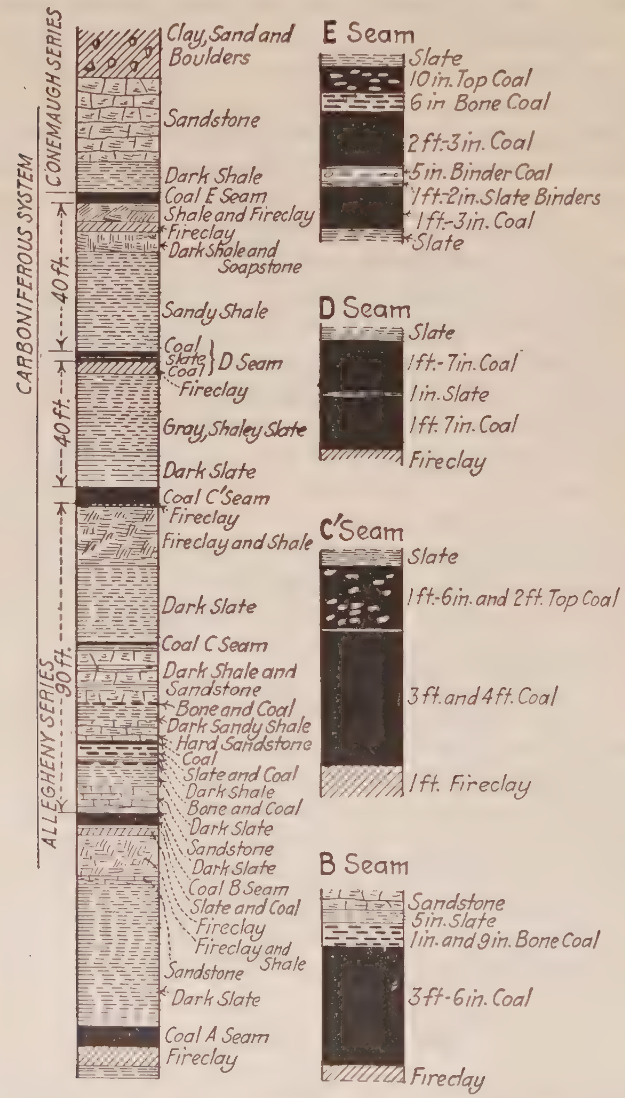

FIG. 1.—COLUMNAR SECTION OF STRATA NEAR PORTAGE, CAMBRIA CO., PA.

slightly over 26 ft. Generally, the headings in the upper seam are driven over those in the lower one, but sometimes this is not done; headings are 9 to 10 ft. wide. The fireclay over the coal is taken down and loaded out in both headings and rooms; the bone streak above this, wherever possible, is left up for a roof, and generally it stays up well. Where a heading or a room crosses a robbing line in the lower seam, a break a few inches wide is always found in the seam and the top, the coal bending and dropping a maximum of 3 ft. vertically in a distance of 8 ft. horizontally. Fig. 2 shows this occurrence.

The experience in that district is that any two seams with this thickness of strata between them can be mined successfully, if the lower seam has been entirely removed before mining the upper one; that the cost of mining the upper seam has not been increased perceptibly by the removal of the lower one; that the percentage of recovery has not been materially affected, nor has the element of danger been increased. It was the opinion that the upper seam could be worked three months after the lower one had been mined. It was generally necessary to drill holes to the lower seam to drain the water from the upper one.

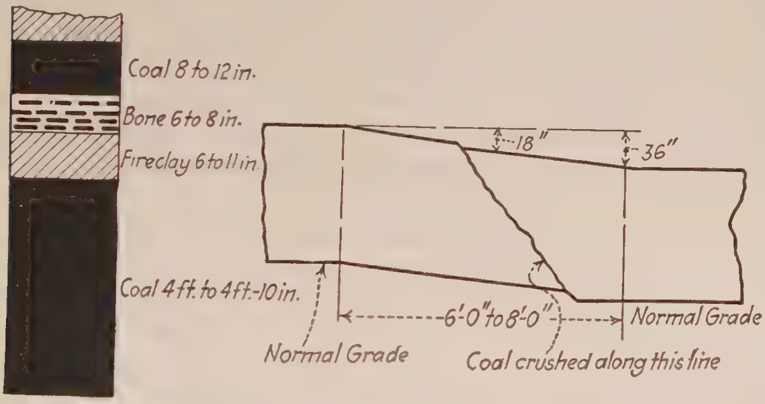

Fig. 2.—Break in Redstone seam when passing from solid coal to robbed area in Pittsburgh seam.

At one mine, the engineer for the lessor stated that a recovery of 90 per cent. was being secured in the Redstone seam and that the cost of mining was not more than 4 to 6 per cent. more than if the lower seam had not been mined. In view of the fact that surface is often badly broken by robbing at depths of 200 ft. and more, the small amount of damage done to the upper seam was almost unbelievable.

George's Creek, Maryland

In the George's Creek, Md., and Connellsville, Pa., regions, extensive mining is now being done in the Tyson or Sewickley seam in areas where the underlying Pittsburgh seam has been removed. In the George's Creek field the Tyson seam varies from 28 to 36 in. thick; the top is a sandy slate or shale, which shoots well and stays up well. The Pittsburgh coal has a total thickness here of 14 to 16 ft., of which 12 to 14 ft. has been mined. The interval between the two seams is from 77 to 81 ft., composed of shales, slate, and 16 ft. of sandstone. Very little attention has been paid to the location of the workings in the upper seam with regard to those below. In these mines the greatest damage found in the upper seam was an area about 150 ft. wide over which the roof was so badly broken to a height of 6 to 8 ft. above the coal that it would have been unsafe to drive rooms under it. This section was directly over a barrier pillar in the lower mine between one robbed section and another. The damage could not all have been due to the lower workings, as nowhere else could more than a few cracks be found, or a very infrequent break, similar to Fig. 2, but having a drop of only 18 in. The only other trouble experienced was where rooms or headings had been driven in the upper seam and robbing had been done afterward below them. The water from the upper seam usually drains through cracks to the lower one.

Connellsville, Pennsylvania

In the Connellsville region, where the Sewickley seam is being worked it has an average thickness of 5 ft., the Pittsburgh coal averaging 8 ft. The bottom of the Sewickley coal is usually a hard fireclay, although sometimes a thin slate occurs between the fireclay and the coal. The top is either limestone, sandstone, or slate. The interval between the seams is 72 to 142 ft. of shales, sandstone, limestone, and slate. Careful examination of 13 mines in the upper seam showed the existence—over many robbed areas of the lower seam—of surface cracks in which percolating water and air had disintegrated the top rock, causing local falls sometimes as high as 4 ft. These conditions were not observed where robbing had not been done in the lower coal. Gas was frequently noticed entering the upper seam from the mines below. Where sufficient time had elapsed after robbing the lower seam, nothing was found in the upper seam that indicated any material increase of cost or of danger, or a diminished recovery of coal. The surface over many of these mines had been broken by robbing in the Pittsburgh coal, but in spite of this, numerous pools of water, in one case covering an area of several acres, accumulated in the upper seam and had to be removed by pumping. These pools were over areas which had been mined out by the Frick Coke Company.

In two cases, where the upper seam had been developed first, when robbing was done in the lower seam, subsidence of 3 to 4½ ft., with heavy roof falls, occurred in the upper one.

THE POCAHONTAS FIELD

During recent years, several instances of workings above No. 3 seam have occurred in the Pocahontas field. In one case, No. 4 seam, with an average thickness of 4 ft., is being worked over an area where the lower seam, 5 ft. 5 in. thick, has been robbed. The roof of the upper seam is 5 ft. of slate under heavy sandstone; the bottom is 3 ft. of fireclay. The interval of 60 ft. is mainly sandstone, with some slate. Robbing in the lower seam produced no visible effects in the upper except possibly some roof falls which, however, were not different from those usually expected with that kind of roof. There is some question in this case as to whether the robbing in the lower seam had been thorough enough to cause serious falls.

In another mine, No. 6 seam, averaging 3 ft. 8 in. thick, with sandstone top and fireclay bottom, is being worked where No. 3 seam, 5 ft. 9 in. thick, has been mined beneath it; the interval of 150 ft. is almost entirely sandstone, except a few feet of slate over the lower seam. The only evidences of disturbance in the upper seam are in two places where subsidence and roof falls occurred, the robbing in the lower seam having been done after the upper-seam workings were driven. In all other places conditions in the upper seam appear normal and water frequently collects over robbed areas.

In still another mine, a higher seam, about 340 ft. above No. 3, is being worked where the lower seam has been entirely removed. The only trouble here has been due to surface cracks which allow water to enter the mine, from which it does not drain well to the lower seam.

In a mine in southwest Virginia, robbing in a seam 5 ft. 10 in. thick caused cracks in the surface 570 to 794 ft. above, along a line more than 1000 ft. long. An upper seam occurs 445 ft. above the lower one, and some trial headings were driven in this to learn the extent of the expected damage done by the robbing. Except for some cracks, through which air came from the lower workings, absolutely nothing was found in the upper seam, either in levels or in the character of the coal, to indicate that the lower seam had been removed.

NATAL, SOUTH AFRICA

In a paper by Wm. Taylor Heslop,[1] the following experience at St. George's colliery, Natal, is described. There are two seams here, the top one about 4 ft. thick; the bottom one, including 9 in. of carbonaceous shale, has a total thickness of 5½ ft. Between the two seams is a bed of laminated sandstone, varying from 3 to 5 ft. thick. The immediate roof of the upper seam is about 17 ft. of shale, the total cover being 250 to 350 ft. of interbedded sandstones and shales. The seams are approximately level. At first, the top seam was totally extracted from about 8 acres and the area was allowed to settle for two years before the bottom seam was started. Several trials of this scheme, with both room-and-pillar and longwall mining, showed that the weight of the gob on the intervening sandstone caused trouble and loss of coal when working the bottom seam. The plan finally adopted was to drive all workings in the bottom seam, connecting the haulage ways by inclines to the top seam. The bottom seam was developed by blocking it into pillars 39 ft. by 75 ft., the working places being 15 ft. wide. The top seam was then developed in the same way, the workings being exactly over those below. When robbing commenced, a lift 18 ft. wide was driven 39 ft. across the end of the top pillar; when this was completed and the fall had occurred, a corresponding lift was driven in the bottom seam, making the gob line in the top never more than 33 ft. (18 ft. lift plus 15 ft. working place) in advance of the bottom seam. The overhang of the solid roof protected the working lifts and the sandstone between the two seams protected the lower lift from the gob above.

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[1] *Jnl.* Chem., Met. and Min. Soc., So. Africa (October, 1921).

EXPERIENCE AT GLASGOW, SCOTLAND

In a paper before the Mining Institute of Scotland, Dec. 9, 1922, the following experiences are described. Near Glasgow, the Main and the Jewell seams lie about 710 ft. below surface. The Jewell seam, the lower one, is 22 in. thick and is separated from the Main seam, which is 24 in. thick, by 4 ft. of fireclay. In some places the upper seam was worked first, but in one place the bottom seam had been worked out for about 3 years before work started in the upper seam. The upper coal was found to be friable and produced much slack; although the roof was hard, it was broken, as a result of the first working, and the roads were difficult to keep open. The bottom, being soft, had a tendency to heave, and in some places it had dropped away from the coal. Care had to be taken

Section	Output per Shift at Coal Face, Lb.	Miners' Wage Rate, Cts.	Machine Cutting per Ton, Cts.	Brushing, per Ton, Cts.	Repairs and Overhead, Cts.	Total Cost, Cts.	Timber Cost, Cts.
Main (top) coal, where bottom is mined out........	3277	$x+19.8$	$x+5.0$	$x+6.4$	x	$x+27.2$	$x+1.2$
Main (top) coal, solid......	6101	x	x	x	$x+3.1$	x	x
Jewell (bottom) coal, solid.	2975	$x+63.2$	pick places	$x+3.1$	$x+8.7$	$x+47.2$	$x+0.6$
Jewell (bottom) where top coal has been mined out..	2200	$x+32.0$	$x+17.4$	$x+41.2$	$x+23.2$	$x+\$1.106$	$x+7.2$

when the mining machine was crossing an old road in the lower seam. By this method, less timber was required and less dirt was produced than when the lower coal was worked after the upper coal had been taken out.

The table shows the comparative costs of mining in both seams and under both conditions; x representing the lowest cost (the shilling has been figured at 24 c.).

The statement is made that when either seam is worked the other is more or less damaged, to an extent depending upon the nature of the strata. Where the top seam was worked first, the damage consisted in hardening the coal, which was of no importance when machines were used. Where the bottom seam has been worked first, the damage consists in a large percentage of slack, which is not objectionable if the coal is for coking purposes. "Hence, one might say that the injury would be either real or apparent, according to condition."

CONCLUSIONS

Experience in all the above-mentioned localities seems to warrant the following conclusions:

1. Mining an upper seam after a lower one has been removed can almost always be successfully done when the thickness of intervening strata is 19 ft. or more.
2. The lower seam should be entirely removed from any area, and time should be allowed for settlement, before working is started in the upper seam; the more complete the extraction, the less will be the likelihood of trouble in the upper seam.
3. Working in an upper seam should not be attempted while robbing is going on in a seam below it.
4. Percentage of recovery in an upper seam will not be materially reduced, nor the cost or danger of mining be greatly increased, because a lower seam has been removed.

(*Discussion of this paper begins on page* 414.)

Examples of Subsidence in Two Oklahoma Coal Mines

By J. J. Rutledge, Baltimore, Md.

(New York Meeting, February, 1923)

On Sept. 4, 1914, Mine No. 1 of the Union Coal Co., Adamson, Oklahoma, suddenly caved, entombing thirteen miners whose bodies were never recovered. The seam of coal mined, the Lower Hartshorne, averaged 5 ft. in thickness and dipped at an angle varying from 22° to 30°. The strata overlying the coal seam were sandstones and shales.

The mine was worked by the pillar-and-room method, the rooms being driven directly up the dip of the seam. All coal was blasted off the solid. Rooms were turned on short centers and room pillars were thin and irregular, as shown in Fig. 1. Fig. 2 is a cross-section of the seam showing the mine workings.

Some cracks had been observed on the surface, it is said, at least a month before the squeeze occurred, but the collapse of the mine workings came very suddenly. As is usual when squeezes occur on such steep dips and the pillars fail, the crush progressed rapidly down the dip until it reached solid coal; there was a line of break, or draw, over in the solid coal. Several well-defined cracks appeared on the surface, after the mine had squeezed, but there was no appreciable subsidence of the surface, the weight seemed to be thrown entirely to the dip.

About one year after the squeeze, and after the workings of the adjoining mine, No. 4, had been retimbered and made safe, the writer inspected the tenth north (right) entry, or lift, off of No. 4 slope, which was below and fairly near the lowest caved portion of mine No. 1. The roof shale was composed of thin layers, which had been bent in somewhat the same manner as the leaves of a book having flexible covers are bent when the book is folded or rolled up in the hands.

Though the mine workings are immediately beneath the streets and buildings of the mining town of Adamson, business was not disturbed by the squeeze, so far as any subsidence of the surface was concerned. No surface monuments were available and no elevations were taken, but the mine workings had been accurately surveyed and traversed, and the streets of the town overlying the mine workings had been carefully platted on the mine map, so that it was possible to locate accurately the cracks on the surface.

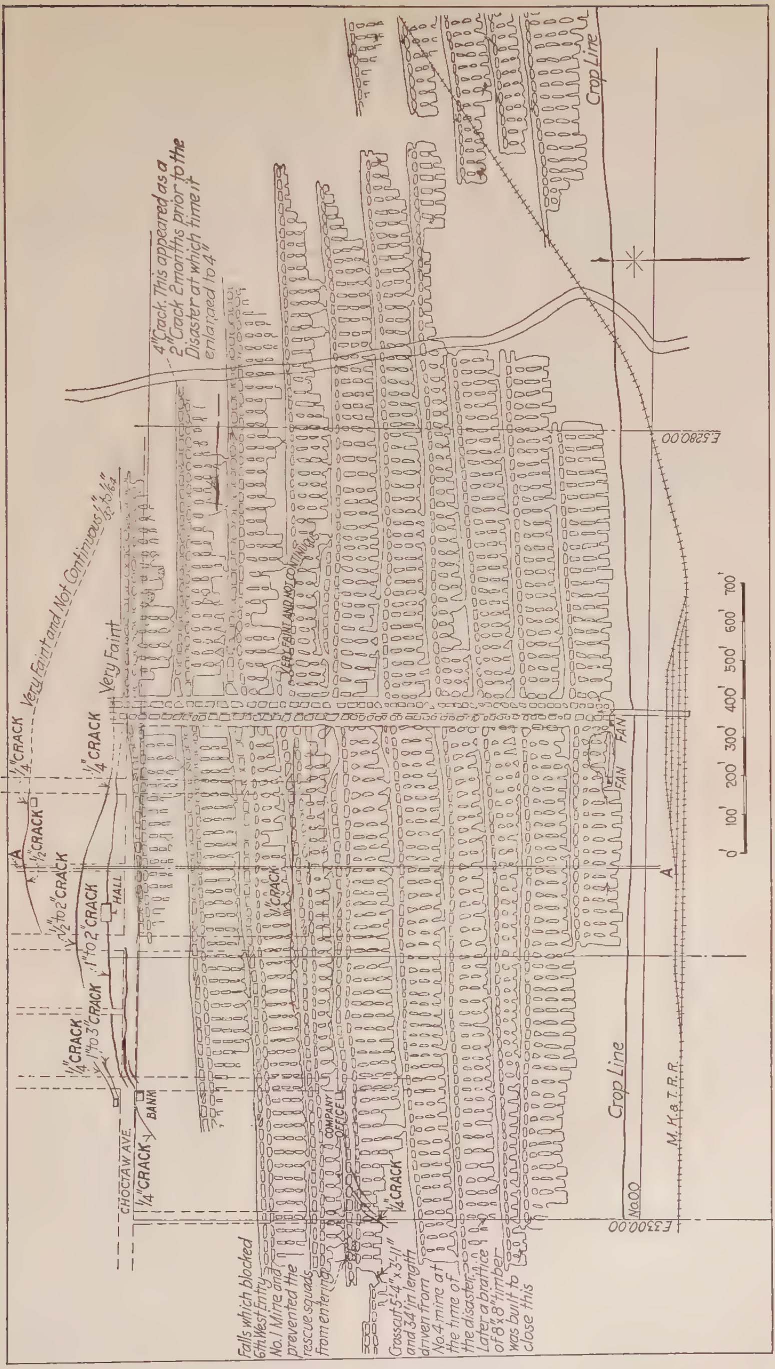
1/4" CRACK
A
1/2" CRACK
1/2" to 2" CRACK
1" to 2" CRACK
1" to 3" CRACK
1/4" CRACK
1/4" CRACK
1/4" CRACK
CHOCTAW AVE.
BANK
HALL
Very Faint and Not Continuous 1/32" to 3/64"
Very Faint
4" Crack. This appeared as a 2" Crack 2 months prior to the Disaster at which time it enlarged to 4"
VERY FAINT AND NOT CONTINUOUS
1/4" CRACK
COMPANY OFFICE
1/4" CRACK
Falls which blocked 6th West Entry No. 1 Mine and prevented the rescue squads from entering
Crosscut 5'-4" x 3'-11" and 34 ft in length driven from No. 4 mine at the time of the disaster. Later a brattice of 8"x8" timber was built to close this
Crop Line
Crop Line
FAN
FAN
A
E.5280.00
E.3300.00
No.00
M.K.&T. R.R.
0 100' 200' 300' 400' 500' 600' 700'

FIG 1.—MINE NO. 1, UNION COAL CO.

In eastern Oklahoma, where most of the coal seams have fairly large dips, such subsidence as has occurred does not appear to have greatly damaged the surface. This is doubtless due to the dip of the seams, the cases investigated being where the coal seams dipped at angles varying from 10° or 12° to 35° and 40°, the average dip in most of the mines being about 20°.

Underground mining operations began to affect the Irving school, in Henryetta, early in 1917, and continued to affect it until the fall of 1922, when the building was considered unsafe and was torn down. The building was a one-story structure, about 70-ft. square; it was built of fairly large paving bricks, rough and uneven in shape, laid with rather thick mortar joints—in some cases 1 in. thick.

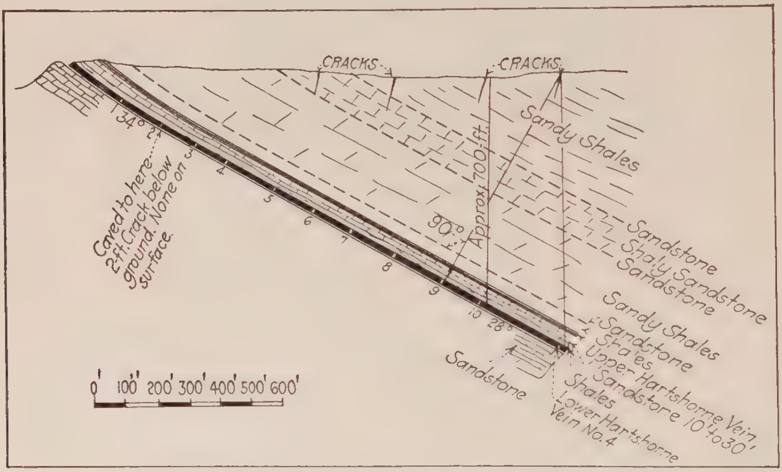

FIG. 2.—GENERAL SECTION OF STRATA AT MINE NO. 1.

The school stood immediately over the workings of mine No. 2 of the Crowe Coal Co., formerly The Whitehead Coal Co., the fifth north entry off the main west entry passing under the middle of it. The coal seam, known as the Henryetta, is approximately horizontal and, at this point, lies at a depth of 165 ft. Its average thickness is 3 ft. and it is practically free from any persistent partings.

The mine is worked by the pillar-and-room method with two entries; the entries of each pair usually being driven parallel to each other on 39-ft. centers. The entries were 12 to 15 ft. wide, with an entry or stub pillar 12 to 18 ft. wide between them. Rooms 40 to 50 ft. wide were turned off both entries at right angles. The room pillars were only from 5 to 15 ft. thick. Entries were brushed in the roof to a depth of about 2½ ft., or to a total height on the entry rail of about 5½ ft.; rooms were not brushed. Much of the brushing was gobbed in the cross entries. The coal was undercut in the lowest part of the seam by shortwall electric mining machines.

The area beneath the school building was entirely worked out, the nearest solid coal being about 100 ft. west of the west end of the building. Neither room nor entry pillars were robbed, but subsequent to the first brushing, the entry had been brushed unusually high to overcome a swag in the roadway.

In 1918, because of a swag, or depression, in the roadway of the fifth north entry, the roof was brushed a second time, the total cut being in some places 9 ft. in height and extending from *A* to *B*, Fig. 3. In all probability, the heavy blasting incidental to the second brushing was a contributing cause of the subsidence that damaged the school, although there had been signs of subsidence a year before.

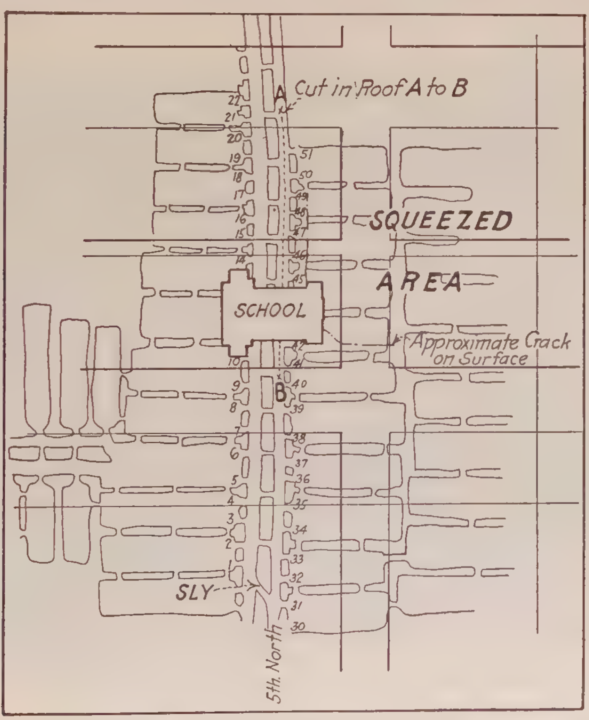

FIG. 3.—PART OF MINE NO. 2 SHOWING LOCATION OF SCHOOL.

All the rooms on each of the entries, near the cut, were caved. At a point about 840 ft. north of the main west entry, there was a cross-over from the main fifth north entry to the back entry. This cross-over started at a point on the back entry about opposite room No. 1. A stub entry was driven west from the face of room No. 4 or No. 5. All the rooms on the back entry were still standing up to room No. 11, where the caving began. These rooms were fairly well timbered near the ribs, but needed timber over the roadways.

The strata overlying the coal seam was mostly shale and thin beds of sandstone. The immediate roof over the coal seam is a rather limey gray shale. The roof in the mine workings is fairly good in character, as shown by the width to which the rooms can be driven—as much as 45 ft. in some cases and almost universally 40 ft. wide. The floor is fairly hard gray shale, that does not readily heave.

Cracks began to appear in the walls of the building early in 1918 and finally became so wide that some of the bricks that had been subjected to stress, because of the subsidence, were replaced by new bricks. The building continued to be affected by subsidence and was abandoned in the autumn of 1922.

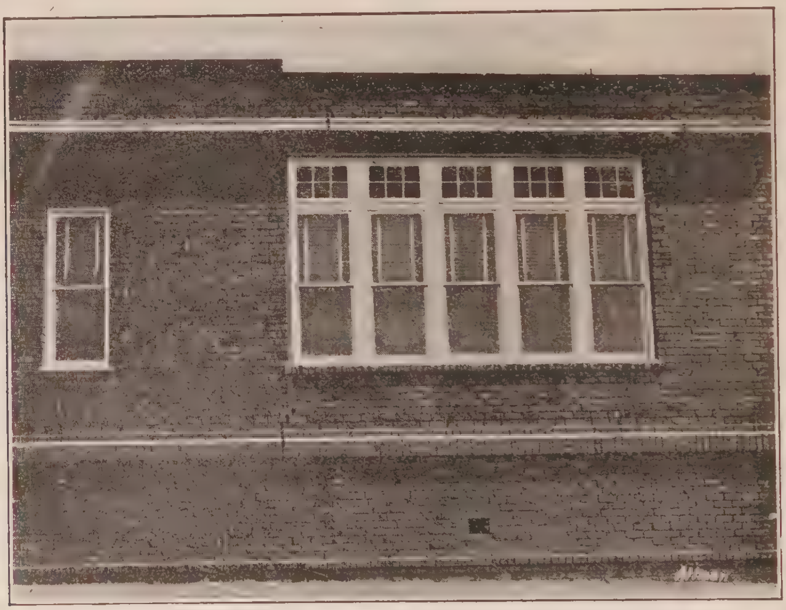

FIG. 4.—CRACK IN WEST WALL AT SOUTHWEST CORNER OF BUILDING.

The photographs shown in Figs. 4 and 5 were taken in the spring of 1922; they show a crack on the surface of the ground extending up to and connecting with a crack in the wall of the building. Near the upper end of this crack, the metal cornice had separated at a joint until there was a gap of 6 in. between the two ends that formerly abutted. There was some settling of the adjacent corner of the building but it was insignificant in amount compared with the horizontal movement exhibited in the cornice. The two sections of the cornice had apparently been separated by forces acting in a horizontal direction. Similar extreme horizontal force was noticed on brick buildings overlying longwall mines at several places in the northern Illinois coal field where subsidence investigations were carried out. In many cases, the horizontal component appears to be much stronger than the vertical.

Attempts were made to arrest the subsidence by retimbering the entries beneath the school building, but with no apparent benefit. As previously noted, the bricks used in this building were unusually large, with thick joints. As a result, the walls of the building did not withstand the effects of subsidence so well as the walls of other buildings subjected to subsidence elsewhere, which were constructed of smaller or more evenly sized brick with thinner joints. Had the brick been smaller, with thinner joints, it might not have been necessary to demolish the building. There was no very appreciable subsidence of the surface. The building was on one of the streets of a city of about 16,000 population, but there was no apparent inconvenience to the citizens as a result of the squeeze, and traffic on the streets was not disturbed.

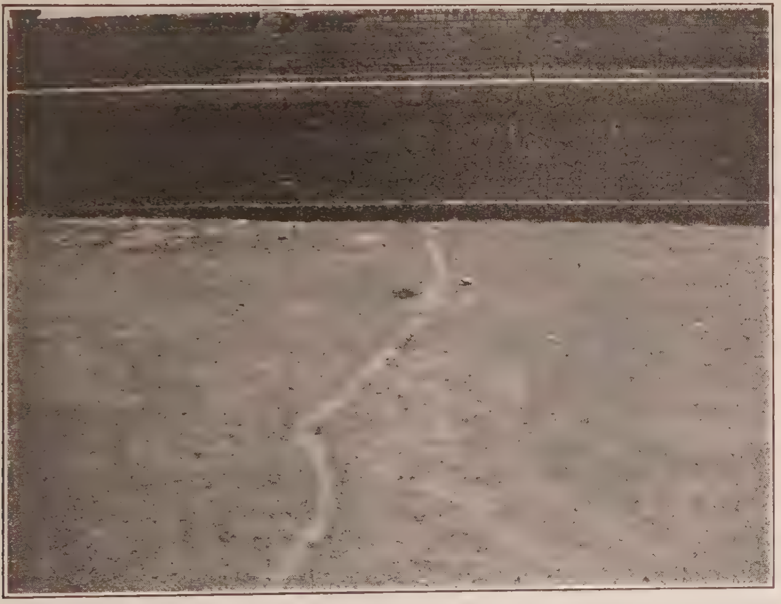

FIG. 5.—SAME CRACK AS IN FIG. 4 SHOWING CRACK IN SURFACE OF GROUND.

Broadening the consideration of the subject, I found that in Illinois, there was no subsidence of the surface where 50 per cent. of the seam was mined and the remainder left as pillars to support the surface. The seam was practically horizontal, under a cover of about 600 ft., the thickness varying from 10 to 12 ft. In mines under investigation, where the method of mining was panel by pillar-and-room and the room pillars were robbed, generally by slabbing, there was subsidence in the panels, the surface sinking to a maximum depth of 2 ft. and the surface cracking in concentric cracks, opening to a width on the surface of 6 in., but the subsidence did not extend beyond the limits of the panel. A barrier pillar was left around each panel and the subsidence ceased before it reached the surface over the barrier pillars, the overlying roof strata of the panel apparently bridging over from one barrier pillar to another, and the surface sinking being confined to the interior of the panel.

In another place, where the method of working was panel by pillar-and-room and where good barrier pillars were left around the panels but the room pillars were mined, and the coal lay horizontally at a depth of about 500 ft., with a thickness of 8 to 10 ft., the roof broke and the surface sank to a total depth of about 3 ft., but the sinking did not extend beyond the panel, the roof apparently bridging over from one barrier pillar to another, the sinking of the surface being confined to the panel and being greatest at the middle of the panel.

In the longwall fields, the method of mining being longwall advancing, the mine workings have passed under fairly good sized cities, canals, railroads and navigable rivers, and no great damage has been done to buildings and waterways; in fact, some old citizens of the towns overlying the longwall mines do not believe that subsidence of the surface is a result of longwall mining as practiced in northern Illinois. They cannot notice the subsidence of the surface and do not notice any effects of subsidence on surface buildings and improvements; hence, they do not believe that it exists.

When buildings are damaged as a result of adjacent or underlying excavations, whether in connection with railroad or subway construction or as a result of mining operations, there is extended discussion of the cost and the amount of the resulting injury, generally with a view to allocating the responsibility and pressing claims for payment. The writer is of the opinion that in many cases the damage resulting from subsidence over mining excavations is less than that commonly occurring from other causes. Some investigations were therefore made of certain residences, public buildings, grain elevators and other brick and tile structures in cities, which had been damaged because of foundation settlement, improper drainage, or other causes having no relation whatever to mining operations.

One cylindrical hollow-tile grain elevator was cracked at the base from the surface to a height of 5 ft., but no other cracks were observable. The top of the elevator, about 90 ft. above the surface, was visibly out of plumb, but although the interior of the broken tile was visible in the crack at the base, the structure was evidently still stable and was in use, although on a main street in an important city.

A certain public building, erected at least 75 years ago, of brick construction, four stories in height, was critically examined. This building rested on a limestone ledge, except at one corner, where an old sewer ran close to the foundation; the walls at this point had settled and a crack had extended about two-thirds of the way up the height of the wall from the ground. At one place, near the crack, one end of a window-sill was 1½ in. out of level, but the sash frame had been releveled and no subsequent movement had been noticed, although the crack occurred over 40 years ago. The building was condemned when the cracking occurred but was continued in use and no trouble had been experienced.

At another old brick one-story building, formerly used as a brewery, a brick chimney 30 ft. in height was very much out of plumb, though smoke was issuing from it and it had been in service for some time. An investigation developed that the walls had separated, horizontally, at a crack, for a distance of 3½ in. Apparently there was very much more movement horizontally than vertically. In fact, there seems to be a direct relation between the height of the brick buildings and the amount of horizontal displacement at the cracks, the one-story buildings separating at least three times as much horizontally, at the cracks, as three- and four-story brick buildings.

Small residences of brick and stucco construction, were inspected when cracks appeared in the outside walls and the veranda floors began to sink. Several of these buildings were damaged seriously; in fact, more than the brick buildings studied which overlay longwall mine workings. In all cases it was found that the walls were too thin, in most instances 9 in. instead of 13 in., and had been laid on clay instead of more solid ground. Poorly designed rain spouts had permitted the rain water from the roofs of these houses to fall against the walls of the houses and the water had percolated down to the clay and had washed it away from under the wall and subsidence had resulted. The municipal engineer of the city, when questioned regarding the cases of damage to residences, declared that his greatest trouble was to get the engineers and architects to leave sufficient footing for the residences they designed.

The foregoing shows that buildings may be kept in use, apparently without complaint and often without particular notice, when injured by foundation settlement or otherwise to an extent which, if occurring as a result of subsidence from underlying mining operations, would have resulted in criticism, complaint, and substantial damage claims.

ACKNOWLEDGMENTS

The author desires to acknowledge his indebtedness to George M. Brown,[1] of McAlester, for the use of information from reports relating to the Adamson and Henryetta subsidence investigations; also to C. E. Schurch, of Henryetta, and J. S. Cameron, mining engineer, Crowe Coal Co., Henryetta, for maps of Mine No. 2; and to J. W. Koster, formerly junior mining engineer, U. S. Bureau of Mines, McAlester, who placed on the map much of the data regarding the squeeze at Mine No. 1.

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[1] A Sudden Squeeze in an Oklahoma Mine. *Coal Age* (1914) **6**, 533.

DISCUSSION

(This also includes the discussion of the papers of George S. Rice, J. Parke Channing, and Howard N. Eavenson)

Benjamin F. Tillson, Franklin, N. J.—The cracks caused by caving operations, at a depth of approximately 300 ft., in limestone overlying the orebody at Franklin, N. J., are nearly vertical, but so far the operations have not been deep enough to furnish any evidence bearing on the general problem. In controlled caving, with the supporting ground gradually withdrawn, it is possible to cause an alternation of stresses that will break up masses of rock weighing hundreds of thousands of tons in such a way as to prevent serious injury to workers or workings through sudden shock.

H. G. Moulton, New York, N. Y.—The cracks resulting from subsidence at Miami appear to fall into two general groups. A series of large cracks accompanied by extensive subsidence occur at an angle of approximately 70° from the nearest underground excavation. The flatter cracks were, as a rule, later ones and were accompanied by little or no settlement. As the cracks shown in the foundation of No. 4 hoist were small and the movement indicated was horizontal rather than vertical, there might be two phases of subsidence, one a main fracture and extensive settlement, and the other a later and lesser settlement followed by minor cracks at a flatter angle. Is the relation between the subsiding volume and the excavated volume maintained irrespective of the depth?

F. W. McNair, Houghton, Mich.—The effect of the dome theory depends altogether on the type of material caved. The idea that the packing might result in the increase of subsidence as it gained depth is right; but if the material is fine, if the fracture is fine, there is no depth relation. The character of material in all types of subsidence is a controlling factor in the angle and character of break.

William Kelly, Vulcan, Mich.—When opening up a new and unexpected orebody in Menominee, some years ago, we had a proposition rather different from that of Mr. Eavenson. The dip of the orebody was between 70° and vertical, and the ore was struck at a depth of about 1000 ft. The mining system was by rooms and pillars, filling the rooms, and then removing the pillars in vertical slices, and filling. More or less settlement occurred as the workings extended upward, and we were undercutting our workings all the time. Our conclusion was that in this case the best way to mine was to start at the top and work down.

J. Parke Channing, New York, N. Y.—Mr. Kelly's remarks indicate that in mining a vertical orebody it is not safe to mine the bottom first; it always costs more and there is more ore lost.

George S. Rice, Washington, D. C.—H. I. Smith, the mining supervisor of the leased lands on the public domain, writes me as follows regarding Mr. Eavenson's paper:

"The Utah Fuel Co. was undecided as to which would be the more difficult in the case of two thick beds of coal with an intervening stratum of 20 to 30 ft., to work the lower bed after the upper had been worked or to work the lower bed first. But it was not thought possible to work more than one bed. I had occasion to sample a mine in the central Pennsylvania field in which one bed had been mined some years previously and which was then mining a bed probably 3½ ft. thick 20 ft. above it. In places, the coal had separated from the hard sandstone roof and several large cracks extended through floor and roof; at other places the floor had separated from the coal. But these did not appear to form any serious obstacles to the recovery of the coal.

"My most intimate association with working an upper bed of coal after the lower one had been worked and abandoned was in Indiana where the No. 5 bed overlay the No. 4 bed by approximately 30 to 40 ft. When laying out the work on the upper bed, no consideration was taken of the workings on the lower bed, so the entries in the upper bed paralleled at some distance the entries in the lower bed, but some distance to one side; the entries in the upper bed crossed at right angles the chain pillar of a pair of room entries. The room pillars, which were originally very narrow, were largely recovered but no attempt was made to recover the chain pillars of the room entries. This resulted in a very heavy grade as the main entries in the No. 5 bed crossed over the chain pillar in the No. 4 bed, there being an equal dip on the far side. The foreman and miners did not know that the coal beneath had been worked out until after much of the area had been worked. The roof of the upper bed was bad in places but was not considered, by the miners, an unusual condition in this bed of coal. However, there were evidences in places, where the roof fell, that the condition was caused by the bending movement from the removal of the coal beneath. Cracks on the surface over this area indicated that there was an intervening area between the lower workings and the surface where a large portion of the cracks were taken up in bending and not in separation.

"My opinon, based on observation, is that where coal has been removed from the lower bed a certain amount of breaking and filling has occurred immediately over the bed extracted, but the radii are increased at a higher point to such an extent that the rocks can adjust themselves to a bending movement without serious breaking. On approaching the surface, the bending has been transformed to tension, which probably results in considerable shattering of the rocks near the surface.

"Where the overlying strata are laminated sandstones or shales, the amount of bending will undoubtedly greatly exceed the amount of bending that can be anticipated in the mass of sandstone and limestone. Very thin strata of soft material, which becomes a semifluid under extreme pressures, may seriously affect the mining conditions by its tendency to flow under varying conditions of pressure and may be a more predominating factor in the conditions of the immediate roof or floor than great thickness of flexible or unflexible strata associated with the coal measures.

"An attempt in Indiana to mine an upper bed two years in advance of working the lower bed showed that the time period was not the proper consideration; but that the method of mining was the prime thing to be considered and that complete settlement of the overlying or underlying strata must be obtained before the coal above or below is mined. In the case here cited, it was not customary to mine coal pillars; but it was planned to get approximately 72 per cent. of the coal on first mining, leaving room pillars 6 ft. or less in width. These pillars sometimes stand for years; at other times they cave shortly after being abandoned. In this instance, a panel in the upper bed had been worked out and abandoned, while in the panel beneath the rooms were in various stages of being driven, varying from the first room having been completed to room necks just being turned. The upper panel started to cave and within two days after the closing of the rooms and entries in the upper bed the squeeze, if it might be called such, worked downward and closed the rooms and entries in the panel beneath. In the operating of a single bed, the entries would not have been affected by any caving in the rooms.

"I cannot entirely agree with Mr. Eavenson's first conclusion in regard to mining coal successfully when the distance between the beds is 19 ft. or more, as he makes no reference to the thickness of coal that can be mined under these conditions. The falls in pillar areas over thick coal beds, if the intervening strata were sandstone, would undoubtedly work into and break up the overlying coal bed beyond recovery. So some deduction should be made as to the relation of the thickness of the coal bed to the intervening strata.

"With the second conclusion I agree, at least temporarily. I concur with the third conclusion, with the addition that pillar mining should not be done in an upper bed when mining is going on in the lower. The fourth conclusion must be taken conditionally. There are undoubtedly many places where it has not been practicable to reopen an upper bed after the lower bed has been mined. The Utah Fuel Co. lost over 100 acres of good coking coal in the Sunnyside mine by mining operations beneath."

The thickness of the lower bed is not mentioned by Mr. Smith, but I think he refers to a bed which is from 8 to 20 ft. thick, depending on the locality. The problem of mining beds above and below another, previously or simultaneously mined, is one which very much concerns the Government at this time, because the coal is being leased on the public domain and not sold under the Act of February, 1920. It is assumed from the terms of the leasing act that Congress, in changing the method of disposing of the mineral lands, did it in part to advance the conservation program. Hence, the best practice in mining the coal is expected of lessees by the Interior Department.

Coal has been so abundant in the far West that there has not, in the past, been due care taken to recover all the coal, or even a large portion of it. The tendency has been to mine the cheaply obtained coal along the outcrop and not attempt deep mining as long as outcrop coal was available, also to mine only the most cheaply mined bed.

The operating regulations formulated by the Bureau of Mines for the Department of the Interior had to be framed before any leasing was done. In a measure, that hastened the decision as to what the regulations should be, and did not give the opportunity of building up the code gradually and amending it as the need developed, but the general provisions of the regulations were discussed and approved by a number of prominent western mine operators who responded to a general invitation to a conference called by the Interior Department.

Touching on the point regarding the mining of a lower bed before an upper bed has been mined, the regulations provide:

> *Section* 94.—(*a*) The lessee, where more than one bed of coal is known to exist in the leased lands, shall not draw or remove the pillars in any lower bed before mining the available coal in each known upper bed of such thickness that it can be mined under the then existing commercial conditions either alone or in combination with the thicker beds, unless it shall be decided by the mining supervisor that the workings or conditions for subsequent mining in any or all of the upper beds will not be seriously injured by the extraction of the pillar coal in the lower workings.
>
> (*b*) Where mining operations are in a bed that lies either below or above another bed in which mining has been or is being carried on, and where the vertical distance between the two beds is less than fifteen times the thickness of the lower of the two beds, as far as practicable the lessee shall, if the room-and-pillar system is employed, so arrange the pillars of room and the panel pillars that those in the lower bed shall be vertically beneath those in the upper bed or vice versa; modifications of this provision may be necessary in steeply dipping beds. Where the longwall method is employed and the goave or excavation properly packed, this provision is void.

In Section 95, also, there is a specification as to the thickness of the pillars that must be left in mining for certain depths of overburden.

At the time this part of the regulations was being formulated and discussed by mine operators and mining engineers of the Bureau, I had in mind the difficulties that had arisen in the Pennsylvania anthracite district. When mining was started there, operators did not think there would be caving, as the roof was very strong, and complete extraction was not considered necessary. So the earlier maps show no attempt to columnize the pillars; and frequently pillars rested on spans, which ultimately gave way. This caused the collieries to columnize the pillars as far as is practicable.

You will notice the stress I have laid on the method of mining. From my observations in working longwall in northern Illinois and Colorado, and also from my observations abroad, especially in Scotland, I believe that where the longwall method is adopted and the workings are well packed, there is so little damage done to the overlying beds—they come down so evenly—there is no real damage ensuing from the prior mining of the lower bed. In fact, those persons with whom I discussed the question in England thought that, in certain cases, where the coal was hard, prior mining rather helped because it loosened the coal so that they did not have to use explosives.

In 1911, I visited a mine near Edinburgh, Scotland, where they were mining a bituminous coal, about 3 ft. thick, which had been left from a former mining of a layer of cannel coal forming the bottom bench. The lower portion of the bed had been mined years before. The only effects that could be seen were flattened pieces of the old props and timbers intermingled with the clay; the old fractures had been sealed tight.

Second in importance is the character of the formation, particularly of the underclay, in considering what ultimate damage is done to a coal bed by the prior mining of a lower bed.

One of the most interesting cases of the effect of a thick, soft, underlying shale or clay is at Wemyss, north from Edinburgh, where there is a multiple bed mined by panel longwall. This particular coal absorbs oxygen rapidly and is subject to spontaneous fires, so they tried many methods of mining before adopting this one, which seems to give security from the fires, or reasonably so. The pillars surrounding the panel were left with only a few openings arranged to be quickly stopped up in order to shut up the panel in case a fire did start. However, no fires had occurred since adopting this system. They formerly occurred very frequently in room-and-pillar workings.

The bed, pitching 15 or 20°, was mined inside each panel by four successive slices from the bottom up, each slice with its face parallel with the strike, starting at the lower side of the panel. By the time the face of the first slice had advanced up the raise to the line set for the panel boundary pillar, the natural squeezing of the soft clay floor gradually lifted the trackway to about the right height to begin again at the bottom of the panel for the next slice. Meantime the roof in the roadway was brushed. Three slices were thus taken up and then the few feet of the remaining roof coal, or as much as could be secured, was taken on the retreat.

The bed was 24 ft. thick. There were some slate partings, which gave a certain amount of filling for the pack walls; cogs or cribs, also, were used for roof support. It was a utilization of natural conditions, *viz.*, a thick, relatively soft clay floor, which raised up under the overburden pressure, so the successive slices one after the other could be made. Therefore, in mining operations, as concerns the factors in causing damage to overlying coal, I think the method of mining is all-important, and, second, the nature of the strata.

Eli T. Connor, Scranton, Pa.—I had occasion to investigate carefully the effect on overlying measures of a complete crush in the Dunmore bed in Scranton, 750 ft. below the surface. It had been mined, recovering about 66⅔ per cent., by the room-and-pillar method when a crush started, which completely closed the workings. It was overlain by nine beds of coal, four of which had been mined 7 years before, and had been closed by squeezes. The uppermost beds, the 5-ft. and 4-ft. were, respectively, 50 and 70 ft. below the surface and had been mined by the room-and-pillar method.

The subsidence caused by the crush of the pillars in the lower bed of the Dunmore, caused the subsidence on the surface of from 1½ to 2 ft., but with no noticeable effect in the two upper beds, a very careful inspection of all the workings in the upper beds indicating that the settling of the superincumbent mass overlying the bottom of Dunmore bed had been gradual, but complete. The only noticeable effect on the surface was where reservation pillars had been left for the support of buildings, or for other purposes; around such pillars the strata cracked slightly at the surface, but the subsidence of 1½ to 2 ft. did no material damage to the surface structures.

From that observation, I do not think any general rule would be determined, and I agree with Mr. Rice that the character of the strata must be taken into consideration.

Regarding Mr. Eavenson's paper as to the effect on an overlying seam of mining a lower bed, our experience in the anthracite region seems to indicate that it is safe to mine completely a lower bed without appreciable damage to an upper one, provided the upper one is solid. If the upper bed has been mined by the room-and-pillar method, there is likely to be some distortion, which may cause increased expense to recover the remaining pillars of coal.

I am particularly interested in backfilling methods, which were quite largely adopted in the anthracite regions of Pennsylvania 30 or 40 years ago. But backfilling has not been carried to its logical conclusion. German engineers visited the anthracite region in the early nineties and observed the methods of backfilling by hydraulic methods, adopted the practice, and have since obtained far better results than have been attained in the anthracite region.

I visited a mine at Essen, one of the Krupp mines, where the early mining was by room-and-pillar method, extracting five or six of the uppermost beds and found that they had backfilled, with waste material, the chambers, extracted the pillars, and then backfilled the borders of the uppermost five or six beds. Last summer, they were mining a bed 5 or 6 ft. thick, between 500 and 700 ft. below the surface, hanging about 15° pitch, by the longwall method, the longwall face advancing on the strike line. They were using chutes to carry the coal to the car along the face, 350 ft. in length. After every second cut, they backfilled the space with waste material so that the subsiding roof came down gradually upon the filled material, caused some pressure on the face that helped the coal to come down after having been undercut by machine, and causing practically no effect upon the overlying strata.

I tried to find out what the French and German engineers had done in the matter of determining definitely how much subsidence reached the surface, but was told that they had not run definite levels over the surface, but they estimated that under a part of Essen the surface had subsided about 8 ft. after ten or twelve beds of coal had been extracted, the aggregate thickness of the coal being about 40 ft. They said that there was no noticeable effect upon the surface structure.

I visited two or three mines at St. Etienne, France, where they were mining a bed of coal from 20 to 25 ft. thick, which stands on a 25° pitch. They have mined it by the slicing method, from the outcrop downward, and backfilled the spaces from which the coal was extracted. A shaft is sunk deep enough to establish a level with about 250 to 300 ft. on the pitch. Gangways are driven in the top rock, from which they drive the tunnel back to the floor, or foot wall, of the coal bed, and then a small plane to the rise until the filled material is reached. From that plane, gangways are driven right and left, following the foot wall to the limit set for the panel or to the filled material on that side; then from these right and left gangways, a short chamber is driven to the roof and to the bottom. That makes an opening about 8 or 10 ft. wide, and 50 or 60 ft. long. That block of coal is taken out and the space immediately filled. The operation is then repeated.

In one mine, by that method they had extracted all the coal from the outcrop downward for 600 ft.; the workings were about 650 ft. below the surface. The filled material compresses in about the same proportion as was determined by the U. S. Bureau of Mines, and by William Griffith and myself.

At St. Etienne, the compression under the town was about 15 per cent. of the height of the coal bed, and that compression is almost exactly reflected to the surface. The lesson that I learned is that the backfilling of mine operations by suitable material ought to permit the recovery of almost any bed of coal under highly improved surface improvements without serious damage thereto.

George S. Rice.—I found some interesting records bearing on this subject, in Upper Silesia at a coal mine where they were using granulated slag for filling. The coal bed is about 20 ft. thick and the level records were very remarkable. The maximum subsidence was about 3 in. Granulated slag probably re-cements, so there would not be as much compression as in other materials.

Eli T. Conner.—At the Krupp works, I was given a paper showing the pressure of various materials as experienced in mining practice. According to this paper, tests show that granulated slag is not as strong as sand. Sand, properly introduced, will carry more weight with less compression than any other material that they have tested, which confirms the tests made by the U. S. Bureau of Mines, and by Mr. Griffith and myself.

Thomas H. Claggett, Bluefield, W. Va.—I believe the damage to an under seam can be minimized by systematic pillar drawing in one direction. The greatest damage to the upper seam is usually where the first break occurs, due to the pillar mining in the lower seam. Subsequently, with systematic pillar mining, the roof will bend, instead of break, and the cracks will open and close successively. There is little damage after that in the upper seam.

H. G. Moulton.—Mr. Eavenson has given a careful study of what is going to be in the future one of the most important problems in coal mining; that is, the manner in which several seams overlying one another may be removed. That problem has two aspects, one sociological, the other commercial. Heretofore the amount of coal available in the country has been so great, in proportion to the requirements, that the problem was mainly one of getting production quickly at the lowest cost. Now when coal is becoming scarcer and more valuable, the question of obtaining more coal is important, not only from the Government standpoint of leasing the coal on public lands, but also in connection with the conservation of coal.

Howard N. Eavenson.—When writing this paper, I did not intend to make the point that it would be necessary, or even advisable, to start at the bottom seam, but that in a great many cases more coal is lost by being left in the pillars, on account of mining by pillars, than if the bottom seam were entirely worked out.

As far as the first conclusion is concerned, Mr. Smith might possibly have better modified his statement by making the thickness of the seam 9 or 10 ft. I think the condition he mentioned in Utah was very little different. Mr. Paul told me that the case in Sunnyside was different. I was told by some of the operating officials that there was an interval of

10 to 20 ft. where the bottom seam had been worked. It ranged from 9 to 16 ft.

The point that Mr. Claggett brought out is an important one, and was partly what I had in mind when I said the more complete the extraction the less is the likelihood of trouble. That should be modified by saying, the more complete and more uniformly it was done, possibly the better it was done in the upper seam.

J. Parke Channing.—In the discussion, reference has been made to rooms and pillars, in which it was stated that it was desirable to have rooms over rooms and pillars over pillars. In Michigan, about 35 years ago, when two large iron orebodies were being mined by room-and-pillar, no attention was paid in one of the mines to the relative positions of the rooms on successive levels. In the other mine great care was taken that pillars should come under pillars and rooms under rooms. In the first mine, there was taken out by room work about 40 per cent. of the ore. The pillars then caved and the material was drawn as completely as possible. About half of the remaining 60 per cent. was recovered, making a total extraction of 70 per cent.

In the other mine, in which the pillars were very carefully laid out, the extraction eventually was much better than it was in the mine in which columnizing of pillars had been neglected.

H. G. Moulton.—Mr. Eavenson's conclusions would lead to the thought that longwall mining with complete removal does less damage to the overlying seam than the room-and-pillar method.

The commercial aspect has a bearing on operating, and also on financial, problems. The location of shafts and other underground workings and the method of mining will be affected by the consideration of the possibility of recovering coal from overlying or underlying seams. From the financial standpoint, an engineer valuing a coal property must take into consideration the probable amount of coal that can be recovered, and the percentage of recovery often depends on the possibility of working overlying or underlying seams, and the costs of such operation.

In the past it was advisable to include, for valuation purposes, only the recoverable coal from the principal seam; there will come a time when substantially all coal will be recovered. We are now in a transition period, and this paper furnishes material that will assist the engineer in valuing a property carrying more than one seam of coal within the same area.

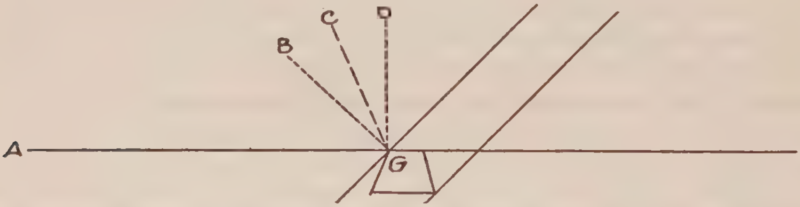
Henry H. Otto, Lansford, Pa. (written discussion).—A seam with a slate or a free-falling roof can, as a rule, be mined and robbed more successfully than one that has a hard sandstone or conglomerate roof. On the other hand, a hard sandstone roof over a seam that is being disturbed as a result of mining in a lower seam will make it possible for the seam to be mined if the disturbance is not too great.

In one case a seam, 11 ft. thick with a slate roof breaking up at least 10 ft. above the seam, dropped a virgin seam about 80 ft. above it. The upper seam was 5 ft. thick, had a hard sandstone roof, and the coal was very hard. The affected area of the upper seam covered about 30 acres and was pulled 18 in. from the roof. The pitch of the seam was about 8°. Less trouble was experienced in mining the disturbed area than the rest of the seam, and considerably less explosives were used. The roof was so hard that after a large area was robbed, it was necessary to shoot it down.

In mining a seam varying in thickness from 30 to 48 in. and lying about 10 to 20 ft. above a 6 to 8-ft. seam that had squeezed, the roof was broken in many places; the greatest difficulty encountered was the continued squeezing on the main transportation roads. About 50 per cent. of the coal was being recovered in first mining. It is doubtful if robbing can be conducted under these conditions, principally because the irregular mining of the lower seam made it impossible to columnize the workings. The lower seam squeezed when the first mining had removed about 62 per cent. of the coal. The squeeze was, in all probability, caused by geological conditions peculiar to that colliery and adjoining ones, which did not warrant so high extraction in first mining. In both of the foregoing cases the surface was disturbed; in the first instance, the depth was about 450 ft., and in the second, it was about 200 ft.

In heavy pitches, seams with hard sandstone roofs can be more successfully mined when caved by lower seams, than can one with a falling roof. In the southern anthracite fields, particularly in the upper levels, the Mammoth vein was mined and robbed before the thinner overlying seams were touched. Unless the pitch was very steep, the robbing was not done nearly as well as is being done today. The Primrose seam in many cases was mined and robbed before the overlying Orchard was mined. In one case, mining of the Primrose was well under way before the Orchard gangway was started. When the Primrose was about 75 per cent. mined, it collapsed and closed the Orchard vein about 260 ft. above. The Primrose was 12 ft. thick, and the Orchard about 8 ft.; the pitch of the veins was 45°. In another instance, the Mammoth vein on 68°, about 40 ft. thick, was mined and robbed when mining in the Primrose was started. The Primrose was virgin, had a slate roof, was 8 ft. thick and lay 300 ft. above the Mammoth. Difficulty was experienced in mining the Primrose and eventually the gangway was closed by a squeeze brought by the continued settlement from the lower seam.

One method of protecting transportation roads in a case like this is to drive the gangway in the rock underneath the seam and rock-hole back to the seam; the breast is then driven in the usual way. A good recovery can be had if the gangway does not close, which rarely happens to rock gangways. If the roof over the Primrose is slate and falls readily, it is necessary to drive chutes instead of full width breasts; this adds to the cost of mining. This condition usually prevails in the southern anthracite district where Mammoth is mined first. Because of the great thickness of the Mammoth vein, it has been the custom, in sections of the anthracite field, to drive gangways in the Skidmore vein, which as a rule carries a good top. If it does not carry a good top or is too far away or too close to the Mammoth, rock gangways are driven. In a number of instances, the Skidmore gangways held up well until robbing of the Mammoth was well under way, when the weight of the Mammoth robbing crushed the rock over the Skidmore and closed the gangway.



B–G PERPENDICULAR TO STRIKE LINE, D–G PLUMB LINE, C–G LINE OF BREAK CAUSED BY MINING DOWN TO GANGWAY G.

All seams, as far as practicable, should be mined in their proper sequence, but no hard and fast rule can be laid down as there are too many variables entering into each colliery's economic geology. It is essential, before recommending a change in mining methods, to know thoroughly the local conditions. From observations of breaking lines in the anthracite fields, it is important to know local conditions relative to the fracture lines of the various rocks and surface features. In the northern field, where the rock has not been shattered, the surface will break almost vertically, approximating a ½ to 1 slope for flat seams. In the eastern and western middle, the breaking line will approximately follow a line made by bisecting the angle formed by erecting a perpendicular to the line of strike, and dropping a plumb line to the same line of strike, as shown in the accompanying sketch.

In the southern field, around Lansford and vicinity where the geological conditions are much different from the other fields, the line of break is between 30° and 35° from the horizontal. This seems to hold for all veins from 10 ft. up to 40 ft. or more, and at various depths. This again emphasizes the importance of studying the peculiarities of each region.

Where the robbing is done in a methodical and orderly manner, clean breaks through the overlying seams will be made and the rest of the seam will go down and cause little damage to them if the rock interval is thick enough to avoid the vein falling too quickly after each breast is mined and before robbing is well under way.

As good recovery is not obtained under the best conditions of mining an overlying seam, after the lower one has been robbed, as can be obtained if the seams are mined in proper sequence, neither is it advisable to mine the upper seam while robbing is in progress in the lower. The proper interval depends entirely on the thickness of the underlying seam and the character of its roof. It is difficult to lay down a rule to fit the various conditions.

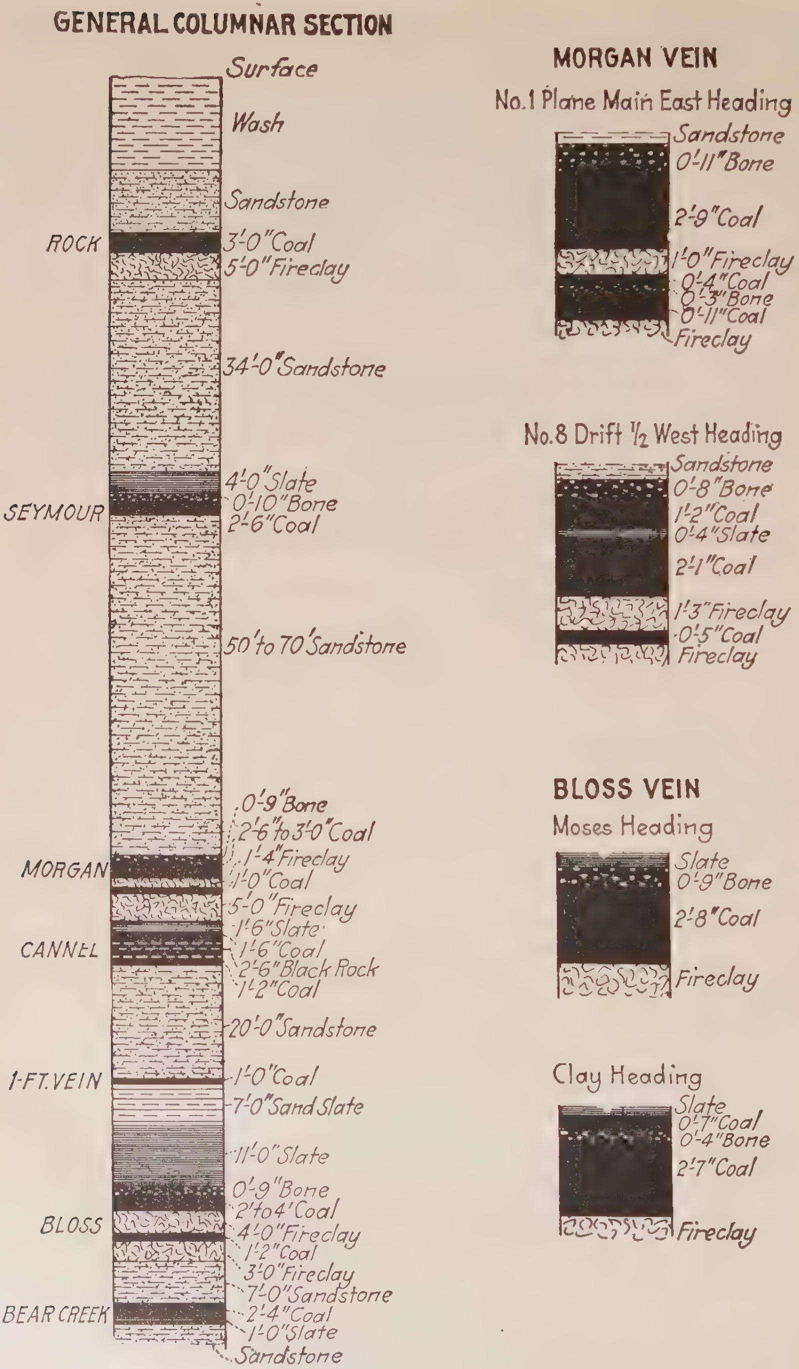
Douglas Bunting, Wilkes-Barre, Pa.—At the operations of the Morris Run Coal Mining Co., in Tioga County, Pa., the measures contain several seams of nearly the same thickness, the principal seams being the Bloss, Morgan and Seymour. The Bloss seam, being of the best quality, was mined first. The columnar section of the measures shows that the seams vary from 3 to 5 ft. in thickness and the interval between the Bloss and Morgan seams is normally 50 feet.

The mining in the Bloss seam, within the area under consideration, was started about 60 years ago and had been completed and the workings caved 10 to 20 years prior to starting the Morgan seam mining in 1910. The percentage of recovery from the Bloss seam is undeterminable, but it is presumed that the recovery was comparatively low. Two drift headings in the Bloss seam have been maintained through this area; the effects on one of these over which robbing in the Morgan seam has progressed will be later explained.

The total area of Morgan seam mined embraces 427 acres, of which 63 acres have been robbed. The depth of the Morgan seam below the surface varies from 25 to 175 ft., with an average depth of about 100 ft. The rooms are driven 42 ft. wide on 72-ft. centers and the headings and back headings on 50-ft. centers. A typical plan of the workings is shown.

The theoretical yield from these workings would be 62 per cent. on first mining, whereas the actual yield based on tonnage produced was 60 per cent. from first mining and 25 per cent. from second mining, or an apparent loss of 15 per cent., which is largely accounted for in the faulted and roll areas that are prevalent in this seam.

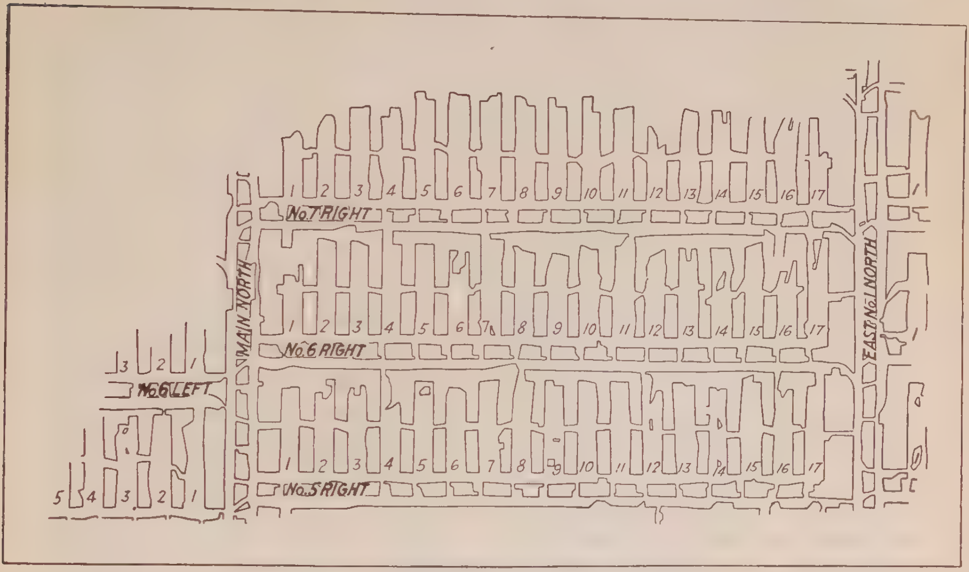
No serious difficulties have been encountered in the mining of this seam, with no material increase of cost or danger, and a good recovery on first and second mining. There has been no attempt to keep the headings or rooms over those in the Bloss seam. The most general effect of the Bloss mining on the Morgan seam was a separation of the seam from the sandstone roof varying from 1 to 15 in. The sandstone roof was cracked in places, but not sufficiently broken up to cause trouble. Some of the conditions found in the Morgan seam are shown. The



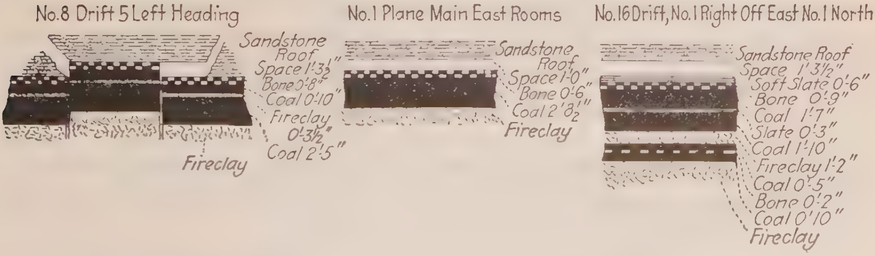
Vein sections at Morris Run.

condition of broken sandstone roof is probably due to a pillar or lack of uniformity of extraction in the Bloss seam. There is no doubt that one of the conditions most favorable to mining the Morgan seam was its strong overlying sandstone.

In the case of a Central Pennsylvania mine, cited by Mr. Eavenson, where the C′ seam had been mined and the E seam is being successfully mined whereas the D seam could not be mined on account of damage from mining in the C′ seam, it would seem that conditions favorable to mining the E seam were the overlying sandstone and the large interval above the C′ seam. The conditions unfavorable to mining the D seam would be its close proximity to the C′ seam, absence of sandstone contiguous to the seam, and possibly irregular robbing in the C′ seam.



NORTHWEST DISTRICT OF THE MORGAN VEIN.



SOME OF THE CONDITIONS MET IN THE MORGAN VEIN AT MORRIS RUN.

It appears that the leading factors incident to the succesesful mining of an overlying seam following the removal of an underlying seam are the following:

1. Thickness of underlying seam.
2. Thickness and nature of strata overlying the upper seam and intervening the two seams.
3. Percentage and uniformity of extraction in the lower seam.

F. E. Zerbey, Wilkes-Barre, Pa.—What area was worked out under the Morgan vein? To what extent was the Morgan vein affected by the worked-out area? Would the conditions be better under a slate roof than under a sand rock?

Theo. A. Schmidt, Morris Run, Pa.—With the exception of a few haulage roads and a waterway, the entire area in the Bloss vein, under the Morgan vein, has been worked out.

The effects of the caves in the Bloss vein were evident in many places in the Morgan vein in several ways. The largest area where the vein had parted from the roof was about ½ acre, which showed a drop of 15 in.; other places varied in area from 100 ft. square down to a width of 20 ft. Where the area was small, the space between the top of the vein and the roof was small, being about 1 in. At another place on a left heading in No. 8 drift, the vein dipped about 12 per cent. for 50 ft. then continued flat; on this grade the sandstone roof was badly broken and the loose rock had to be taken down. The removal of this rock assisted materially in reducing the grade for haulage. The sandstone roof was cracked in many places, but seldom broken into small pieces that would cause trouble. At another place on the main north in No. 8 drift, the roof and bottom are 6 in. nearer each other than they were 6 years ago, but the past 8 months has shown no movement. At another place, a person on the heading could see a light in the back heading by looking across the top of the pillar; in this case we placed a gob wall on top of the pillar and plastered over it to keep the ventilation to the face. Only in wet seasons did we have to contend with any water; then it was handled from one pool to another and finally found its way to the caved area below.

Conditions could not be better in our case. When the left headings in No. 8 drift reached the clay belt, a robbing plan was made and the line of break was set at 45° with the rooms and headings. Robbing was carried on uninterruptedly, but the roof paid no attention to our lines or plans; occasionally a large sand rock would drop when it was released on all sides. Sometimes the roof and bottom would get a few inches nearer each other during an idle day; sometimes the topping would have to be taken from a loaded car to get the car out of the room. The roof never sagged to the extent of losing a pillar. All the robbing on the left side of No. 8 drift was brought back to the third pillar from the main north heading. A recent inspection of this area showed that the pillars were in good condition; and beyond the third pillar the roof and bottom are still from 3 to 4 ft. apart on the headings. This is no doubt due to our method of driving the headings, which are 21 ft. wide with 8 ft. of roadway, and the other 13 ft. gobbed to the roof.

F. E. Zerby.—My experience has been that a better extraction was obtained where we had a slate roof; the roof would break back of the pillar and would not squeeze over any pillar.

Theo. A. Schmidt.—That is, no doubt, true where there is a good slate roof under a heavy cover; but in our case the cover over the robbed area was from 40 to 60 ft. I do not consider that any comparison with the cover you have in the anthracite district with deep wash. We have areas, over an acre in extent, where there is about 6 to 14 in. of slate between the top of the vein and the sandstone; where this slate was effected by the squeeze, it broke into small pieces and added considerable refuse to the loose coal. I feel satisfied that it is only because of the sandstone roof that we were able to get an extraction of over 90 per cent.

Mr. Smith, Wilkes-Barre, Pa.—What effect did the robbing in the Morgan vein have on the surface and the working of the upper vein?

Theo. A. Schmidt.—The surface over the robbed area is unimproved land; part of it is a 10-acre swamp. The only visible effect on the surface was near this swamp, where a settlement of a few inches was noticeable with crevices 2 or 3 in. wide. This drained most of the swamp, but we never noticed it in the mine. During a wet season, a stream about 3 in. in diameter came over the face of the coal in one of the east headings; this came from a creek 50 ft. ahead of the heading and over 30 ft. of cover. To stop this, a furrow was plowed from higher ground and the stream sent over a new bed. The coal is now robbed and we have had no trouble since.

We are taking no chances in regard to the effect on the upper vein. The robbing schedule planned in the Morgan vein permits no robbing under the area of the Seymour or upper seam. There has been trouble on the haulage road in the Bloss seam because of robbing in the Morgan seam. About 30 or 40 acres in the Morgan seam with the double-track haulage-road running under the entire length were robbed with no signs of a squeeze for years. When the robbing on the right side was brought to a point over the haulage road, the pillars along this road started to chip. Eight 10 by 10-ft. dry-wall pillars were erected in the affected area in the Bloss seam with good results. Five months of idle time followed this period; shortly after the second month the squeeze started north of the dry-wall pillars and before it was stopped, several hundred tons of slate roof had fallen. Subsequent to that experience, all robbing over the haulage road was stopped.

Summary of Discussion of Papers by Messrs. Rutledge and Rice*

Both of these papers were fully illustrated by lantern slides. Mr. Rutledge gave illustrations of a high-school building over longwall mines in Illinois, under which all of the coal had been extracted, the longwall faces meeting beneath the building. The building settled 1.9 ft. but without interruption to the school work; $500 covered all of the damage to the interior of the building. This was shown as a contrast to the building discussed in Mr. Rutledge's paper, where only 60 per cent. of the coal had been removed by the room-and-pillar method and yet the building had been practically demolished.

Mr. Rice pointed out that an interesting feature of the case cited by Mr. Rutledge was the extent of the draw over the solid coal with the room-and-pillar method of mining, this being contrary to the typical draw experienced in connection with room-and-pillar methods as set forth in his own paper. He said that the natural arching of the roof between the pillars probably developed first, but that the final cause of the damage to the building and extent of the draw over the solid coal was due to the complete collapse of the pillars as a result of overloading. He stated that the relation of pillar to rooms as used near the outcrop had been carried on the dip without increasing the size of the pillars or doing any packing, so that at a vertical depth of 700 ft. there was a pressure of approximately 3000 lb. per sq. in. on the coal in the pillars, which was in excess of the strength of the average bituminous coal.

Past-President Ludlow said that he had found it possible to remove all of the coal from flat veins 4 to 5 ft. thick without damage to overlying buildings; that damage was done not by the amount of settlement but by irregularity of settlement, and that either complete packing or complete removal would result in less damage than mining by the room-and-pillar system without extracting the pillars. He also said that the lateral movement was more dangerous than the vertical movement. He cited a case in which the excavation of the coal from a nearly vertical seam of anthracite had damaged a school house nearly 1200 ft. distant. The movement that damaged the building resulted from the inclined, unsupported strata tipping into the open vein, the resulting pull opening crevices in the ground which had cracked the building. The movement was stopped by filling the upper part of the mined-out vein, thus preventing further tipping of the strata.

Prof. R. M. Raymond had observed that subsidence over metal-mining operations in wide, steeply inclined veins resulted in cracks parallel with the veins and 400 to 500 ft. from the outcrop. The action in this case appeared to be the result of failure under shear, breaking a triangular section from the unsupported hanging wall and letting it settle into the excavated area of the vein.

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* Summary prepared by the Ground Movement and Subsidence Committee.

Dr. Fred. W. McNair, Mr. Rutledge, and Mr. Rice discussed the relation of subsidence to the rate of advance of the face in longwall mining operations, the result of the discussion indicating that there are not sufficient data available as yet to determine any definite relationship.

William Kelly stated that the break varied with the character of the overlying rocks, citing cases in which the fracture changed its direction and inclination with changes in the composition of overlying measures.

Dr. McNair discussed the mechanics of subsidence, stating that there had not been much laboratory work done in this connection. He thought that the subject was broad enough and interesting enough to justify definite, controlled experiments on a laboratory scale.

Mr. Rice said that tests made by the Bureau of Mines on the crushing strength of specimens of iron ore and of coal showed that the average unit strength was less in the larger specimens than in the smaller ones; with small cubes of coal the ultimate compressive strength had ranged from 2500 to 3000 lb. per sq. in., and in the largest cubes down to 900 lb. per sq. in. An application of the figures obtained by testing coal, to pillars in place in the mine, indicated that the pillars were much stronger than the tests on samples of the coal would have indicated, as mine pillars are nearly always much wider than their height, whereas the testing of the compressive strength of specimens in the past has usually been done on cubes. Tests made by the Bureau of Standards on slab specimens, in which the width was greater than the height, showed an ultimate unit strength much greater than that shown by the cube-shaped specimens.

Edward O'Toole stated that failure in a pillar shows first at the top, tending to break so as to form a pyramid at the upper part of the pillar. He described mining operations in Kentucky, underlying a mountain overlying the excavated area, the excavation being 2600 ft. vertically below the surface. At this depth the pressures were very great, resulting in so-called "bumps," and showers of coal from the ribs and slate from the roof. The practice at this property was to drive an advance crosscut to relieve the pressure. In his opinion, the topography affects the location of breaks on the surface and the draw resulting from subsidence sometimes brings into the excavation water from streams, the channels of which are in advance of the vertical plane of the face. Speaking of the unusual types of pressure and subsidence occurring in connection with mine operations at great depth, he said that, in his opinion, these varied with the character and stratification of the overlying measures in such a way as to indicate that they were often the result of geological conditions, that is, the effect of strains resulting from stresses arising from the formation, and that the topography of the country at times played an important part in the development of such stresses.

H. G. Moulton stated that a similar action was apparent in connection with the steam-shovel operations of Chino Copper Co., at Santa Rita, N. M. The Chino deposit consists of a number of orebodies surrounding a central core of granite. The slope of the excavation nearest the central core tended to slide on a much flatter plane than the one that proved a safe slope on the outer side, indicating the possible existence of some overthrust or geological stress in the central core, which was released by the removal of the weight of the ore on the flanks, with resulting movement along relatively flat slopes.

H. A. Buehler spoke of a subsidence occurring over a coal seam 400 ft. below the surface. The seam was 3 ft. thick, and 250 ft. above it there was a limestone quarry worked by steam shovels, with about 150 ft. of cover over the limestone operations. The excavated part of the coal seam was packed with gob and the overlying strata subsided 18 in. The subsidence extended up to the surface for a considerable distance in advance of the face. One noticeable feature was the fulcrum effect of the subsidence. Several monuments that had been established 200 to 300 ft. in advance of the coal face rose as much as ¼ to ½ inch.

R. Dawson Hall said that such occurrences were not uncommon, being the natural result of the mechanics of subsidence. The strata overlying coal seams in general formed a monolith, and did not act as separate, noncontinuous beams; the draw is nothing more than the result of the tension due to the bending over the top of the pillars, and that as a result of this tension the strata might fracture in advance of the face and by reason of the bending the strata might be lifted over the coal pillar. The falling of the strata in broken masses is only a secondary action and the formation of domes reaching to the surface takes place only where the measures are sufficiently shallow; such action is, as a rule, due to vertical shear and not to the bending-moment stresses set up in strata. The roof may be stronger instead of weaker after breaking, in that a weak, beamlike structure has been replaced by an arching structure. After speaking of the conditions in India, he called attention to the fact that no other country has had as much trouble from air blasts, which result from sudden falls occurring as a result of great weakness in vertical shear; this causes the roof to break straight down without developing any strength in tension and compression.

Howard N. Eavenson and H. G. Moulton called attention to the existence of numerous faults in the strata overlying some of the Indian properties under discussion, which might have a bearing on the problem of air blasts. Mr. Eavenson said that investigations of subsidence in certain Indian properties showed results contrary to all European experience, in that the area of subsidence is always less than the area underneath.

The chairman then brought before the meeting the suggestions made in Mr. Rice's paper, to the effect that much might be gained by a systematic collection of data on subsidence through questionnaires sent out by the Committee to members of the Institute. It was the sense of the meeting that much could be accomplished in this way; the chairman therefore appointed a subcommittee, consisting of George S. Rice, chairman, Howard N. Eavenson for the coal-mining industry, J. Parke Channing for underground metal-mining operations, and Louis S. Cates, for open-cut mining. This subcommittee was asked to prepare questionnaires and advise further as to the nature of the information required and the best method and time for collecting it.

As George S. Rice expected to leave for Europe in May, the chairman requested Mr. Rice, who had already been designated by the President of the Institute to act as its representative at the Sixth International Mining Exhibition in London, to take up with the English and Scotch mining societies the question of coöperation with the American Institute of Mining and Metallurgical Engineers in the study of ground movement and subsidence. The appointment of Mr. Rice for this purpose was later confirmed by the Board of Directors.

Mine-drainage Stream Pollution

By Andrew B. Crichton, Johnstown, Pa.

(New York Meeting, February, 1923)

No more important question has come before the coal industry in the past decade than the prevention of stream pollution by mine drainage; especially in Pennsylvania, where large areas of coal land have been developed. It may be thought that the problem of labor or lack of transportation outweighs in importance that of stream pollution. We can see possible remedies for labor and transportation problems but there is no known satisfactory solution of the mine-drainage problem. The subject has been given little serious attention, for, until recently, it has been little understood; but in the past year it was brought to the attention of the coal-mining industry by the introduction, in Congress, of several bills designed to prevent the discharge of acid waters from mines or other sources into navigable streams, or their tributaries.

The first legislation proposed was the Appleby bill, H.R. 7369, designed particularly to prevent the oil pollution of coastal waters. It contained the following provisions, which the coal men regarded as referring to the coal industry.

> It shall be unlawful to throw . . . any refuse matter of any kind . . . into any navigable waters of the United States or into any tributaries of any navigable waters from which the same shall float or be washed into such navigable waters.

The chemical, steel, paper, and other manufacturing industries, as well as the mining industry, objected to this clause, the miners claiming that it would prevent the discharge of waste or mine drainage into any waters of the United States, for all streams or tributaries finally reach navigable waters.

The next bill introduced was that of Congressman Briggs, H.R. 7430, an amendment to Section 13 of the Pollution Act of March 3, 1899. The original act referred chiefly to oil and oil refuse, but it provided also "that no material of any kind should be deposited on the bank of a navigable water, or tributary, which might reach the waters and interfere with or obstruct navigation." This last provision could be said to apply to mine refuse, which would produce highly acid drainage.

The Rosenbloom bill, H.R. 8733, also an amendment to the act of March 3, 1899, provides that it shall not be lawful to deposit or discharge from "any source whatever any free acid or acid waste, or anything that may become acid after being deposited into navigable waters or tributaries of the United States." Under this bill, Army engineers may prohibit the discharge or may regulate the entrance of wastes or may require the treatment of such discharge to destroy its acidity. This bill aims at the mine drainage entering the Ohio river and its tributaries. The first draft was proposed by Major-General Lansing H. Beach, chief engineer, United States Army. General Beach, in his report, states that the treatment of acid mine waters is so simple and inexpensive a matter that he believes there will be no opposition on the part of the owners of mines concerned. He suggests that lime be deposited in a small box, to be built at each mine opening, through which the mine water would flow, thus neutralizing the acid.

J. T. Travers, supervisor of streams, fish and game division, state of Ohio, in the same report states that the equipment will cost about $65 for each opening and the supplies from 75 c. to $1 per day. It developed, however, at the hearing held in Washington, that the Travers system was applicable only to manufacturing plants discharging 3000 to 7000 gal. per day. Mr. Travers thinks that coal-mine drainage is not so bad as that from manufacturing operations and that it would be comparatively simple and inexpensive to treat mine drainage. He apparently has not considered seriously the statements of mining engineers as to the enormous quantities of acid mine water daily flowing from the coal mines of the country.

There would be no opposition to treatment in this manner, if the problem were as simple and inexpensive as indicated in the foregoing paragraphs.

The Chemical Alliance, in a report made at the request of the chief engineers of the United States Army, stated that 6500 tons of sulfuric acid is discharged into the Ohio River and its tributaries every day, or 2,357,500 tons per annum or nearly half as much as is now produced in the United States. Neutralizing this amount of acid would require 3250 tons of lime per day, or 1,170,000 tons per annum.

Charles Dorrance, vice-president of the Hudson Coal Co., whose 30 mines produce 9,000,000 tons of anthracite annually, has stated that these mines pumped 15 tons of water for each ton of coal produced; and that neutralization of the water of the anthracite region, basing the estimate on the cost of this company, would require an initial expenditure for installation of $39,000,000 and the annual operating expense would be about $18,000,000. He estimated that the mine drainage from the fields of the anthracite region was 700,000 gal. per day; and that neutralization would add to the consumer's bill about 50 c. per ton.

P. C. Madeira, of the Anthracite Coal Operators' Association, estimated that an average of 18 tons of water was hoisted for each ton of coal.

The Act of 1899 placed the matter in the hands of the army engineers, but at the congressional hearing the question was referred to the Department of Commerce for investigation and report; thus jurisdiction is rather confused at the start. In Pennsylvania, Governor Pinchot has promised legislation to prevent stream pollution. Various congressional candidates have based their claim for election on this issue.

Effect of Pollution on Water Supply

The question of pollution is squarely before the industry. It is important, it is serious, and the difficulties of the situation should be understood by the country and more reliable data should be obtained on which to base proper legislation. Otherwise laws will be enacted that will become a burden to industries and the people as a whole. Already stream pollution has been a cause of inconvenience and expense to both mining companies and to the public. Until rather recently, stream pollution has been a mining-community problem. If a water supply was destroyed through mine drainage it was customary to go farther afield for a fresh supply. Attempts have been made and much money spent in the building of treatment plants, which have been unsatisfactory and later have been abandoned. Many communities have had their water systems polluted and have gone farther up stream, at great expense, for fresh supplies. Johnstown, Latrobe, and McKeesport have had to abandon their original sources of supply. Barnesboro, Altoona, Connellsville, and many other Pennsylvania towns have acid water during certain periods of the year and eventually must seek new sources.

Nearly all of the important streams of central and western Pennsylvania used 20 years ago for water supply are now seriously contaminated. Pittsburgh, in the center of one of the greatest mining and manufacturing districts of the world, is a particularly glaring example of a city suffering from contaminated waters. As industry expands, there is a constantly growing demand for pure water for both domestic and industrial purposes; at the same time, the extensive development of coal mines in the vicinity produces more and more mining drainage. For much of the year, mine drainage is small compared to the pure water flowing into the streams above Pittsburgh, and the acid water is so diluted as to render it harmless. This period grows shorter and the period of contamination longer. The situation demands serious attention.

The headwaters of some of the streams lie beyond the coal measures, but practically all of these areas have been developed and little more fresh water can be secured from such points. Western Pennsylvania has been developed from the standpoint of mining without regard to the future water supplies.

Areas underlain by coal measures that have been developed and partly or completely mined have very little run-off except in wet weather. The broken ground allows easy penetration of surface water, which is later pumped out or drawn from the mine carrying considerable acid with it. The drought of last summer emphasized the seriousness of the problem in the Pittsburgh district.

The state and individuals have spent considerable money to solve the problem but without success. It seems time for wise and careful governmental action. Secretary Hoover suggests a careful investigation of the entire problem before the passage of any legislation whatever, but this must not be deferred too long or there will be litigation between the mining interests and those of the water works.

Supremacy of Coal Industry over Water Supply

In the past, the mining industry has been considered paramount; but while coal is a necessity, water is necessary for the industrial centers where the coal is used. Both are vital to the welfare and prosperity of the nation.

Mining has maintained its superior position to industry and the water supply of the latter partly through the decision of the Supreme Court in the Sanderson case, in 1886, which gave the Pennsylvania Coal Co. authority to conduct its work without interference by the plaintiff, who secured his water below the exit of the mine, maintaining that the inconvenience to the plaintiff was not so serious as the closing of the mine would have been. At the same time, however, the Court pointed out that the case "may arise in which such pollution may become a nuisance, and public interest, as involved in the general health and well-being of the community, may require the abatement of the nuisance."

There is now pending in the Court of Equity, Fayette County, Pennsylvania, an important case involving stream pollution between the Mountain Water Supply Co., Dunbar Water Supply Co., and the Pennsylvania Railroad vs. about 30 operating coal companies.

The Pennsylvania Railroad, about 40 years ago, gave up its water supply at Portage because of mine-drainage pollution. Several million dollars were spent to secure an adequate supply for the future needs of the company and Indian Creek basin was developed to supply the southwestern portion of this system. The drainage area of this development is 110 sq. mi., half of which is underlain by commercial coal measures, none of which was developed at that time. Since then, however, railroads have tapped this section, mines have been opened, and the water supply polluted. It has rendered the situation serious. Very careful study was given to all factors entering into the problem: methods of mining, thickness of seams, drawing of pillars, pitch of the measures, the relation and quantity of water pumped to production of coal, the relation of rainfall to underground water, etc.

Extent of Pollution from Coal Mines

It developed that percolation of water bore no relation to the quantity of coal produced but a very direct relation to the extent of development. It depends directly on the extent of the surface that has been disturbed and the decrease in the run-off owing to the resultant broken ground; for each acre of coal mined much more than an acre of surface becomes disturbed and subject to increased penetration by surface waters.

The records of 170 mines in central Pennsylvania showed that for each acre worked out, water is developed to the extent of 100 to 10,000 gal. per day, the average being about 1100 gal. In other districts of the state, the average was not far from this amount. Shallow workings developed considerable more than the deeper workings. Many mines produced their maximum only when pillars are drawn, resulting in some surface subsidence.

Except with shallow workings, mine drainage flows are quite constant. They do not vary much with rainfall; consequently, in time of drought, the percentage of acid in the streams is materially increased. In one area, where about half the coal had been removed and where the average depth was about 500 ft., and the rainfall 3300 gal. per acre per day, it was found that 27 per cent. of this water percolated into the mine and was pumped out as acid water. The minimum flow was only 17 per cent. less than the average of the year. These figures are not far from the average percolation in other districts, where records have been carefully kept.

The quality of mine drainage is quite as important as the quantity. In practically all the coal mines, the drainage contains sulfuric acid. Of over 300 mines examined in central Pennsylvania, in only four the mine water was not acid and in these the freedom from acid was probably because limestone was in contact with the coal seams. While nearly all seams of coal contain iron pyrites, in the thin seams of central and western Pennsylvania there is a bone coal carrying a high percentage of sulfur at the top of the main seam, a large portion of which is thrown into the gob, but large quantities are hauled outside and dumped in the valleys. This refuse material, both outside and inside of the mine, oxidizes rapidly and furnishes a large amount of acid. The sulfuric acid forms salts as it comes into contact with the soluble minerals present and these salts, in turn, on exposure are precipitated, forming insoluble coatings, and permitting the free acid and salts to flow farther down the stream.

Table 1.—*Typical Analyses of Mine Waters, Grains per U. S. Gal., by E. C. Trax*

	Miller Shaft	Puritan Shaft	Sonman Shaft	Berwind No. 35	Yellow Run Shaft	Morrellville	Argyle	Blair	Howard	Rogers	Oneida
Total solids	266.01	330.18	276.50	146.42	135.33	608.82	135.88	266.17	218.54	175.53	329.48
Probable incrustants	190.15	247.59	200.67	96.26	75.84	368.13	70.59	85.85			
Suspended matter	7.29	9.92	3.21	10.50	8.75	1.20	4.14				
Iron oxide	21.56	40.10	29.37	9.22	10.10	128.11	10.27	......	27.32	28.88	55.19
Aluminum oxide	12.90	6.73	9.14	3.38	4.38	38.21	3.73	......	5.43	11.45	8.72
Calcium oxide	30.55	27.17	20.90	16.47	9.60	20.76	13.13	7.14	29.47	11.40	27.62
Magnesium oxide	9.05	16.25	12.01	9.77	5.66	15.63	4.79	5.57	12.73	5.49	15.33
Sulfuric anhydride, SO_3	129.40	157.52	130.84	72.74	57.50	295.64	70.93	141.89	85.79	55.17	111.95
Sulfuric anhydride (acid sulfate)	20.82	37.33	37.68	10.04	11.67	87.06	21.88	16.47			
Free sulfuric acid H_2SO_4	34.42	61.25	45.50	23.62	28.00	247.06	26.47	115.00	25.04	41.53	50.31
Chlorine	0.70	0.82	0.58	0.47	0.76	2.27	2.70	0.29			
Treatment Required, Pounds per 1000 Gal.											
Lime	7.30	13.06	10.79	5.31	5.00	34.95	5.80				
Soda ash	19.52	29.20	23.78	13.01	11.01	68.90	14.18				

Some Pittsburgh seam mines in the Greensburg district of Pennsylvania show free sulfuric acid as follows:

	Grains per Gallon
Keystone shaft	98.42
Sewickley	61.74
Greensburg Nos. 2 and 3	121.68
Seaboard shaft	137.68
Claridge	169.40
Salem	91.20
Crows Nest	123.96

The average total acidity at sixty mines of the H. C. Frick Coke Co. was 100 gr. per U. S. gal. Analyses of the waters of some of the polluted streams in Pennsylvania at much above minimum flows, are given in Table 2:

TABLE 2.—*Analyses of Polluted Streams*

	Sulfuric Anhydride, SO_3	Sulfuric Anhydride, Acid Sulfate	Free Sulfuric Acid, H_2SO_4
West Branch Susquehanna River at Moss Creek	44.40	5.90	10.60
Quemahoning, at dam	4.20	0.40	0.50
Conemaugh, at Portage	35.23	6.80	12.40
Conemaugh, at South Fork	24.25	5.40	9.70
Shade Creek, at mouth	4.72	1.00	1.40
Paint Creek, at mouth	37.70	7.40	14.10
Red Stone Creek, at mouth	32.89	6.77	9.33
Jacobs Creek, at mouth	19.49	3.83	8.03
Sewickley Creek, at Hunker	40.27	9.80	16.33
Turtle Creek, at Wilmerding	48.40	16.33	8.75

TREATMENT OF ACID WATER

In treating acid waters, the custom is to neutralize it with lime (converting it into hard water, that is, converting the acid into calcium and magnesium sulfates) then adding soda ash to soften it. The total amount of foreign matter in solution is about the same as before softening. It is the total amount of sulfates rather than the free acid which governs the method of treatment. Water with 4 gr. of sulfates per gallon requires treatment of the same kind, and when pollution exceeds 12 gr. per gal., the amount of soda ash required causes foaming. Treated water is not free from harmful effects, the presence of salts still causing pitting. The human system could stand more than 12 gr. of sulfates per gallon, but it cannot be said to be a healthful water that is contaminated to this extent.

Basing an estimate on the area of coal mined throughout western Pennsylvania, Maryland, and West Virginia, tributary to the headwaters of the Ohio River, and using the average percolation of such mine areas, it is estimated that there is an excess of 5 gr. per gal. of sulfate in the river water, which requires softening before using. To this, however, must be added the acid derived from refuse dumps. In this district, there are over 7000 sq. mi. underlain with coal, only 400 of which has yet been mined. If the mining of 400 sq. mi. produces a water of the foregoing character, what must be in store for the rivers when ten or twenty times this area has been mined? In the past, the acid waters have been largely neutralized by the natural alkalinity of the fresh waters, but once the acid exceeds the neutral point, the depreciation in the quality of the water will be markedly increased and out of proportion to the area of coal development. Fig. 1 shows the coal fields of Western Pennsylvania and parts of Maryland and West Virginia; it also shows the part underlain with coal; in solid lines the streams now acid; and in dotted lines streams that will probably become acid with continued mining.

In addition to the acid of the mines, large quantities enter the rivers from various industrial plants. If all this were neutralized with lime, the amount of sludge resulting would be very large and the arrangements necessary to keep it from the streams would be no small item of expense.

With this general statement of the problem, what are the suggested remedies? Water softening plants will help postpone the evil day, but

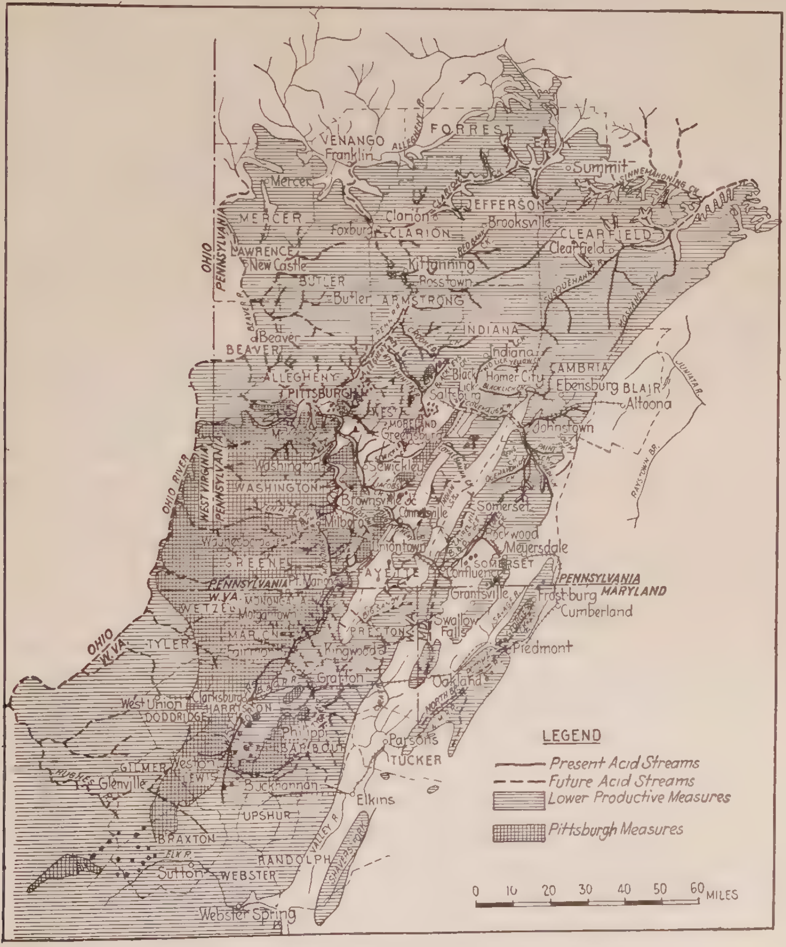

FIG. 1.—PRESENT AND FUTURE ACID STREAMS.

the step from 4 gr. of sulfate to 12 gr., when softening is no longer effective, is short and other remedies must be looked for.

What shall be done with abandoned mines that are constantly throwing quantities of acid drainage into the streams? It may be in time that all pyrite will be exhausted and water again flow pure. One case has been reported where the mine ceased to produce acid water after it had been abandoned 25 years. It does not appear as though we could wait for time to purify the water. Sealing the abandoned mines has been suggested, but this is impracticable in fractured ground. Where the ground can be thoroughly sealed, keeping out air and keeping the pyrites under water, oxidation will certainly be diminished. This plan holds some hope. It has been suggested that mine drainage be conducted from the mines in non-destructible pipes to points on water courses, below which they are not used for manufacturing purposes. As a practical proposition, this is difficult, because the drainage deposits sediment that would tend to clog any system of piping that could be used. An open drainage or concrete duct would offer a better medium than any piping.

In the west, barium has been used, quite successfully, to soften water, but its cost is excessive and water so treated is not suitable for domestic purposes.

Evaporation is the only really satisfactory method of converting contaminated waters into something that can be used, but the cost is prohibitive.

The building of large storage reservoirs for impounding fresh water, for the purpose of dilution has been proposed, but this would be expensive and of only temporary benefit. In the past the problem has been solved by going farther up stream to unpolluted sources, but in many instances no such sources are now available. The water companies have asked that coal mining cease in certain areas, and this the courts still have under advisement. It is certain that coal-mine development cannot be carried on without destroying the water of the immediate vicinity. It would, therefore, seem that about the only means of conserving the water supply is to stop coal mining in those little-developed areas which still contain an ample supply of fresh water. This cannot well be done as between individuals. Some coöperation must be established between state and federal governments and the owners of property. The state of Pennsylvania now recognizes the importance of the problem and is participating in the actions before the courts in that state.

The State Board of Health in Pennsylvania reports 96 water companies with acid-pollution problems. This problem is connected and might well be considered with the matter of the overdevelopment of the coal industry. The coal industry is not to blame for this condition, it is simply a development of that industry and onewhich the public must bear in inconvenience or in cost of preventing the inconvenience.

We cannot learn from Europe how this problem may be handled because the coal mines have not a similar problem. In Great Britain, many of the mines use their water for boiler purposes and without treatment. In Yorkshire, mine drainage is used as a source of domestic supply. In Great Britain, the mines are being worked at depths varying from 1000 to 4000 ft. and they have relatively little water; in some of them water must be taken underground in order to lay dust. British coal mines are not acid producing because the coal is low in sulfur and often is in contact with limestone.

In France, the mines do not produce acid water, probably because an overlying bed of chalk extends from the surface to a depth of 300 feet.

With less than one-tenth of the coal of western Pennsylvania worked out, with continued mining, as at present conducted, and new development constantly being started, it is apparent what the results to the streams will be in a comparatively short time. The mine drainage stream pollution problem is serious. Its extent is not fully realized by the industry or the public. Secretary Hoover's suggestion that the question be carefully investigated in order to determine possible remedies and legislative enactment is wise and should be heartily approved and receive the coöperation of the coal industry.

DISCUSSION

Samuel A. Taylor, Pittsburgh, Pa.—When the committee of the U. S. House of Representatives was considering three bills seeking to prevent the pollution of water along the coast, principally by the discharge of sludge by oil-burning vessels, it was found that the bills, as drawn, would affect every coal mine in the country. With others, I appeared before the committee and presented data secured from the Pittsburgh Coal Co., and some others regarding the amount of water pumped out of the mines. In the Pittsburgh district, it is necessary to handle over 7 tons of water for every ton of coal produced; in Central Pennsylvania, about 5 tons; and in West Virginia the amount is about the same.

George S. Rice, Washington, D. C.—Does the amount of water raised have a direct relation to the rainfall?

Samuel A. Taylor.—No definite relation except in mines having very shallow cover. In some of the deep mines of Illinois, and, even in the Pittsburgh district where the upper Freeport coal is mined, the amount of water pumped per ton of coal is much less than in other Pennsylvania fields. In the Appalachian region, where there are deep mines, unless the streams pass over the outcrop, there is not as much water in the deep shafts as in other fields.

E. A. Holbrook, State College, Pa.—The mine operators are accused of throwing great quantities of acid water into the streams and polluting the streams. Somewhere along these streams are one or more large municipalities that discharge their sewage into these streams. For example, the Monongahela reaches Pittsburgh charged with acid water; at Pittsburgh, an immense quantity of sewage is discharged into the river, but a few miles below Pittsburgh the water is clear and fish live there. If it were not for the acid the sewage would be found a good many miles farther down the river. Is there not a direct reaction between the sewage and the acid from the mine water?

Samuel A. Taylor.—That is absolutely correct. The mine water will neutralize the sewage and the sewage will neutralize the mine water; that has been shown in a number of places. For instance, Turtle Creek empties into the Monongahela River at Bessemer. The entire water shed of that stream is underlain with coal, and the waters from the coal mines are so strongly acid that they neutralize the sewage thrown into that stream. The borough of Pitcairn had been instructed by the state health authorties to put in a sewage-disposal plant. But after a number of samples of the water of the Turtle Creek had been analyzed, it was decided that there was no need of erecting that plant.

The mines get blamed for all of this stream pollution, but if you will study the Monongahela River, you will find the water changes materially during the year. At a number of places there are tin-plate works, the outflow from which materially increased the acidity of the stream, especially at low water. The water plants that secure their water from the Monongahela make daily analyses of their water to determine what chemicals they must put in the river water to neutralize it; this is done under the direction of the State Board of Health.

About 10 years ago the National Government sought to control the pollution of the streams in that region, claiming that the acidity of the water in the Monongahela, and that coming out of the state of New York, injured the steel vessels plying in the streams and also interfered with the operation of the canal locks.

George S. Rice.—The Bureau of Mines at that time found that about 3000 mines were discharging into the river above that point.

Samuel A. Taylor.—There were at least twice as many abandoned mines as there were mines in operation, but the water coming out of the mines in operation was very small compared to that coming out of the old mines, because practically all of the old mines were under shallow cover. But there was no one to handle this water as the owners were either dead, or had abandoned the mines.

J. W. Paul, Pittsburgh, Pa.—The United States engineers investigating the acidity of the river at Pittsburgh reported that the Monongahela is quite acid, while the Allegheny is alkaline; the Allegheny water therefore largely neutralizes the Monongahela at Pittsburgh so that 15 miles below Pittsburgh the Ohio water is practically normal.

J. J. Rutledge, Baltimore, Md.—The question of stream pollution is getting to be pretty serious along the Chesapeake, and some drastic action may be taken; at that time mine water may come in for attention.

A number of years ago Chicago and St. Louis had a dispute about the pollution of the Illinois River, as St. Louis at that time derived its water supply directly from the Mississippi. According to the chemists, who carried on those experiments for two or three years, the water was purified a surprisingly short distance down the Illinois River from Chicago.

C. M. Young, Lawrence, Kans.—This paper deals with a subject which has already drawn some public attention and the importance of which is constantly increasing. It must not be supposed, however, that the seriousness of this situation has not been recognized previously.

About eight years ago the Pittsburgh office of the U. S. Engineer Corps carried on an extensive investigation of stream pollution in that district. The Government's interest in the subject arose from the fact that the War Department has certain jurisdiction over navigable streams, and the Ohio, the Monongahela, and the Allegheny are all navigable. This jurisdiction covers not only the navigable portion of the streams but extends to the head-waters of the streams and their tributaries. I was engaged on this investigation and published an account of it.[1]

In the course of this investigation, a study was made of all of the main sources of pollution and of the effect of the different foreign substances entering the streams. It was found that by far the greatest damage was done by sulfuric acid and its salts and that most of these came from mine drainage and from the metallurgical plants of the Pittsburgh district, the mine drainage being responsible for perhaps nine-tenths of the acid pollution.

It was recommended at that time that the Rivers and Harbor Act of 1899 be so amended as to apply specifically to acids and acid salts and that action be required for the suppression of damage. In recognition of the great difficulties which would lie in the way of the sudden rigid application of such provisions, it was recommended that means be provided for the modification of the application of the law in cases where its rigid application would be an unjust burden. The foregoing statement is not a literal quotation of this report but embodies the spirit of the recommendation. It seems that no action has yet been taken.

Since the effects of acid pollution of the streams, largely due to mine water, are increasing, it seems probable that some protective action will be found necessary in the future. In my opinion it would be much better for the parties most interested, that is the coal and iron industries, to take steps toward finding a solution which will at least be tolerable.

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[1] C. M. Young: "Pollution of River Water in the Pittsburgh District." *Jnl. Am. Water Works Assoc.* (May, 1921) **8**, 201.

It is surely unwise for these interests, which will be most severely affected by prohibitive action, to be unprepared for such action. There can be no plea of ignorance of conditions and if the state of the streams so arouses public opinion as to bring about severe action it is quite possible that hardships will be brought upon these industries.

The fact that no remedy for the situation has yet been presented is not proof that no such remedy exists. The parties most concerned ought to devote earnest efforts to the search for such a remedy and not leave the matter to more or less interested government bureaus. These government agencies are more concerned with the prevention of damage than they are in finding a remedy which will be satisfactory to the coal and iron interests.

At present no organization seems to exist which can properly be charged with the investigation. I believe that the coal and iron interests ought to form a research institution, supported by contributions from the industries, to which such questions as this could be submitted with the hope that some satisfactory solution would be reached and with the knowledge that, if no solution could be reached, the fact that the solution had been attempted would be a strong argument against any severe and precipitate action.

Interpretation of Results of Coal-washing Tests*

By Thomas Fraser† and H. F. Yancey,‡ Urbana, Ill.

(New York Meeting, February, 1923)

Before a new coal-washing plant is installed, or an existing washery is remodelled and improved, considerable experimental work on the coal to be washed should be done. A thorough examination of a coal, to determine whether it can be washed successfully and the best and most economical process for securing a satisfactory grade of washed coal, calls for mine sampling, analytical work, and experimental coal-washing tests which may require several months' time and the expenditure of considerable money. Having collected the required data on the characteristics of the coal and the results secured with different types of machines under different sets of conditions, it is necessary to decide what method of treatment should be used. The interpretation of the experimental data is no small part of the problem.

The first step in the investigative work is to determine from the characteristics of the coal and the manner of occurrence of the impurities whether it is possible to separate the dirt from the coal and produce a washed coal of the required quality. Then it is necessary to select the type of washer or the system of treatment that will produce the greatest amount of washed coal of this grade for the least cost.

To secure this information, tests may be made at laboratories operated by consulting or contracting engineers, at the testing plants of machinery manufacturers, at commercial washeries, or in an experimental washery erected on the site of the proposed plant. Such tests will produce figures in which every factor, including analyses of raw coal and washed coal and yield of washed coal, may vary widely. To reduce these results to a common basis for comparison is very difficult. A number of plans have been proposed for reducing by means of charts or formulas, the results of a washing operation to a single figure which represents, as a percentage, the general efficiency of the operation, and which may be compared directly with a similarly deduced efficiency number for another operation. Another method is to study each set of results as a whole and in relation to one another and then, taking into consideration all the conditions obtaining, to decide which results are the most favorable. To do this requires a knowledge of the conditions under which the plant will be operated and the product marketed as well as a knowledge of the technical problems involved. For instance, if the investigator fails to take full account of the commercial conditions and produces in the experimental work a washed-coal product not suited to the use for which it is intended, the results will be difficult to use. In this case a figure for comparison may be secured by means of an efficiency formula but its value may be open to question. The efficiency-formula method of reducing the results of each test to a single figure, so that a direct numerical comparison may be made, is the simplest and most precise method of comparing the results of competitive tests and is preferable if the number secured by the efficiency calculation truly represents the effectiveness of the operation.

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* Prepared under coöperative agreement with the Engineering Experiment Station of the University of Illinois and the Illinois State Geological Survey Division. Published with the permission of the Director, United States Bureau of Mines.

† Assistant Mining Engineer, U. S. Bureau of Mines.

‡ Assistant Chemist, U. S. Bureau of Mines.

In the experimental work conducted, for several years, at the Urbana station of the Bureau of Mines, an attempt has been made to develop a systematic method of conducting an examination into the washability of a coal and of evaluating the results of experimental work. A careful study has been made of published efficiency formulas and some have been developed, that later have been discarded as useless. The conclusion reached is that a formula which may be applied indiscriminately to all cases and conditions to produce efficiency numbers for direct comparison is out of the question. Some of the methods that have been found useful are described in this paper and their limitations are outlined.

Difficulty in Comparing Experimental Results

The difficulty of interpreting experimental data is illustrated by the results of a series of competitive coal-washing tests shown in Table 1. These results were secured at an experimental washery where three methods of treatment were tried by large-scale tests on commercial-size machines working under operating conditions. The object was to make a two-product separation, refuse and washed coal, with no final secondary coal product, and to produce a washed coal, suitable for the manufacture of blast-furnace coke. The table gives results of one test on each machine chosen at random. None of the three methods stands out decisively as the best method for treating this coal.

TABLE 1.—*Results of Three Washing Tests on Same Coal; Values in Per Cent., Dry Basis*

Test	Raw Coal		Washed Coal			Refuse		
	Ash	Sulfur	Yield	Ash	Sulfur	Amount	Ash	Sulfur
1	12.3	2.24	91.78	8.9	1.04	8.22	51.3	15.02
2	16.9	2.59	78.00	6.1	0.90	22.00	55.9	8.48
3	13.5	2.69	88.83	6.7	0.91	11.17	65.2	17.51

In these tests, the raw coal varied greatly in quality; the feed in the second test was so much dirtier than in the others that the results are not directly comparable as regards either ash reduction or yield of washed coal. If the washed coal produced in the first test, carrying an ash content of 8.9 per cent. and a sulfur content of 1.04 per cent., is sufficiently clean, because of the higher yield the separation made in this test represents the best commercial operation. The other two processes, however, might have produced as good results if tested on a similar raw coal with the refuse discharge adjusted to produce the same quality of washed coal. One variable factor could have been eliminated and the problem simplified, to that extent at least, by making the competitive tests on duplicate raw-coal samples. This would require preliminary sampling and analysis of the lots taken for tests, and the rejection of such lots as differed materially from the average raw coal; or, better, the thorough mixing and separation into duplicate samples of one large lot selected for the tests. While the latter plan would require rehandling all the coal to be used or the installation of a mechanical sampling or distributing device, the small additional expenditure would make possible more valuable comparable results.

Unfortunately, tests are often made on samples that were not taken with sufficient care to insure that they are representative. In several cases, carload lots of raw coal have been shipped long distances for large-scale washing tests and the washed coal shipped to another location for coking tests before any analytical results have been received from the chemical laboratory. In one investigation, a car of southern Illinois screenings was shipped to another state for washing tests and the washed coal was sent to Chicago for coking tests. All the samples were then analyzed, when it was found that the raw coal had 12.4 per cent. ash and 2.36 per cent. sulfur though the screenings produced at this mine for several years had averaged 9.0 per cent. ash and 1.76 per cent. sulfur.

It has been suggested that where the proportion of dirt in the raw-coal sample used does not check with that in the raw material to be treated or where the quality of the coal subsequently changes, the results of the tests may be applied to the new conditions by figuring on about the same percentage reduction in ash and sulfur. For example, in the case of the southern Illinois coal carrying 12.4 per cent. ash and 2.36 per cent. sulfur, the washed coal was reduced to 6.4 per cent. ash and 1.87 per cent. sulfur, a 48.3 per cent. reduction in ash and a 20.8 per cent. reduction in sulfur. The same percentage reduction in the average raw coal carrying 9.0 per cent. ash and 1.76 per cent. sulfur would produce a washed coal containing about 4.7 per cent. ash and 1.39 per cent. sulfur. A study[1] of the occurrence of impurities in a number of coals

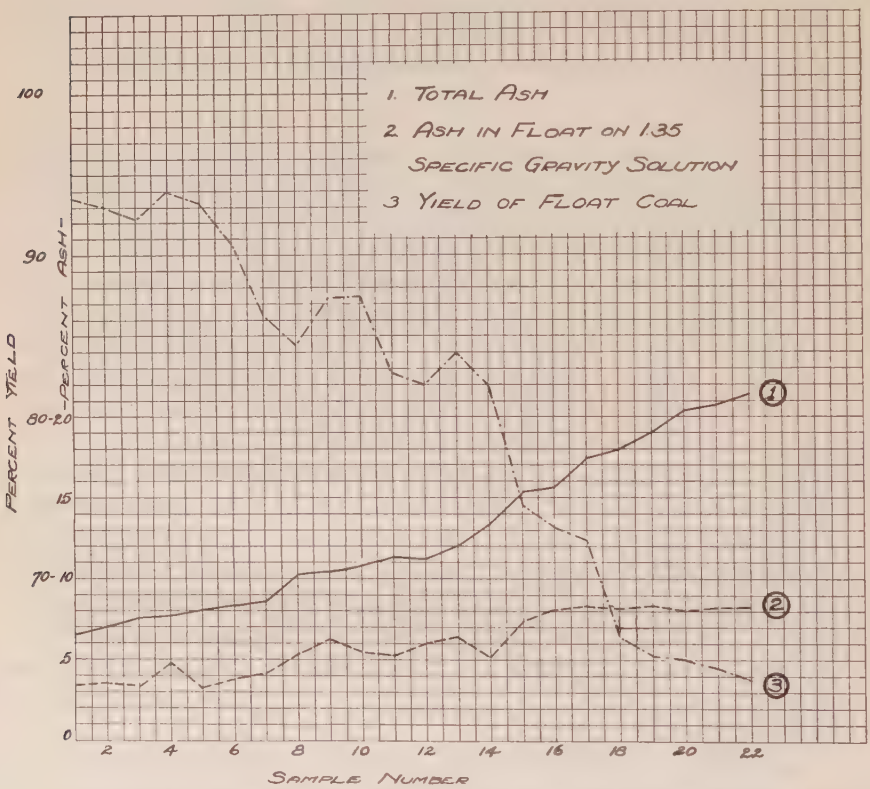

FIG. 1.—FIXED AND REMOVABLE IMPURITIES IN A NUMBER OF SAMPLES OF SAME COAL.

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[1] Distribution of Forms of Sulfur in the Coal Bed. *Bull.* 125, Eng. Exp. Sta., Univ. of Ill.

has shown that usually such a result cannot be expected. In a large number of raw-coal samples of a given coal in which the total percentage of impurities fluctuates widely, the amount of fixed impurities, that is the finely divided ash and the organic and fine disseminated sulfur, run much more uniform. The excess dirt in the samples that are far above the average in ash and sulfur, therefore, is made up largely of extraneous material in coarser particles, which are easily removable.

For a given coal, the percentage of fixed impurities, or the minimum ash and sulfur securable in the clean coal product of a sink-and-float test, holds more or less constant with, however, some increase in the very dirty samples. This condition is shown more clearly in Fig. 1. This would lead us to expect that, in washing coal carrying different percentages of impurities, the results would tend toward production of a washed coal approaching a certain minimum ash and sulfur content in every case rather than toward the removal of the same proportion of the total dirt present in the raw coal. That is, in the case of the Illinois coal mentioned, we would not look for a much cleaner washed coal on treatment of an average sample but the yield of washed coal would obviously be higher because of the smaller amount of dirt to be removed. As a matter of fact, the fixed ash in the coal of this field is ordinarily well above the figure (4.7) derived by calculating a reduction of 48.3 per cent. in ash.

Similarly, on the coal of Table 1, if the second and third tests were repeated on a feed having the same ash and sulfur content as was used for the first test, the change in the results would be expected largely in the yield figures rather than in the quality of washed coal. It is not suggested, however, that the results securable on a lot of the same coal of different ash content be estimated by assuming the production of a washed coal of the same quality and then calculating the washer loss necessary to bring about this reduction in ash. Neither this calculation nor an assumption of the same percentage reduction in ash will take the place of an actual washing test on a truly representative sample. As shown in Fig. 1, the very dirty sample may contain a somewhat larger percentage of fixed impurities than the comparatively clean sample and, in addition, the residual proportion of separable impurities left in the washed coal will probably be somewhat larger when the higher ash coal is treated. On the other hand, a larger reduction in total ash and sulfur is to be expected on the dirty coal; results of coal-washing tests and washery operations generally bear this out.

One washery operator, to determine the advisability of picking the large coal before it is crushed and washed, put both products of the picking tables through the washery and found that the same quality of washed coal was produced from the picking-table refuse as from the good coal or the mine-run coal. It is probable, however, that this kind of an operation is uncommon. Results secured by treating a large sample of picking-belt refuse to recover the pyrite and the clean coal are given in Table 2, also the normal washery operation on run-of-mine coal from the same mine. While a much larger percentage reduction in impurities was secured on the picking-belt refuse, the washed-coal product was still considerably higher in ash and sulfur than that produced from the run-of-mine coal.

TABLE 2.—*Washing Results on Picking-table Refuse and on Run-of-mine Coal from the Same Mine; Values in Per Cent., Dry Basis*

Test	Feed		Washed Coal			Reduction in	
	Ash	Sulfur	Yield	Ash	Sulfur	Ash	Sulfur
Picking-table refuse	23.8	9.30	56.8	8.9	3.40	62.6	63.4
Run-of-mine coal	9.9	2.50	85.0	7.5	1.85	24.3	25.9

In a large number of tests it was observed that samples of coal, varying over quite a wide range in ash and sulfur content and including samples of coals from different fields, showed a marked tendency to reduce, by washing, to about the same percentage of residual free impurities of the different specific gravities in the washed coal, when these samples are treated with the primary object of producing a clean coal of maximum purity rather than a high yield. Table 3 makes this relation more clear. It is particularly apparent in the case of the heaviest part of the refuse, and is much less marked in the case of the lighter fractions, which form the middling products and the bone coal. This would be expected because the coals vary widely in the amount and character

TABLE 3.—*Specific-gravity Analysis of Several Washed Coals from Tests Intended to Produce Maximum Ash Reduction; Values in Per Cent.*

Specific Gravity	Illinois, No. 6, Per Cent.	Tennessee, Bon Air, Per Cent.	Indiana, No. 4, Per Cent.	West Virginia, Eagle Seam, Per Cent.	Alabama, Mary Lee, Per Cent.
1.80 +	0.6	0.4	0.4	0.4	0.4
1.60–1.80	0.3	0.5	0.2	0.6	0.9
1.50–1.60	0.5	0.8	1.5	0.5	2.6
1.45–1.50	0.3	0.8	2.5	0.4	1.2
1.40–1.45	0.9	3.8	3.0	1.0	2.0
1.35–1.40	3.7	11.8	3.6	1.2	8.2
1.30–1.35	8.3	5.5	5.9	5.2	15.7
− 1.30	85.4	76.4	82.9	90.7	69.0

of material of this class they contain, and because this middling material is much more difficult to handle satisfactorily in the washer than is the clean heavy refuse.

This condition is pointed out as a tendency rather than a definite numerical relation and many exceptions to it, no doubt, will be found, especially in the case of very dirty or very bony coals, such as occur in the Northwest and in Alaska. Table 3 gives results secured on coals from Illinois, Indiana, West Virginia, and Alabama using jigs, concentrating tables, and pneumatic tables. These tests were made, primarily, to determine the minimum to which the ash and sulfur could be reduced. Other tests, in which ash reduction was sacrificed, in a measure, to secure a high yield would represent an entirely different set of conditions and would not be expected to check the results here given.

A specific-gravity analysis of the coal will show the variations in raw coals, so far as the density of impurities is concerned; this is the principal manner, from the washability standpoint, in which raw coals may differ. It would seem logical, therefore, that the development of some such relation as the above or a study of the data from this point of view would furnish the best method for comparing, on an efficiency basis, washing tests or washery operations on different coals and under different conditions.

While the Hancock efficiency chart,[2] originated by David Hancock of Birmingham, Ala., is a great help in studying these data, in that it shows the data in their correct proportion, it does not present any method for interpreting the data or reducing them to a specific efficiency number for direct comparison with similarly derived efficiency numbers for other tests. If a method could be devised for reducing the results of competitive tests on a given coal, such as are shown in Table 1, to one number for each test, which would represent the effectiveness of the operation, this would be the most accurate method of determining the most suitable method of treatment to use. A number of efficiency formulas have been proposed for calculating such an efficiency number.

Comparison of Results by Drakeley Efficiency Formula

A method advanced by T. J. Drakeley,[3] which has merited considerable discussion, is based entirely on specific-gravity separations secured by the float-and-sink method. The improvement in the character of the washed coal over the raw coal, with respect to the concentration of float particles in it, is termed by Drakeley the qualitative efficiency of the process. If, therefore, the washed coal consists entirely of particles that float on the test solution, the qualitative efficiency is 100 per cent. The manner of calculating the qualitative efficiency is as follows:

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[2] "Iron Making in Alabama," 3d ed., 240. Geol. Survey of Alabama.

[3] *Trans.* Inst. Min. Eng. (1917–18) **54**, 418.

$$\text{Qualitative efficiency} = \frac{\text{Washed coal float}^* - \text{Raw coal float}^*}{100 - \text{Raw coal float}}$$

By a second equation, Drakeley measures the effectiveness with regard to the recovery of float coal, by determining the amount of float in the refuse obtained, using a float-and-sink bath of the same specific gravity as was used for the raw coal and the washed coal. This factor is termed the quantitative efficiency and is derived as follows:

$$\text{Quantitative efficiency} = \frac{\text{Raw coal float}^* - (\text{Refuse float}^* \times \text{percentage of refuse})}{\text{Raw coal float}}$$

Assuming a case of perfect washing with no float coal in the refuse, the quantitative efficiency would obviously be 100 per cent. In order to obtain a single figure that represents the mechanical efficiency of the washing operation as a whole, Drakeley combines the qualitative and quantitative percentage efficiencies as a product; this is called the general efficiency of the process.

From a consideration of Drakeley's method it is evident, aside from the actual efficiency of the process, that the magnitude of the number secured depends somewhat on the specific gravity of the liquid used for the sink-and-float tests. As a result of an intensive investigation, Drakeley reached the conclusion that the specific gravity of the liquid should be 1.35. He found, by making a fractional specific-gravity separation of raw coal, using successively denser solutions and plotting the cumulative yield of float coal against the specific gravity of the fractions, that a break or decided change in direction occurred corresponding with 1.35 specific gravity. Fig. 2 shows two curves[4] illustrating this point. The marked change in direction occurs at about 71 per cent. yield of float and 1.35 specific gravity. From these data and numerous reported specific-gravity values on air-dried bituminous coal, Drakeley concludes that the float on a solution of 1.35 specific gravity may be considered, from a purely scientific standpoint, as coal; and he so defines coal. The sink in this bath is classed as dirt.

Such a definition of coal and dirt is an arbitrary one. It would appear better to select a density that gives a float coal containing the maximum allowable content of ash for the particular use to which the coal is to be put. Other considerations should lead to the adoption of a float-and-sink bath of a specific gravity depending on the requirements to be met.

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* Values used are percentages of float coal.

[4] *Coll. Guard.* (1920) **119**, 747.

In the present investigation, a large number of fractional specific-gravity separations or specific-gravity analyses have been made. Fig. 2 shows four of these curves plotted so as to make them comparable to Drakeley's graphs. One of these shows a change in direction at a point corresponding to 1.35 specific gravity, as found by Drakeley. One shows a rather marked change at 1.25, while the other two have no definite break at any point. It should be observed that Drakeley examined normal bituminous coal seams in England, and the differences found may well be explained by the differences between American and English coals. Of course, a coal containing little material of intermediate density would show a sharp change in direction. In plotting yield against specific gravity, the change in direction of any part of the curve depends entirely on the amount of the specific-gravity fraction corresponding to that part and not on its quality. Even if an abrupt change in direction does occur at 1.35 specific gravity, it is incorrect to take this as the standard of efficient washery operation if a separation at a higher specific gravity will give a satisfactory washed coal and a larger yield, The Drakeley formula may be used equally well, however, with any density of sink-and-float bath desired.

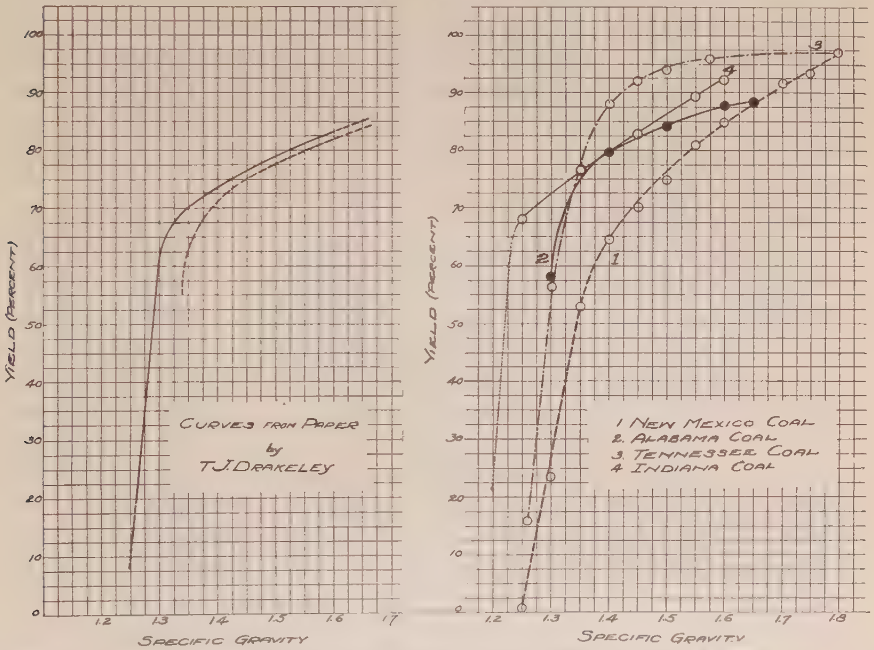

FIG. 2.—YIELD-SPECIFIC GRAVITY CURVES OF SOME ENGLISH AND AMERICAN COALS.

PROPOSED FORMULA FOR COMPARING WASHING TESTS

The qualitative efficiency factor of the Drakeley formula, being a measure of the increase in concentration of float particles in the washed coal, takes no account of the quality of the heavy material left in the washed coal. The error due to this condition is evident from the data given in Table 3, which shows that almost all of the heavy clean slate of high ash content is removed and that the sink material remaining in the washed coal consists largely of middling particles. In an operation on a bony coal showing a very low qualitative efficiency by the formula, the bulk of the sink material in the washed coal may be only slightly heavier than the sink-and-float bath and very little dirtier than the washed coal, so that although there is a large amount of it, the effect on the average ash and sulfur content of the washed coal is comparatively small. Although the separation made in a coal washer is secured by taking advantage of the difference in specific gravity of the constituents of the raw coal, the object is to reduce the ash and sulfur content and it seems logical, therefore, to judge the effectiveness of the operation on this basis.

For this reason a formula has been adopted which combines a quantitative factor, secured by comparing the actual yield of washed coal to the yield of float coal, with a qualitative factor secured by comparing the actual ash reduction to the ash reduction secured by the sink-and-float method. In addition, this method has the advantage that an ash analysis is a more precise determination than the sink-and-float test, particularly on a coal which contains considerable material only slightly different in density from the sink-and-float bath. The formula was derived by combining a factor, similar to Drakeley's expression for quantitative efficiency, with the first of a set of four equations published by G. R. Delamater,[5] or by combining as a product the first of these four equations with the last part of the fourth to form a single formula applicable under all the four conditions described in that paper. This gives the following expression for the general effectiveness of the washery operation.

$$\frac{\text{Actual yield}}{\text{Standard yield}} \times \frac{\text{Actual ash reduction}}{\text{Standard ash reduction}}$$

or

$$\frac{\text{Yield of washed coal}}{\text{Yield of float coal}} \times \frac{\text{Raw coal ash} - \text{Washed coal ash}}{\text{Raw coal ash} - \text{Float coal ash}}$$

Standard yield and standard ash reduction are taken as the yield of float coal and the ash reduction secured by a sink-and-float test on the raw coal, using a solution that will give the most profitable separation in the case under consideration. The specific gravity of the solution to be used is determined by the method described by Delamater. That is, a representative sample of the raw coal is separated into a series of different specific-gravity increments by subjecting it to a sink-and-float test, first on the lightest solution of the series, and then testing the sink product in successively heavier solutions. A study of the data secured will show what particular density of solution gives the yield and quality of washed coal that represents the most profitable operation. The commercial aspect of the problem is presented later.

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[5] Coal Washing Efficiency Calculations, *Coal Age* (May 2, 1914).

The foregoing formula has been used in the study of many operations and is thought to be the most generally applicable. It differs from the underlying principle of the method advanced by Delamater only by combining the two efficiency factors as a product rather than as a numerical average and substituting for the four formulas a single expression which eliminates some inconsistencies in the four.

The qualitative efficiency might be calculated on the basis of sulfur content but as the sulfur values are much more erratic than the ash and do not vary consistently with the specific gravity, the ash content is the more satisfactory.

A method of estimating efficiency based on a complete separation at the most profitable point is most useful in the continuous study of the operation of a given plant, in the routine control and adjustment of the plant from day to day, in the development and improvement of the plant, and in determining the effect of changes in the quality of the raw coal mined. Where we are concerned with only one operation, there is not much likelihood of a radical change in the economic conditions and there will be probably little change in the nature and occurrence of the impurities in the coal that will alter the mechanical difficulties in cleaning. There is therefore little cause to question the comparability of the efficiency figures. In continuous operation on the same coal, sufficient data will soon be accumulated by analyses for it to be possible to determine accurately and permanently the specific gravity of the solution that gives the most profitable separation. The aim will then be to operate the plant so as to duplicate that separation as nearly as possible. Calculating the efficiency numbers will then require sink-and-float tests on the raw coal only. A daily record of efficiency numbers will make it possible to compare the effectiveness of one day's operation with another's or one year's average with another's and to determine the effect of changes in the method of treatment, or of additions to the plant.

Table 4 gives the average results for the operation of two washeries

TABLE 4.—*Average Operation of Two Washeries on Coal from the Same Mine; Values in Per Cent., Dry Basis*

Type of Washery	Ash in Raw Coal	Ash in Washed Coal	Yield of Washed Coal	Efficiency at 1.50 Sp. Gr.
Old pan jig washery..........	15.92	11.90	89	70
New piston jig washery........	17.35	10.50	83	82

on coal from the same mine. The first is a Stewart washery operated some years ago and the second is a two-compartment piston jig washery now in use. During the entire period the property has been operated by the same company and the washed coal used for the same purpose, *i.e.* the manufacture of metallurgical coke for use in the company's own furnaces. The requirements and the nature of the raw coal therefore have probably not changed very much. A specific-gravity analysis of the raw coal made during the operation of the Stewart washery and one made since the erection of the new washery are given in Table 5. If the separation made at a specific gravity of 1.50 is selected as the most desirable and the yield and the ash content of float on a solution of this bensity are taken as the standard for calculating the efficiency numbers, the result is 70 for the old washery and 82 for the new plant. These efficiency numbers are of value because they give a definite numerical measure of the improvement in the operation secured by installing the new plant.

TABLE 5.—*Specific-gravity Analyses of Raw Coal for Operations Shown in Table* 4; *Values Given in Cumulative Percentages, Dry Basis*

Specific Gravity	Old Washery		New Washery	
	Yield	Ash	Yield	Ash
Float on 1.35	66.0	7.7	72.1	7.6
Float on 1.40	73.9	8.4	79.8	8.5
Float on 1.45	79.0	9.1	82.1	8.6
Float on 1.50	82.5	9.7	83.6	9.1
Float on 1.55	85.7	10.3	86.3	9.8
Float on 1.75	90.7	11.9	89.2	10.9
	100.0	15.9	100.0	17.4

The selection of the correct specific gravity for the testing solution is the most difficult step in the efficiency determination by this method. The specific gravity chosen determines largely in what portion of the percentage scale the resulting figures will come. Table 6 gives the efficiency numbers for these two operations based on the separation made on each solution used for the specific-gravity analysis. These figures show the futility of attempting to fix upon a certain density of solution that can be used for all cases. This is one of the many factors that render doubtful the value of an efficiency formula for comparing results secured at plants operating on different coals under different economic conditions, and preparing coal for different purposes. For example, a washery in the central competitive field, preparing screenings

TABLE 6.—*Efficiency Numbers for Plants Shown in Table 4 Using Float-and-sink Baths of Different Specific Gravities*

Specific Gravity of Float-and-sink Bath	New Washery	Old Washery
1.35	81	66
1.40	81	64
1.45	79	66
1.50	82	70
1.55	87	74
1.75	99	98

to be sold on the open market as steam coal, commanding only a very little better price than unwashed coal, can profitably stand but a very small washery loss. The aim must be to remove, perhaps, the conspicuous clean shale and pyrite particles in order to improve the appearance of the coal. This would probably correspond to a separation at 1.80 specific gravity or even higher. A comparison of this plant with a sink-and-float separation at 1.35 as standard gives a figure that is valueless either as an expression of the true efficiency of the operation or as a basis for comparison with other washery operations.

YIELD-CURVE METHOD OF COMPARING RESULTS

For studying the work of a plant in commercial operation by means of an efficiency calculation, it is essential that the results be compared with a complete separation between dirt and refuse at the right specific gravity because the operation is not right if this separation is not made at the most profitable point. But in conducting competitive coal-washing tests to determine what type of washer or what method of treatment will best clean the coal under consideration, the problem is to determine which process will make the sharpest and most complete separation of the heavy particles from the light particles in the raw coal. It is not so essential for the immediate purpose of the work that the separation be made at any particular specific gravity. For example, if one machine makes a complete separation at 1.80 specific gravity, assuming such a result were possible, while another makes a complete specific-gravity separation at 1.35, the latter machine will handle this cleaning problem as well as the former; although, if the average quality of the coal lighter than 1.80 specific gravity is sufficient to meet the requirements, the separation at 1.35 later in the operation of the plant would not be profitable. These two machines are equally effective mechanically except that it is easier to make a good separation at 1.80 specific gravity than at 1.35.

Although the washability of the coal and the most profitable separation to be made must be determined in the investigation of a coal to be washed, having determined the method of treatment that will make the most complete separation of the dirt from the coal, bringing the separation up to the most profitable point is largely a matter of adjustment and control of the operation. As these are two distinct problems, another method for comparing the results of competitive tests has been devised.

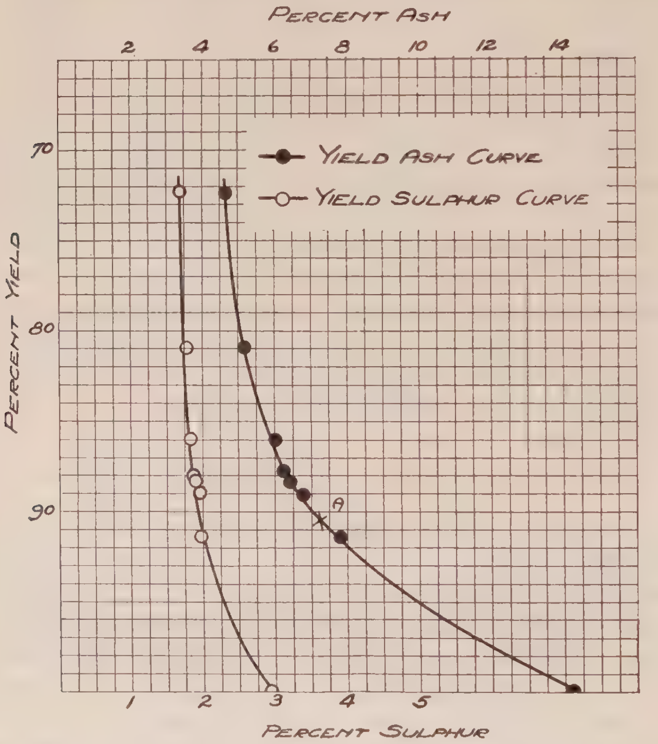

FIG. 3.—FLOAT-AND-SINK YIELD CURVES, ILLINOIS NO. 6 COAL.

Using the data secured by a specific-gravity analysis, a cumulative yield ash curve is drawn; Fig. 3 shows the curves for the specific-gravity analyses given in Table 7. The cumulative per cent. yield of float coal is plotted on the ordinate axis, beginning with the lightest fraction at the top; the average ash figures for these combined float-coal products are plotted as abscissas. From this curve may be read the yield, securable by the sink-and-float process, of coal of any desired ash content between that of the raw coal and that of the lightest fraction, which for the coal of Fig. 3 is between 4.64 and 14.20 per cent.

If a washing test is made on this coal crushed to the same size as the sample used for the specific-gravity analysis, a comparison of the actual

TABLE 7.—*Specific-gravity Analysis of Illinois No. 6 Coal*

Specific Gravity	Per Cent. of Sample		Per Cent. Ash		Per Cent. Sulfur	
	Direct	Cumulative	Direct	Cumulative	Direct	Cumulative
– 1.30	72.35	72.35	4.64	4.64	1.72	1.72
1.30–1.35	8.74	81.09	11.27	5.41	2.14	1.76
1.35–1.40	4.93	86.02	17.78	6.10	2.39	1.80
1.40–1.45	1.82	87.84	20.32	6.36	2.52	1.81
1.45–1.50	0.39	88.23	24.60	6.45	2.62	1.82
1.50–1.60	1.12	89.35	29.90	6.74	2.80	1.83
1.60–1.80	2.13	91.48	49.53	7.74	3.43	1.87
1.80	8.52	100.00	84.04	14.20	13.63	2.87

yield of washed coal with the yield of float coal of the same ash content gives a measure of the completeness of the separation. Table 8 gives the results of a washing test on the coal represented by Table 7 and Fig. 3. A 78.5 per cent. yield of washed coal of 7.2 per cent. ash was secured. This ash content corresponds to a sink-and-float separation at the point *A* on the curve showing a yield of 90.6 per cent. The ratio of these two yields, $\frac{78.5}{90.6} = 86.6$, is the efficiency number. It is a measure of the completeness of the separation at whatever point it is actually made. If the operation is correctly adjusted to the commercial conditions so that the separation approaches that selected as the most profitable, the two methods become practically identical. That is, if the specific gravity of the sink-and-float solution selected for the efficiency determination by the formula is such as to make the standard washed coal ash 7.2 per cent., the calculation becomes $\frac{7.2}{7.2} \times \frac{78.5}{90.6} = 86.6$, checking the number derived by the yield-curve method.

TABLE 8.—*Washing Test on Illinois No. 6 Coal; Values Given on Dry Basis*

Product	Weight, Pounds	Per Cent. of Raw Coal	Per Cent. Ash	Per Cent. Sulfur
1. Raw coal	2050	100.0	14.2	2.70
2. Washed coal	1610	78.5	7.2	1.85
3. Middling	160	7.8	19.8	2.56
4. 2 and 3 combined	1770	86.3	8.1	1.90
5. Refuse	194	9.5	72.1	10.75
6. Loss	86	4.2		

It is not intended that these two methods shall supplant all those previously proposed, but it is thought that they will be useful additions to the methods of investigating coal-washing problems. The determination of the washability of a coal and the most suitable type of washery is a difficult problem and one involving the expenditure of considerable money; therefore every available tool that will assist in interpreting and comparing the results of experimental work should be utilized. In reporting such investigations all the data should be included and all the conditions stated so that the methods by which the conclusions as to efficiency and relative values were arrived at may be understood.

Special Method of Comparing Table-washing Tests

To develop a method for definitely and accurately comparing the results secured in competitive washing tests on tables, the experimental quarter-size tables in the laboratory were equipped with a number of tanks, as shown in Fig. 4. For comparing competitive tests in a commercial way a few divisions with respective products made near the corner where the split between washed coal and refuse is ordinarily made is sufficient. The receiving compartments and sampling chutes on the washed-coal discharge side of the laboratory table are each 4 in. wide, and therefore receive the material discharging across a 4-in. section of the table deck; across the refuse end, 3-in. splits are made.

By this method, in tests where one of the machines used is a table, the separation between washed coal and refuse may be made between any two adjacent compartments as desired. The ash content of washed coal on the two tests may be calculated to approximately the same figure or may be interpolated from a curve to exactly the same figure; the yields are then directly comparable. The following case will illustrate this method of comparison. In a washing test on a commercial size table using a feed crushed to ¼ in. maximum size, only two products were made, washed coal and refuse. The yield of washed coal was 87.1 per cent. with an ash and sulfur content of 8.64 per cent. and 3.42 per cent., respectively. During the test, samples of the raw coal fed to the table were taken at regular intervals. The resulting lot of coal was subsequently washed on the quarter-size laboratory table, the discharge being collected in thirteen compartments.

Curve 2, Fig. 5, shows the work done on this coal using the specially equipped laboratory table. The cumulative yield of washed coal and the corresponding cumulative ash content secured by making the separation between washed coal and refuse at any given point around the discharge side and end of the table are indicated. A separation made between compartments 10 and 11 corresponds most nearly to the results secured on the commercial size table. This shows a yield of washed coal of 85.7 per cent. with an ash and sulfur content of 8.70 per cent. and 3.33 per cent., respectively, as compared with 87.1 per cent. yield and 8.64 per cent. ash and 3.42 per cent. sulfur on the commercial table. From the curves, the yield at any ash and sulfur content or the ash and sulfur at any yield, within the limits of the coal used, may be determined. It is to be concluded that the two tables used in these tests give practically the same results on the coal used. It is believed that this method of conducting competitive table tests will be useful to a washery operator wishing to select the table best adapted to cleaning his particular coal.

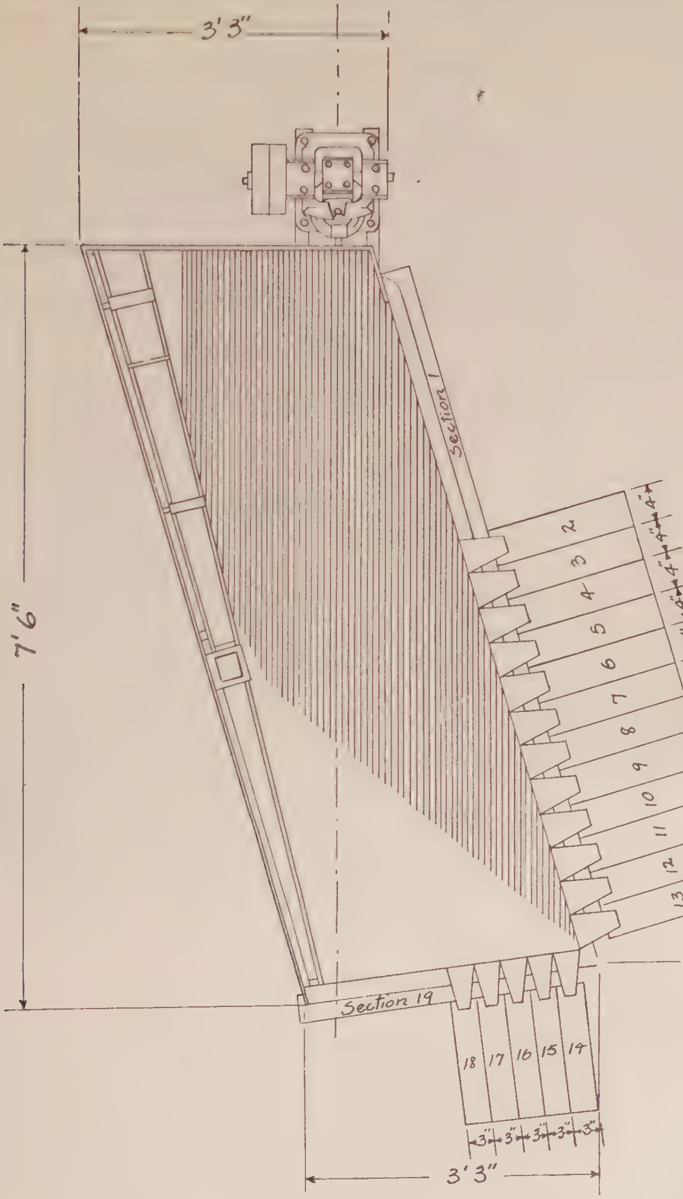

FIG. 4.—ARRANGEMENT OF DISCHARGE RECEIVING COMPARTMENTS ON COAL-WASHING TABLE LABORATORY OF DEPARTMENT OF MINING ENGINEERING, UNIVERSITY OF ILLINOIS.

A separation into seven products, requiring five compartments near the point where the split between washed coal and refuse is ordinarily made, each compartment wide enough to receive about 2 per cent. of the feed, would furnish the range of adjustment generally required. After having determined these points, only 2 per cent. apart, a curve may be drawn from which intervening points can be accurately interpolated.

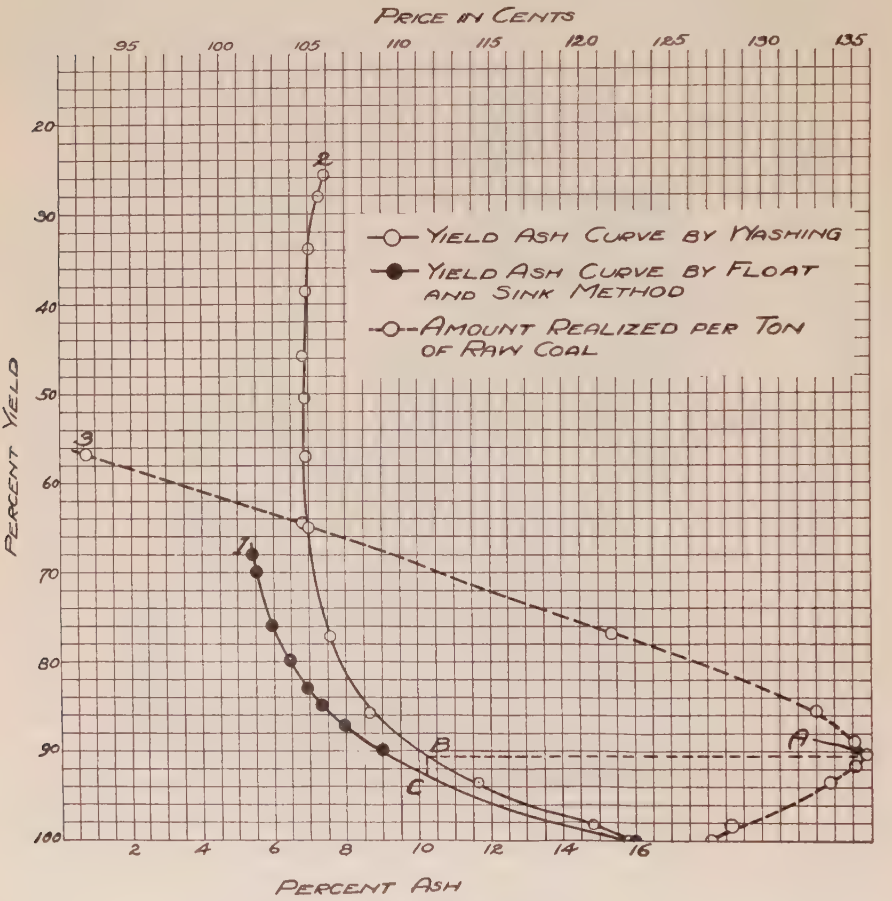

FIG. 5.—YIELD CURVES, INDIANA NO. 4 COAL.

This method of comparing results of experimental tests where one of the washers used is a table is a direct comparison of yields and ash reductions without the introduction, as a standard, of the sink-and-float test, which varies with the size to which the coal is crushed. In comparing the results of a jig test on 0 to 2-in. coal with a table test on the same coal crushed to ½ in. maximum size to determine whether or not there is an advantage in the finer crushing, a sink-and-float efficiency formula would be less satisfactory than this method which would make possible a direct and accurate comparison of yields regardless of the difference in size of the coal treated.

Economic Considerations in Interpreting Results of Experimental Coal-washing Work

The determination of the most profitable separation and the necessary adjustment of the plant to make this separation depend largely on the commercial conditions in the particular operation being examined. This is an important part of the work but one that often receives too little consideration, particularly when the experimental work is done by persons not directly connected with the operation of the property. A washed product of any given quality, within the limits of the coal, may be secured, but the lower the ash and sulfur content of the coal is reduced the smaller will be the output of washed coal. The experimental work can be conducted to much better advantage, therefore, if the engineer in charge is thoroughly familiar with the commercial conditions, the use for which the washed coal is intended, the requirements to be met, the financial advantage of reducing the ash and sulfur below these requirements, the value of the unwashed coal, etc.

These conditions will vary widely at different mines. Often where the washed coal is to be used for a specific purpose, a fixed requirement as to the ash or sulfur in the coal must be met and the operation is not credited for reduction below that requirement. In such a case, if the coal is easily washed, there may be no difficulty in making the required product with any type of washer. The whole problem then is to develop a system that will produce the maximum amount of satisfactory washed coal from a given tonnage of raw material. In other cases, where the coal is high in fixed impurities and difficult to wash, so that the cleanest coal it is possible to make is above the limits of an ideal coal for the purpose intended, the chief object is to secure the maximum reduction in impurities; the yield is of lesser importance.

For instance, an estimate of the most profitable separation on a coal to be made into metallurgical coke for use within the organization might be made in the following manner. The coal is not marketed as such and increase in selling price does not enter into the calculation, but there is an increase in value of the coal because of the decrease in coke and limestone consumption of the furnaces, increase in capacity of furnaces and ovens, higher yield of gas, etc. At different plants the estimates of this increase in value range from 10 to 50 cents per ton of pig iron per unit ash reduction in the coal. Assuming a saving in pig-iron production cost of 20 cents per ton of iron per unit ash reduction, a coke consumption of 2000 lb., and a coke yield of 70 per cent., the saving per ton of coal is 14 cents per unit ash reduction. Neglecting the saving in freight and handling charges on the coal, this 14 cents may be taken as the increase in value of a ton of coal per unit ash reduction. On this basis, the relative advantage of the two separations made in the washery operation represented by the Hancock chart, Fig. 6, may be calculated. One operation taking out the refuse represented by the black area of the chart gives a washed coal of 8.1 per cent. ash and a yield of 86.3 per cent. Removal also of the middling product represented by the shaded area gives a washed coal of 7.2 per cent. ash and a yield of 78.5 per cent. Assuming the value of the raw coal containing 14.2 per cent. ash at $1.50 per ton, the net value of the washed coal produced from a ton of raw coal in each case is as follows:

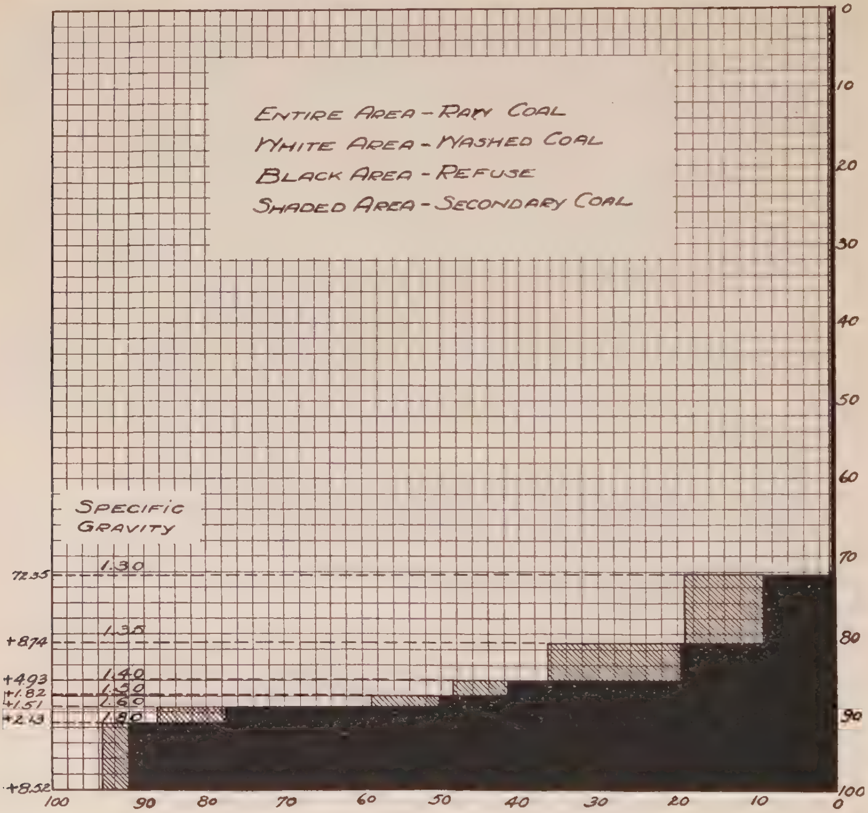

FIG. 6.—HANCOCK CHART, WASHING TEST ON ILLINOIS No.6 COAL.

Value of raw coal per ton	$1.50	$1.50
Units decrease in ash	6.1	7.0
Increase in value due to washing	$0.854	$0.980
Value of washed coal per ton	$2.354	$2.480
Washed coal produced per ton raw coal	1726 lb.	1570 lb.
Value of washed coal from a ton of raw coal	$2.036	$1.946

This is a hypothetical case used only to show the type of calculations involved. Other factors entering into the enhancement in value of the washed coal have not been taken into account. It serves, however, to show the bearing of commercial conditions on the coal-washing operation.

Another common type of problem is found in the plant designed to prepare coal for sale on specification, the contract providing for delivery of coal of a specified ash content and providing a penalty for excess ash and a premium for a decrease in ash content below that specified. Such a selling arrangement provides a specific increase in selling price per unit decrease in ash content and the relative values of the products for different adjustments of the plant may be calculated in the same way as for the coking-coal operation. In this way, the net values of output secured by making the different separations shown to be possible in the operation represented by curve 2, Fig. 5, have been calculated assuming the following conditions: Contract price $1.50 per ton of coal of 10 per cent. ash content; penalty for excess ash above that specified, 4 cents per ton of coal per unit of ash; premium for ash lower than that specified, 4 cents per ton per unit ash content. The results of these calculations have been plotted to form curve 3, Fig. 5, showing the net value of the output of washed coal per ton of raw coal corresponding to the different yields of washed coal securable. This curve shows the maximum value of the output at the point *A* corresponding to a yield of 90.5 per cent. of washed coal. This is, therefore, the most profitable separation to make under the marketing conditions assumed. A different set of penalty and premium provisions, of course, will change these values, but the method of calculation is the same. Curve 2 at the point *B* shows an ash content of 10.3 per cent. corresponding to this yield of washed coal. Curve 1 shows a 92.8 per cent. yield of float coal of 10.3 per cent. ash content. The efficiency of the separation at this point, as determined by the yield-curve method, is then $\frac{90.5}{92.8}$, or 97 per cent.

If the sulfur content is the important consideration, a calculation on this basis may be made in the same way.

Other conditions apply in other cases. Where the coal is sold in the open market, for example, even if the washed coal brings no higher price, it may help the selling organization; and, if the washery enables the operator to sell coal when competing mines are idle, this may be the sole consideration in determining the value of the washery operation. There are also many exceptional conditions such as a high-sulfur coal made suitable for the manufacture of metallurgical coke, or the production of a utilizable fuel from a worthless raw material. In the examination of a Brazilian coal, it was found to be perfectly feasible, commercially, to operate with a washery loss of 50 per cent. in order to produce a saleable washed-coal product.

Precision and Limitations of Efficiency Calculations

Having determined approximately the range of separations that are possible with the coal under investigation, a consideration of the economic conditions will indicate the most profitable separation to make. The solution that makes this separation should be taken as the standard for the sink-and-float bath for application of the efficiency formula to operations on this coal.

When using these methods of comparing tests, it is necessary to consider the degree of precision in the calculation of efficiency numbers. The greatest source of error in both the formula and the yield-curve method is probably in the results of the sink-and-float test, because of the difficulty of securing a representative sample as well as the possibility of error in making the separation. The precision of this test also varies, in a measure, with the solutions used. Closer checks are generally secured on solutions of high specific gravity than on lighter solutions. The methods suggested here have the advantage that the sink-and-float test, which is the least precise determination involved, enters into the calculation only once.

A more complete discussion of the probable errors in experimental coal-washing work will be presented later in a more extensive publication. The efficiency numbers recorded here are carried out only to the nearest integer because the study made has indicated that figures to the right of the decimal point are not significant.

The great diversity of commercial conditions shows that the universal application of a formula to produce efficiency numbers for direct comparison of different plants, operating on different coals and preparing coal for different purposes, is of doubtful value. This is only one of the many variable factors that enter into and complicate the problem.

In determining the effectiveness of a washery in operation, the nature of the coal is a more important factor than the type of washer used. Coals differ widely in the ease with which they may be cleaned. A very dirty coal will show a much higher qualitative efficiency than a comparatively clean coal, because the excessive dirt in the high-ash coal is very largely removable. Some coals require much finer crushing than others, in order to break apart the dirt and the coal, and it will be more difficult to make a complete specific-gravity separation on the fine coal than on the coarse. Friable coals will contain more fine material than harder coals even when crushed to the same maximum size. Coals may differ widely in the friability of their impurities, some containing shales and clay that disintegrate in water and make cleaning much more difficult, although the sink-and-float test does not show it. Differences in the proportion of middling material in raw coals affect greatly the degree of separation securable. A complete specific-gravity analysis, therefore, would be much more satisfactory than a sink-and-float test on one solution as a basis for comparing operations on different coals. Two coals, for example, might have the same proportion of sink in a 1.35 specific-gravity solution, with in one case the bulk of this between 1.35 and 1.80, and in the other case practically all heavier than 1.80. All these variable conditions complicate the problem of comparing widely different operations, and render it impossible to devise a formula to produce comparative efficiency numbers which may be interpreted unreservedly as indicating the relative effectiveness of the different plants. Such formulas may be of assistance, however, when due consideration is given to the conditions that do not enter into the efficiency calculations; therefore, in reporting results of such investigations, these qualifying conditions should be fully described. The bare statement of relative efficiencies would be of little value.

The efficiency calculations become much more definite and dependable when applied in a narrower sense for the specific purposes described in this paper and it is believed that their chief value is for such uses; that is, for the comparison of operations on the same coal to determine the most suitable method of treatment, or for comparison of results secured with the same equipment on different coals to determine their relative washability. The comparison of results secured on the basis of yields and analysis of products measures only the completeness of separation from the mechanical standpoint, and does not take into account the other factors that influence the choice of a washing method, such as cost of installation, water consumption, power consumption, supervision required, upkeep cost, and rate of depreciation.

DISCUSSION

Ray W. Arms, Chicago, Ill. (written discussion).—The authors are correct in saying that an efficiency formula, giving a single figure to express the result of a coal-washing test, would be undesirable. One of the conditions under which a single figure would be misleading is in the separate disposition and use of the bone or middle product. When this can be removed and used as a plant fuel, a better grade of clean coal and a refuse higher in ash will result from the washing operation, but the efficiency formula contains no reference to this condition.

The latest practice is to have coal dry cleaned instead of washed and the efficiency numbers are equally valuable in this new art to develop the best practice on any particular coal. They will not only assist in selecting the best machine, but will control the operation of plants.

The special method of comparison of table-washing tests described by the authors is particularly adapted to dry cleaning, for all machines developed so far for handling coal up to 2½ in. are in the form of tables.

It is possible with these tables to take off several products in zones near the coal-refuse junction to determine dry-cleaning efficiency and develop yield-ash curves such as the authors describe. In fact, this method could well be used to make a comparison between wet and dry tabling.

The statements regarding economic considerations apply with still greater force to dry cleaning, with which process it is much easier to change the character and bulk of the clean and middle-ash products to suit varying market conditions; hence all the more necessity for knowing the yield and quality curves for an entire series of specific gravities. With these data, an operator may govern the quality and the yield between known limits to suit conditions of competitive and non-competitive seasons.

These methods of interpretation will probably find their chief application, in dry cleaning, to the determination of the proper screening schedule for any coal. Dry cleaning will always require closer sizing than washing but in this respect also coals will differ.

J. R. CAMPBELL, Scottdale, Pa.—In 1910, G. R. Delamater and I worked on the dry cleaning of coal. The 3-S table is a big step in advance of the older dry processes. I do not agree with all of Mr. Arms' statements in regard to sizing. A washing plant should be as simple as possible and the coal should be treated unsized, if possible. One of the drawbacks of the present method of dry cleaning is too much sizing. We who have been through the washing problem do not like to recognize a middle product. Two products are essential—clean coal and refuse.

G. R. DELAMATER, Harrisburg, Pa.—In 1914, I published some coal-washing efficiency formulas in an effort to start work along this line. While undoubtedly there were objectionable features to this work of mine, others have become interested in this subject, and the authors have been able to make use of my formulas. I agree with them that at the present time, at least, the main value of a single efficiency figure would be for comparison from day to day of the operation of one plant, rather than for a comparison between plants. It would be an advantage to the management to be able to tell from one efficiency figure just how the operation of a certain plant compared with that of the previous day, week, month, or year. I do not think, however, that it is possible to get a figure that can be used for direct comparison of plants in different districts or different parts of the country, where conditions may be far from parallel.

I have been much interested in the development of dry concentration, for the past ten or twelve years, having had considerable experience with the Bonson process which was one of the first to attract attention. The handling of the fine coal in the wet washery is about the most serious problem of all. It is exceedingly difficult to dewater the fine coal and practically all coal losses may be traced to the fine coal in the water; 90 per cent. of the operating difficulties also are traceable to wet fine coal. High moisture content of coal charged to byproduct coke ovens affects coke yields and loosens the carbon deposit on the walls of the coking chamber of the oven, resulting in leakages and losses of byproducts. By screening out the fine coal before washing and bypassing it around the washery, mixing it in with the washed oversize coal, I have obtained very satisfactory results, including greatly increased yield of coal from the washery and, on account of greatly decreased moisture content of the coal charged to the ovens, an increase in the capacity of the byproduct coke ovens themselves. In a recent installation of this kind, the capacity of the byproduct coke oven plant was increased 10 per cent. by the reduction of the moisture content of the coal charged to the ovens.

Dry concentration will have its place, but in most instances probably a combination of the wet and dry processes will be found most satisfactory, the wet for the larger sizes and the dry for the smaller. However, concentration of the fine coal should be avoided wherever possible and careful preliminary investigation made to determine how far down the scale of sizes concentration must be carried to accomplish the desired results, bypassing as much of the smaller coal as possible.

R. H. Sweetser, Columbus, Ohio.—Those who are making coke for the steel plants have one of the biggest jobs in the steel industry in reference to the elimination of waste, because the steel men recognize that the making of good steel begins at the coal mine. Using 600,000 cars annually (5 per cent. extra ash in the form of slate and clay means about 30,000,000 tons of useless material in our annual production of bituminous coal) to carry waste that should be left at the mine is one of the greatest wastes in this country, and I hope that the problem of cleaning coal for coke ovens will be worked out so that men who must make low-ash coke will be able to interpret the different methods of cleaning coal. I have just come from two coal cleaning experiments; one at the Ohio State University and one in the anthracite region, both on bituminous coal. With dry cleaning, close-sizing seems to be necessary for proper cleaning.

This paper speaks of the necessity for low sulfur and low ash in steam coal; they are even more important in coking coals. With pig iron at $20, every per cent. of ash in the coal over 6 per cent. will decrease the value of the coke by 30 c. per ton, which means a great deal if there is 6 per cent. excess ash in the coal.

Carl A. Wendell, Washington, D. C.—I have been identified with coal washing since 1903 and believe that radical changes in methods are needed if usual losses are to be overcome. Coal washer efficiency calculations will help toward better operation but stereotyped washer installations do not answer the purpose. In present-day coal washers there is much handling, screening and pumping, all of which costs money. We are trying to do too many things. We should wash for only the main products, which are clean coal and refuse. If a secondary product is recovered we should not rob ourselves of good coal by adding such to the secondary product.

In England they found, some years ago, that the coal best prepared for coal washing was crushed to 1 in. In this country I have washed all kinds of coal on all kinds of apparatus and have found that the results on coal crushed to 1¼ in. and screened 1¼ to ⅝ in. and from ⅝ to 3/16 in. did not show any appreciable difference in the ash content of the washed coal. The difficulty begins with the coal below 3/16 inch.

Dry cleaning will not solve this problem, as the finer sizes cannot be properly screened, because of a large percentage of moisture collecting in the fines. We shall have to devise some method by which coal washing is accomplished without hand adjustments of the apparatus, based upon individual opinion, and where the coal is cleaned automatically, regardless of ash content. One method by which coal washery losses can be eliminated is the Trent process, in which the fines are ground wet to 200-mesh, mixed with a certain amount of oil or tar, and stirred. The result is a plastic substance, practically free from ash and moisture, which is screened out of the water, which now contains only finely divided impurities. It is handled, much as any other material, by conveyors and grab buckets.

In this process, advantage is taken of the variance of surface tension between coal and oil on the one hand as against water and ash on the other; globules of oil are formed in the water which, during the stirring, become full of coal and grow larger the longer they are stirred, leaving the ash locked out, so to speak. There is no secondary product, no loss, and no sizing of any kind.

George S. Rice, Washington, D. C.—I recently visited the first commercial air cleaning installation for coal; this belongs to the St. Louis & Rocky Mountain Coal Co. and is located near Raton, N. M. I was greatly impressed with its compactness and efficiency. The plant has a capacity of about 55 tons per hour and, in contrast to washery plants, has no tanks or sludge ponds. It was claimed by the operators that the results compared favorably with the work done by their washeries. As the product goes to coke ovens, an improvement of over 2 per cent. in ash is of greatest advantage. At the time I visited the plant it had not been running regularly, they had to make many changes in the setting of the machines. The spiral separators treated the coarser coal, from 1½ to ¾ in. Coal below ¾ in. was divided by screening into five sizes by as many separate anti-gravity screens, in duplicate. On the floor below there were air tables or air jigs for each size.

I was favorably impressed with the advantages of dry cleaning of the small sizes which are such a problem in the washeries, especially in the disposal of the sludge and dirty water. Again in the arid part of the country, getting water is a problem. Another advantage is the escaping of trouble from freezing, which is such an acute problem for washeries in the northern part of the country. There is perhaps not so much trouble from freezing within a washery having good housing, as when the washed coal is in transit; there is certainly great advantage in obtaining dry coal, whether for coke oven or for boiler use.

The hazard of coal-dust inflammations in dry cleaning plants should not be difficult to overcome, but the designs of plants I have seen indicate that the designers were not fully aware of the dangers. In the first place, the machinery should be covered as much as possible and dust hoods with an exhaust system should be installed. Open motors, switches and electric fuses should not be used in the screening and dry cleaning part of the building, but should be separately housed or else be of enclosed explosion-proof type. All ledges and beams should have surfaces slanting at least 45° or steeper, obtained by sheet iron or cement covering so that there will be no accumulation of dust to be dislodged. Smoking or the use of open lights should never be permitted in such plants. Most coal-dust inflammations—they do not reach the stage of true explosions—have occurred after shutdowns in a plant or while cleaning up, when the air is charged with the finest float dust.

RAY W. ARMS, Chicago, Ill.—So far in our test work on dry cleaning we have covered sizes ranging from 2½ in. down to 150-mesh. Through that range we can clean coal technically, but whether we can clean all of the sizes commercially remains to be seen. On the upper end of this series we get as good a cleaning as by the wet process, getting as low ash in the clean coal and usually a higher ash in the refuse. It remains to be seen also whether we shall be able to develop the enormous amount of air required for cleaning coal above the upper size indicated.

One objection to the cleaning of larger sizes is the enormous amount of air required, but in view of the larger capacity when treating the coarser sizes, the air requirements are not excessive. On the smaller sizes the capacity of the tables is so low that probably at some point between ¹⁄₁₆ in. and 100-mesh we shall have to stop, the actual point depending on the coal. In some cases a large amount of ash can be removed by cleaning ¹⁄₁₆-in. coal; in the West Virginia fields it has proved impracticable to go below this size.

Two main objections to dry cleaning are the number of sizes and the dust. Dry cleaning does require closer sizing than wet cleaning, but, on the other hand, companies using coal washing methods have lately used close sizing with better technical results. If eight or ten sizes are not demanded for good commercial results, we certainly shall not use them;

and if the reduction in ash that can be accomplished on two or three sizes is satisfactory, we shall probably confine our efforts to those sizes. However, if a lower percentage of ash in the cleaned coal is obtained by operating on ten different sizes, and it requires ten tables to handle the capacity, what is the objection to treating a different size on each table? Instead of a feeder to distribute the coal to the various tables, a screen is used to feed a separate product to each of these units, and the screen is not much more complicated than a feeder; the result will be a cleaner coal and cleaner refuse.

Naturally, the dry-cleaning plant will make a lot of dust. At Raton they have added new units to the dust collecting system; this again is a problem of determining just how far we must go to get what may be considered adequate dust collection. It is my belief that dry cleaning plants, as they are now designed, will cause much less dust than ordinary screening tipples.

Last week I went through two plants in the Pocahontas field where the dust made it impossible to see from one end of the building to the other. In spite of this condition, men were carrying lighted torches and were smoking, without breaking any rules. Not much has been said about the dust in such plants, but as soon as dry cleaning is suggested, dust is always cited as an objection.

R. Dawson Hall, New York, N. Y.—The hope of success with washing methods based on the use of fine and heavy sands for the separation of clean coal, described as being of variable specific gravity, seems doomed to failure. Neither the Chance nor the Conkling washer can differentiate between clean coal and refuse when the clean coal is of variable specific gravity, for the operation of both is based on the buoyancy of coal in a liquid of some definite specific density. I understand that some high-ash particles in Vancouver coal are lighter than some particles that have less ash; as a result, the engineers who are cleaning such coal must depend on flotation, in which the tendency to sink or float does not depend on the specific gravity of the coal.

Dever C. Ashmead, Kingston, Pa.—In plants under operation the specific gravity of the liquid in the Chance washer can be, and is, varied from 1.65 to 1.75, to suit the coal being washed.

R. Dawson Hall.—That is doubtless true; the mixture can be graduated so that the specific gravity best suited for the whole coal product can be used, but there is no way of suiting the gravity of the liquid to the needs of any particular lump of coal when the good coal is of many different specific gravities. So long as dependence is placed on gravity, a coal of greatly variable density, especially in the smaller sizes, can not be cleaned satisfactorily by a gravimetric method.

Robert Hamilton, Birmingham, Ala. (written discussion).—A coal washer with 100 per cent. efficiency, I presume, means one making a perfect separation at a predetermined ash content without any loss of coal. This ash content will vary considerably depending on the purpose for which the washed coal will be used. For metallurgical purposes, the ash content should be as low as possible; for commercial sales, it will depend entirely on market value of washed product. If these premises are granted, the following formula is suggested as a practical method for determining efficiency of any washer.

$$\textit{Efficiency} = \frac{a}{b} \times (100 - c),$$

where a = per cent. ash content predetermined.
b = per cent. ash content washed coal.
c = per cent. of clean coal lost in operation.
= per cent. of refuse × per cent. of float at 1.37 sp. gr.

Applying this to the results obtained by the same washer operating on different coals:

A.—Average of one year's work (samples taken daily) on coal containing 4 per cent. inherent ash $= a$

Washed coal, 4.63 per cent. ash $= b$

Refuse, 6.75 per cent. of raw coal; float in refuse, 3.1 per cent. of refuse.

$$\textit{Efficiency} = \frac{4}{4.63} \times [100 - (6.75 \times 0.031)] = 86.4 \text{ per cent.} \times 99.79 \text{ per cent.}$$

$= 86.21$ per cent.

B.—Average of one month's work (samples taken daily) on coal containing 8 per cent. inherent ash:

$$\textit{Efficiency} = \frac{8}{9.228} \times [100 - 0.92] = 86.69 \text{ per cent.} \times 99.08 \text{ per cent.}$$

$= 85.89$ per cent.

In this method, the ash content in raw coal is not considered, as any change in quantity thereof should not affect the efficiency of a jig, but will affect its capacity.

At the above-mentioned washer, in the second example, it was necessary to reduce the capacity of the jig nearly 30 per cent. to secure the desired results. It is a three-cell, double-plunger type producing three products: washed coal of 1.37 sp. gr., boiler coal of 1.37 to 1.56 sp. gr., and refuse over 1.56 specific gravity.

When car-load washing tests are being made, it is customary to secure actual weights of raw coal, washed coal, and refuse; also, to make allowance for fine coal remaining in circulating water. But in every-day practice, how often is it possible to secure actual weights of raw coal, refuse, or fine coal carried off in waste water? This latter loss is not generally considered, but in a number of cases it was found to be 0.5 per cent. of the raw coal delivered to the washer, or a loss of 5 tons per day on an output of 1000 tons.

DAVID HANCOCK, Birmingham, Ala.—Ash and sulfur tests alone never indicate correctly the comparative efficiencies. Ash and sulfur content are influenced too greatly by variations in the feed coal. We find large variations due to irregular quantities of roof, bottom, and parting shale loaded by the individual miner, and also when different seams of coal are under consideration the character of the coal and the quantity, size, and density of the impurities are usually quite different.

A specific-gravity analysis in the former case shows large variations in the quantity of sink heavier than 1.80, but a fair degree of correspondence in other respects; in the latter case, large differences are found in the quantity and size of the various impurity fractions obtained as sinks by use of solutions of various densities.

The comparative efficiencies of different processes of coal washing can then be better obtained by a determination of the efficiency with which each fraction of impurity, lying between certain arbitrarily chosen densities, is removed.

The efficiency with which a given class of impurity is removed is properly expressed by dividing the quantity rejected as refuse by the quantity in the original raw coal. Since it is often impracticable to weigh the total refuse, the same result can be obtained by float and sink tests on the raw and the washed coal, correcting the results for the change in moisture content and the diminished total weight of washed coal.

$$E = \frac{I_r - YI_w}{I_r}$$

where E is efficiency; I_r the percentage of a given class of impurity in the raw coal; I_w the percentage of the same class of impurity found in the washed coal; and Y the percentage yield. All weights, of course, are first reduced to dry basis.

The Drakeley formula for qualitative efficiency reduces to these same terms except that it fails to make the correction for yield.

It is convenient to split the impurity into four or five fractions according to density, and a coal washer will show a decidedly different efficiency on each of these fractions. It is also frequently necessary to take into consideration the size of the impurity particles and to split each density fraction into four or five more parts according to size. Very marked differences in efficiencies are also found in this respect.

In such cases we would have from 16 to 25 different efficiency figures, all of which it would be necessary to consider and the combination of these into a single figure would be impracticable if not impossible.

The most desirable and obvious line of investigation seems to be to establish the value of E for certain selected densities and certain sizes of impurity particles by means of tests on coal washers of standard designs and, by plotting the results, to obtain a series of size-efficiency-density curves. Such information would aid very materially in the prediction of washer results from untried coals and also in the comparison of efficiencies of different plants. I have made a few such curves from the fragmentary information that I have and have published the results.[6]

The problem is complex, as every one realizes, and the authors have made a praiseworthy effort to clear up some of the obscure points. The float-sink-yield curves and the yield-ash-value curves especially clarify the problems considerably.

The use of graphs, diagrams, and charts so as to get a complete picture of the whole problem is much more productive of results than an attempt to compress so many variables into a single formula.

B. M. Bird,* Seattle, Wash. (written discussion).—The views expressed in the paper are somewhat divergent from those held by the Northwest Experiment Station of the Bureau of Mines, regarding certain points in the subject treated.

Almost everyone familiar with coal-washing work will agree that, before any washing tests proper are begun, a series of screen-sizing tests should be made on a representative sample of the raw coal, and a complete series of float-and-sink tests run on each of the resulting sizes. On our western coals, in addition, it is necessary to determine what degree of crushing of the bony products will break the bond between the coal and associated impurities. Then, having the proportion of each size of raw coal and the possible recovery of each, the next step is the interpretation of these data in the light of commercial and economic conditions. This work permits of laying out a systematic line of attack for the washing tests and the planning of the flow-sheet of a suitable washery. The investigator knows what he wants to do, and so the washing tests on various mechanical devices resolve themselves into the selection of the device that will come the nearest to accomplishing what the float-and-sink tests have indicated as possible, and the study of economic conditions has shown to be profitable. In other words, the washing tests are supplementary to the screen-sizing and float-and-sink tests and the interpretation of the actual washing tests should be tied up to the flow-sheet planned from them. Each washing test is one little part of the whole plan and fits into its place in the flow-sheet laid out from these preliminary screen-sizing and float-and-sink tests.

The authors may not have felt that a discussion of the best methods of interpreting screen-sizing and float-and-sink tests belonged in this paper, but I cannot see how anything so inseparably bound up with this work can properly be left out. It is the very backbone of the matter and puts the washing tests on a basis where they can be interpreted in the best manner possible.

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* Associate coal mining engineer, Bureau of Mines.

[6] *Coal Industry* (Jan., 1923) 6, No. 1.

The authors' objection to Drakeley's formula is fair. The "float" and "sink" are not the same grade of material, that is, having the same ash content in the feed, in the washed coal, and in the refuse. On page 448, in the last part of the second paragraph, however, it is stated "that a formula which may be applied indiscriminately to all cases and conditions to produce efficiency numbers for direct comparison is out of the question," and this statement is repeated in the fourth paragraph of page 468. If this be true, the authors' objection to Drakeley's formula is not a matter of great importance. Efficiencies are a relative matter, and in any given washery, or in the case of washing tests on the same coal by different methods, efficiencies can be compared and improvements in the results will be reflected by improvements in the efficiency as computed by the Drakeley formula.

The authors' objection to the Drakeley formula on the score that it depends on specific gravity separation rather than on ash and sulfur removal is in its favor. Drakeley depends on the effectiveness of the separation between particles of coal and impurities which are in a free state. After all, it seems more reasonable to judge the process on that basis. When coal is washed in a machine which depends for its separation on differences in specific gravity, the desired reduction in ash and sulfur should be expressed as the removal of a certain proportion of the particles having highest specific gravity, and the washing apparatus should be judged by how near it comes to performing the desired separation. The jig does not remove ash and sulfur as such, but removes the particles of highest specific gravity.

The statement on page 456, concerning the relative accuracy of float-and-sink data and ash analyses, is not a valid objection to the Drakeley formula. The efficiency figure obtained by any formula is not more accurate than the sample of the products of the washer. Take, as an example, a jig working on coal sized between 2 in. and 1 in. To insure its being representative, fully a ton of each product would have to be taken as a sample, which is impractical; so a 300- to 400-lb. sample is taken and a probable error of about 5 per cent. is introduced at this stage of the work. The float-and-sink data in Table 9 are arranged in a convenient form to compare the ash obtained in a series of float-and-sink tests over a range of sizes and specific gravities. It is evident from a consideration of them that the error in properly conducted float-and-sink work is undoubtedly under 1 per cent.

This table illustrates how closely the ash contents agree with the float-and-sink work. Since the ash content is the average ash of the material between the two specific gravities, it is natural that the ash in the material between 1.55 and 1.70, a difference of 0.15, should not

TABLE 9.—*Float-and-sink Tests of Wilkeson Beds Nos. 2 and 3*

(For comparison of ash in products over a range of specific gravities and sizes)

Size	Bed No.	Specific Gravities				
		1.35 to 1.40	1.40 to 1.45	1.45 to 1.50	1.50 to 1.55	1.55 to 1.70
Egg	2	17.3	22.1	26.7	31.9	40.5
Through 3 in., over 1½ in.	3	16.8	22.0	27.4	31.9	40.7
Nut	2	16.7	22.2	27.2	31.6	40.2
Through 1½ in., over ¾ in.	3	17.0	22.3	27.5	32.2	39.8
Pea	2	16.5	21.9	27.3	32.6	40.9
Through ¾ in., over ⅜ in.	3	17.3	22.5	27.2	32.0	41.1
Buckwheat	2	16.5	21.4	27.3	32.4	42.0
Through ⅜ in., over $\frac{3}{16}$ in.	3	17.3	22.5	27.2	32.0	41.1
Slack	2	16.9	22.3	27.3	31.7	44.3
Through $\frac{3}{16}$ in.	3	16.2	22.5	27.4	32.0	39.7

agree so well as that for only 0.05 difference. These results are no different from others obtained in the state of Washington. They are given because the two beds are remarkably similar, and also are of the same rank as most Illinois coals. They check almost within the error of opposite quarters in coning-and-quartering and indicate that float-and-sink results are sufficiently accurate for efficiency calculations.

E. R. McMillan and I have always held that the principal use of any efficiency formula is to assist the washery man in adjusting the washing apparatus. Now the authors' formula and one based on float-and-sink data are on an equality, as far as the time required for sampling is concerned, but once that is completed, there is quite a difference in the time required to make the results available for use. Of course, one sample could be dried, crushed, and analyzed in about 8 hr., but where a number are run, as must be done in practice, it takes practically two days to make chemical analyses, as judged by the Carbonado washery where we are now working. In other words, the washery is two days ahead of the means of controlling it. Now, comparing with float-and-sink data, it requires 20 min. to make the float-and-sink tests of each sample, say, of 300 to 400 lb., or one hour for three products, if that is the number being made. Add one-half hour for draining the surplus water from the samples, weighing, and calculating the efficiency. Wet weighing is amply accurate for any kind of work except on very fine sizes, the error being well within that of the sampling. In the state of Washington, with its difficult underground conditions, the washery problems often change several times during a day, so that any efficiency formula based on chemical analyses would only be a check on how well the washery man had guessed two days before.

In spite of what I have just said, however, I do not favor Drakeley's formula in its present form. I am, however, in favor of a formula based on float-and-sink data because in the solution of problems around these washeries one needs his results at the earliest possible moment.

The authors should include in their paper an example showing the use of their formula. "Standard yield" has two possible meanings, "the percentage of float in the raw coal," or "the percentage of the original float in the raw coal that was recovered in the washed coal."

In the discussion on page 462 and following, have the authors taken into account the effect of tonnage handled per hour on the work of tables? To make a comparison between the work of these two tables, it would be necessary to have these figures. The laboratory table may have been operated on a light feed, whereas the commercial table may have been overfed. It has been my experience that the operators of commercial tables will nearly always beat the recoveries predicted from laboratory-table tests. In a plant, the table operators have plenty of time to vary the adjustment until they find the best possible conditions; there is little opportunity for such work in the ordinary laboratory table test.

In the summer of 1921 we began to study zonal products from the tables much as the authors advocate in their paper, but made float-and-sink tests of the products in addition to the ash analyses and screening tests. From the results obtained in this manner, I recommend this method of studying table tests.

The meaning in the first sentence of the last paragraph on page 464 is not clear. Crushing might have quite a marked effect which should be considered in this connection. The right way to perform this test is to run float-and-sink tests on two samples of the coal, one on 0 to 2-in. coal and the other on a duplicate sample of the same coal crushed to pass ½ in., using the same "permissible bath" on both. The proper specific gravity for the bath would, of course, have been selected from the results obtained on a complete series of tests on the raw coal. Any increased yield of coal secured by crushing as indicated by the float-and-sink tests should be taken into account in considering the results by the two methods of treatment.

THOMAS FRASER (author's reply to discussion).—The discussion of efficiency calculations and coal-cleaning has been gratifying. The sink-and-float efficiency formula presented in the first part of the paper is a consolidation and simplification of the four formulas proposed in 1914 by Delamater. The method used by Drakeley is very similar. All these formulas are useful for comparison and, including the one presented in the paper, are based on sink-and-float data. We followed Delamater's practice of using the ash in the float coal rather than the concentration of float coal in the washed coal as the standard of qualitative efficiency because it is more practical; it eliminates a large error due to differences in ash content of sink material, and the ash analysis is a more accurately standardized determination than the float-and-sink test. There is a quite definite relation between specific gravity and ash content of a given coal and use of the ash content is equivalent, for this purpose, to using the specific-gravity analysis data and in addition secures the above advantages. This relation between ash content and specific gravity was worked out by Eckley B. Coxe in his early work on anthracite and was made the basis of a method of calculating the ash content by determining the specific gravity. The figures on ash content of different specific-gravity fractions of two Washington coals presented in the discussion shows how closely this relation holds even in this case on samples from different beds. It is not clear, however, how these figures are interpreted as indicating the probable error in the sink-and-float test itself, because no data on the sink-and-float separation are given. If any comprehensive series of tests has been made on duplicate samples to determine how closely the sink-and-float test can be checked and what size of sample is necessary to secure good checks on the different sizes of coal, this data would be a valuable addition to the published literature.

The methods with which we have worked most are the yield-curve and the multiple-sample method for comparison of table tests. The yield curve is simply another method of presenting the data used by David Hancock in preparing the Hancock chart, a sample of which was published by the Alabama Geological Survey in 1912. We made a series of sink-and-float tests by this method, named it a specific-gravity analysis, and plotted the results in the form of a cumulative yield-ash curve. By comparing actual washing results with this curve, the qualitative efficiency can be made 100 and the efficiency determination become merely a comparison of yields. This method has been used with success in the testing work of the Bureau at the Urbana Station and by E. R. McMillan at the Seattle Station.

The multiple-compartment method of sampling is particularly applicable in making tests on the pneumatic tables; the zonal samples are easily kept separate and complete because there is no flow of water to carry away the fines. In the investigation of some special problems, such as cleaning very bony coals, recleaning refuse, or cleaning a coal in which the refuse cannot be distinguished from the coal by visual examination, we have found this to be the only way in which the best separation securable can be ascertained. The application of adequate methods of evaluating results in dry-cleaning tests is of particular importance now because of the large amount of experimental work being done with the pneumatic tables and the widespread interest in their use for cleaning coal.

Design and Operation of Roberts Coke Oven

By M. W. Ditto, Chicago, Ill.

(New York Meeting, February, 1923)

The conversion of the beehive coke plants, in this country, to byproduct plants has been slow, because the coal supplies were near the centers of the steel industry. With the growth of this industry, especially with its development around Chicago, it became necessary to transport large tonnages of coal from the eastern districts and then convert it into coke. The losses due to transportation costs were partly offset by the value of the byproducts recovered.

To save the transportation costs, it was desirable that the coal deposits of Indiana and Illinois be utilized. This coal had always been classified as non-coking; it was also considered unsuitable for the metallurgical field because of the high ash and sulfur contents. The conditions, however, were promising enough to start experimental and development work, which crystallized in the design of the Roberts coke oven.

The fundamental features of most coke ovens, with respect to the application of heating gas and the recovery of byproducts, are the same and, in the last few years, the tendency has been toward the better application of heating to the walls, higher thermal efficiency, and, by the control of heating conditions, the increase of byproduct yields. Structural features also have been improved and the use of high-grade refractory material has allowed the use of higher temperatures with the resulting higher rated capacity per oven per day. Great improvements have been made in both recuperators and regenerators, particularly with reference to the individualizing of each oven with respect to its adjacent oven, so that each can be operated as a unit if desired. The highest development of the flue type of oven has been applying individual regenerators for the recovery of waste heat.

After some research work, it was found that the high-volatile Illinois coals could be utilized for coke by obtaining the better application and control of the heating conditions in the oven. Most of this high-volatile coal that had been used gave a good coke structure under certain conditions. The fact that coke was occasionally made from this coal, demonstrated the possibility of making coke from it at all times provided the conditions under which the coke cell or structure formed could be isolated.

One problem in coke-oven design is to heat uniformally a surface as large as a coke-oven wall so that the results of the heat distribution will be uniform throughout the entire coking mass. Yet, the necessity of this is demonstrated by the fact that when a coke-oven wall is uniformly heated and a sufficient quantity of heat made available for the coking mass a good coke structure will be made from a large number of coals that otherwise do not give good results.

A coke-oven wall of standard size is approximately 43 ft. long and 14 ft. high, or about 600 sq. ft. With about 1200 sq. ft. of surface exposed to a cake of coal approximately 14 in. thick and weighing 30,000 lb., it is necessary that each square foot of the surface exposed to the coking mass be as nearly the same temperature as possible. One of the conditions in the coke-oven chamber that influences the quantity of heat required is the taper, which will vary from 1¼ to 2½ in. so that there is approximately 15 per cent. more thickness in the discharge side of an oven than on the pusher side, which means that at least 15 per cent. more heat must be made available at that zone. As this variation in thickness is gradual, the heat supply along the wall must be graduated. The variable heating conditions in the height of the wall must also be compensated for both, because of structural features in design and because of the different heat requirements in each vertical zone.

The Roberts oven is designed in three types: regenerative, recuperative, and the combination-regenerator oven, which can be heated by coke-oven gas or by blast-furnace and producer gas; or where it is necessary to preheat both the air and fuel gas. The fundamental principles of these designs are the same. The regenerative type necessitates the reversing of the flow of the air and gas; the recuperative type permits the flow of air and gas in one direction continuously. The ovens are mounted on concrete foundations, which are simply large flat pads of reinforced concrete of sufficient strength and area to support a number of ovens in each battery.

Roberts Recuperative Oven

In the recuperative oven, the foundation contains the ducts through which all the air required for operation passes. By this means the tendency to overheat the concrete foundation is overcome in a simple and effective manner; the incoming air absorbs the heat as rapidly as it is transferred to the foundation, so that the foundation is maintained at approximately atmospheric temperature.

A longitudinal section of the ovens and a cross-section of the concrete pad are shown in Fig. 1. Seven ducts passing through the concrete, the full length of the battery, carry air at atmospheric temperature in the bottom of either the recuperator or regenerator oven, depending on which type is used; Figs. 1 and 2 show how these ducts are connected to the recuperator.

The foundation of a battery of coke ovens must be stable for, if because of expansion and contraction the foundation is ruptured, there is great possibility of rupturing the brick structure above, which would allow short circuiting of air and gas, which might result in bad operating conditions.

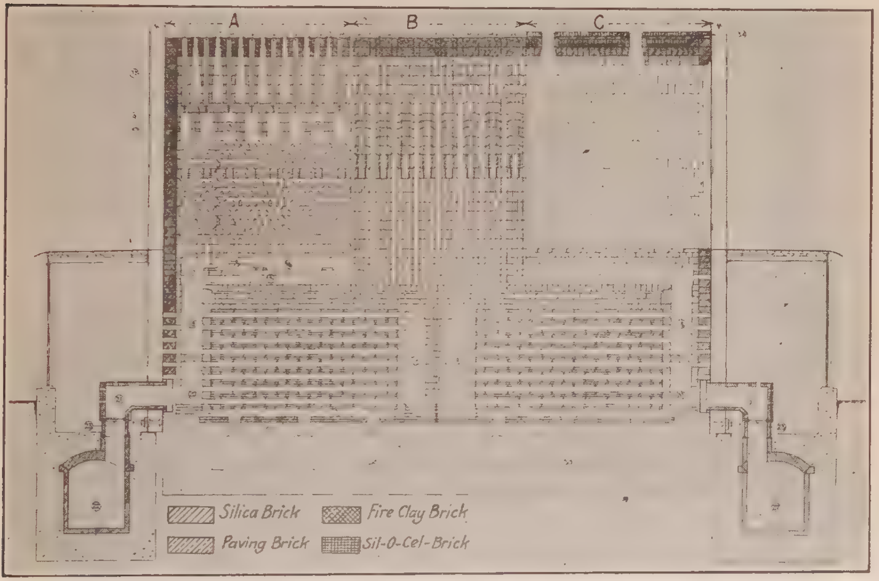

Fig. 1.—Longitudinal section through oven and cross-section of concrete pad. *A*, section through primary burners; *B*, section through secondary burners; *C*, section through oven chambers.

Directly on top of the concrete pad are two courses of fireclay brick. These act as heat insulation for the concrete and form a surface on which the silica brick slides when expanding or contracting. From the top of this fireclay course to the bottom of the battery nothing but high-grade silica brick is used. The supporting walls between recuperator or regenerator are made up of silica straights and shapes. They are 18 in. thick and are placed directly under the oven heating walls, so that the maximum support is provided for the entire structure above. Expansion joints are provided for the lower part of the oven chamber, the location of which is shown by the heavy black lines in (*a*), Fig. 3. This construction effectively prevents leakage from the oven chamber in the recuperator or regenerator chambers; the expansion joint also allows for the vertical expansion of the recuperator. The linear expansion is controlled by buckstays on the ends of the oven and by auxiliary buckstays at the end of the recuperators. The small buckstays are fastened to the main buckstays but may move independently. This arrangement permits the free movement, either vertically or horizontally, of the recuperator brick regardless of the movement of the oven brick or the supporting walls. The recuperator is made of silica brick, which always expands when going from atmospheric temperature to working temperatures thus permitting the control of the joints, so that leaks are reduced to a minimum. The heavy silica blocks are so laid that all vertical joints are broken. In this part of the structure are located the sole flues and the waste-gas equalizing flues for each heating wall.

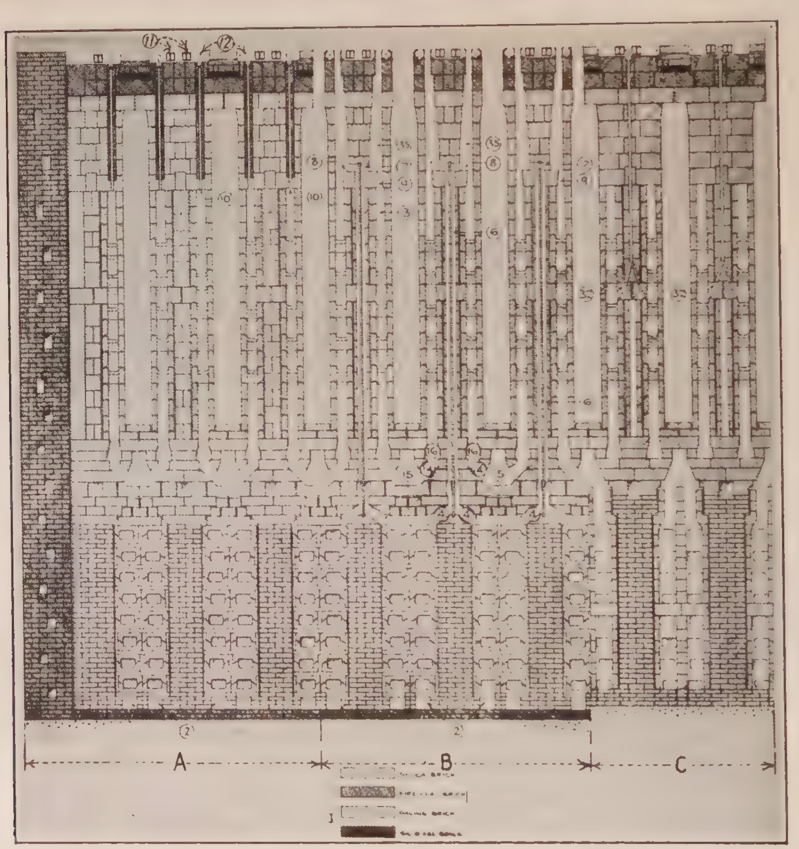

Fig. 2.—Sections through burners, charging holes, and air ducts. *A*, section through primary burners; *B*, section through charging holes and air ducts; *C*, section through secondary burners.

From the sole of the oven to the top of the combustion-chamber wall, two adjacent ovens are made up of two heating walls and an intermediate wall. This triple wall is about 30 in. thick and forms a structure of exceptional strength between adjacent oven chambers.

The center of the separating wall is composed of large flat blocks and shapes through which pass the ducts for air and the supply of secondary-fuel gas. On either side of this separator wall are the heating walls of one side of two adjacent ovens. In these walls is incorporated the space for the combustion of fuel gas.

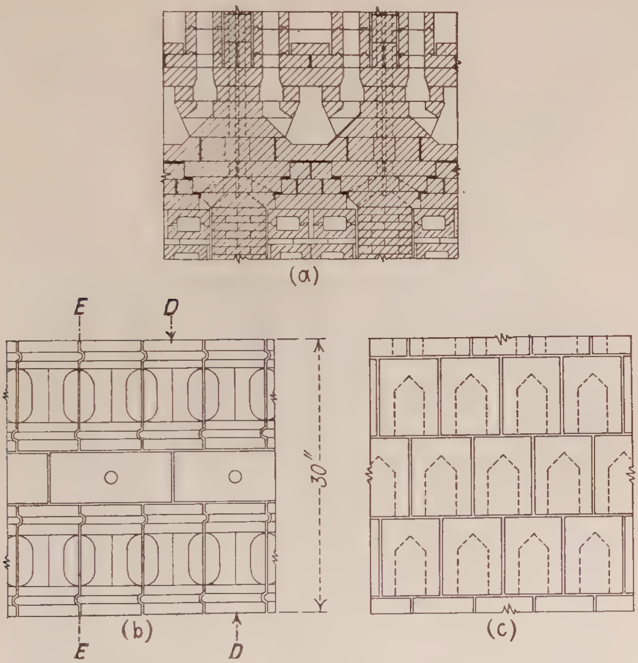

Fig. 3.

The shape of the brick of which this wall is constructed and the way they are assembled form one of the features of the Roberts oven. The brick is shown in Fig. 4; the manner of laying is shown at (*b*) and (*c*), Fig. 3. Each wall is backed by the center wall; the three walls together are sufficiently strong to withstand any stress placed upon them by operating conditions. From (*b*), Fig. 3, it is evident that the only stresses the bricks have to resist are in the direction of the arrows *D*, and they will resist these the same as if solid. They are so laid in the wall that the ends in registering form an arch and the pressure along the line *EE* is on this arch. The entire wall, therefore, is practically as strong as if there were no openings in it; at the same time ample open space is allowed for the passage of the burning gases through the wall.

In laying, each course is offset half the width of a brick, breaking the vertical joints, while at the same time the centers of the brick in one course come directly over the openings of the course below, as shown at (*c*), Fig. 3. From the standpoint of strength, the effect of this is the same as if each brick were reinforced over 50 per cent. of the area exposed to the coal, and the next brick above or below has the same amount of reinforcement offset on a new center, thereby increasing the strength enormously. The strength of this interlocked wall structure and the comparatively weaker construction of flues is at once apparent and the effect upon the extraction and disposal of the heat from burning gases in this wall is influenced in a marked degree, as compared to the flow of gases through a flue.

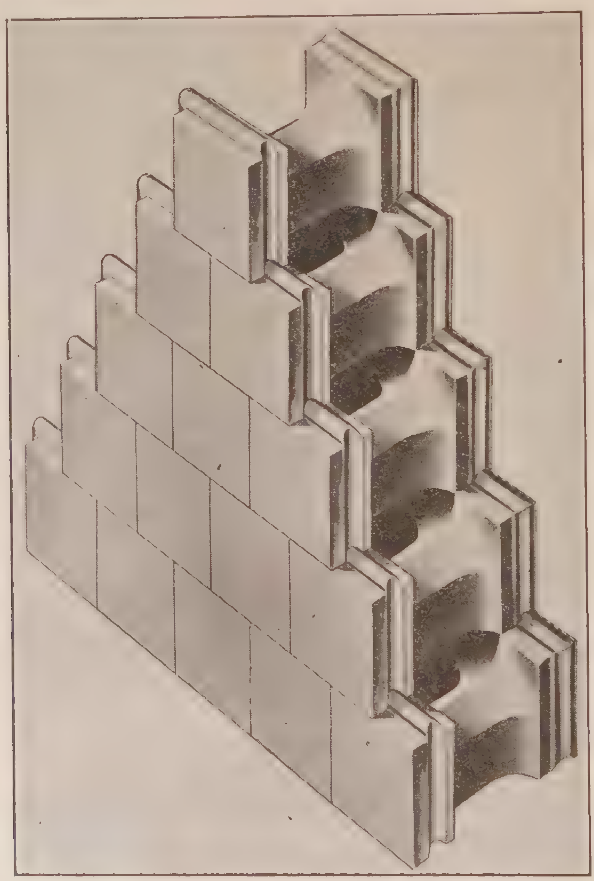

FIG. 4.—BRICK OF SEPARATOR WALL.

Experience has shown that it is difficult to hold a flue-like structure tight, there always being a tendency to develop leaks between the combustion and coke-oven chambers, causing overheating by the introduction of gases from the oven chamber in the combustion flues. This uncontrolled supply of gas easily causes hot spots and uneven heat effect upon the coking mass.

A flue structure is unsupported throughout its length, amounting to more than 700 sq. in. in the smaller type of ovens, while the greatest unsupported surface in the Roberts wall brick is only a little more than 14 sq. in.

A common comparison is to measure the total length of a joint exposed to the coke in a coke-oven wall, which is all right when comparing flue-type ovens, but this comparison does not hold true with a structure such as the Roberts oven, because the strength of the wall structure is dependent on the fact that the combustion chamber and the two faces of heating wall are incorporated in one brick, the rupture of which depends on the cracking of this brick in the center of the combustion chamber.

The inherent weakness in the flue-type structure is overcome by the use of thicker tiles between the combustion and coke-oven chambers, this tile in some cases being 6 in. thick and the average is well over 4 in. The Roberts wall is only 3 in. thick between the burning gases and the coal charge; this thickness is uniform from the top to the bottom, and from end to end of the wall. All the parts of this wall are uniform in strength, heat absorption, and conductivity. With the thin wall through which the heat is conducted to the coal, there is still the full strength of a solid wall 30 in. thick. This particular construction permits the apparent contradiction of the belief that thin coke-oven walls are fragile.

Near the top of the Roberts wall, and immediately below the nozzle for the introduction of the primary gas, is a short space in which the baffle brick are omitted; this forms the mixing chamber. While this chamber is short and comparatively narrow and the necessity for great strength is reduced at this point in the wall, the same keyed construction is used, only the baffles being omitted. Above the combustion chamber, the wall is solid except for the openings through which pass the gas ducts and the openings for the top air adjustment.

The crown of the oven is silica but, as the temperature above this point is low, clay shapes form the top of the battery. Clay is a better heat insulator than silica and is also more resistant to the weather. It is also possible to introduce high-grade heat insulating material in portions of the battery top, thereby lessening radiation losses.

Above the clay top brick are the gas headers that supply gas to the primary and secondary burners. These headers are rectangular in shape and the spaces between them and the brick are filled with a cement mixture so that the top of the battery is perfectly smooth.

The ends of the ovens are faced with clay shapes, which act as heat insulation and are also a protection from the weather. These clay shapes lie directly behind the buckstays and cover the entire space between the oven doors. A layer of silica, 33, Fig. 1, between the clay and the wall brick, protects the clay from direct contact with the heating gases. The clay shapes at the ends are made as large as possible, in order to reduce the number of joints; also they are so designed that all joints may be readily pointed up in case the cement falls out from weathering.

As the expansion of clay and silica are quite different, the clay used for these ends will not move vertically to as great an extent as the silica in the heating walls. The silica will, therefore, slide on the clay and, because of the great friction caused by the pressure of the buckstays, the clay will have a tendency to follow the silica in a vertical direction, thereby opening the joints between the clay shapes. This will cause leakage of the burning gases from the wall to the atmosphere, which leakage has caused considerable annoyance and damage, particularly when starting up new batteries.

This difficulty is overcome, in the Roberts oven, by placing on the buckstays a steel angle 34, Fig. 1, that extends over the top of the clay insulating brick. The bottom of the buckstay is secured to the concrete pad, so that the clay shapes are securely held in place and their vertical movement is prevented. The silica slides on the clay without breaking open the joints in either the heating or the insulating walls. The entire surface of the insulating brick remains perfectly bonded and unbroken.

At no time during the heating up or when operating can leaks be detected in this part of the oven. The buckstays remain cool and are unwarped; the alignment of the exterior of the oven is most noticeable. The system of rigidly holding the clay insulating brick in place lessens the labor necessary in keeping the jambs pointed, thus making the luting of the doors much easier.

The door consists of a cast-iron frame tapered to fit the oven chamber. The brick forming the lining of the door are placed in the frame from the back and, being wedge shaped, are firmly keyed into place; the space in the frame behind the brick is filled with powdered insulating material and a steel plate is bolted to the frame. This gives a door that fits in its frame in the oven and has the lining brick firmly keyed so that they will not shift. The insulation prevents radiation and the backs of the doors are so cool that the hand may be placed on them.

The jamb into which this door fits is also of cast iron; the cast-iron jamb is preferred as it can be accurately set and so cemented in place that it will not move or leak. It does not spall, as does the brick jamb, and always presents a smooth surface for luting the door. This care in fashioning the door and jamb prevents leaks around the doors and lessens the labor in luting.

The buckstays used on the Roberts oven are heavy, for the strain placed on them is enormous. In most of the early installations, where the buckstays were of insufficient strength, they bent to such an extent that it was difficult to place the doors in the oven.

The usual practice in the United States is to carry the charging car, or larry, on rails laid directly on top of the brickwork of the ovens. As this car carries from 13 to 16 tons of coal, it sets up considerable vibration in the brickwork of the ovens when moving over the top.

In the Roberts system, the larry is carried on rails laid on top of the buckstays, so that the weight is carried direct to the foundation of the battery and the top is relieved of the shock from a heavy moving load. In addition, the battery top is cleared of all obstructing rails, there is less danger to the men on top from the moving larry, also the rails carrying the current for operating the car are placed on the side, well out of the way.

OPERATION OF THE ROBERTS OVEN

The operation of the Roberts oven is extremely simple and may be readily followed in Figs. 1 and 2. The air enters through the tunnels 30 and passes up the smaller ducts 1 into the air equalizing duct 2. The openings 1 are regulated by dampers 3, which are controlled from the outside and are easily seen through openings left in the ends of the recuperator walls for this purpose. From the equalizing ducts 2, the air passes up around the outside of the recuperator tile. This passage is alternately between the tile and the supporting pier and then between the tile so that each tile is surrounded on three sides by the ascending air. In this manner, the air is thoroughly heated and the tile maintained at an even temperature differential, and at the same time the supporting piers are kept at a temperature no higher than the recuperators.

As the air flows countercurrent to the waste gas, it meets progressively hotter tile as it ascends and, on reaching the outlet ports 4, has attained the temperature of the waste gases at this point. The air then flows through the ducts 6 to the top of the heating wall, horizontally through the passage 7, then downward through the air ports 9 that surround the primary-gas nozzles 10. The quantity of air admitted to the air ports 9 is controlled by the slide brick 8. This brick is reached from the top of the battery through the openings 35 so that, by means of a short iron rod, the operator may accurately regulate the size of the opening 9.

As the recuperator tile is of silica with a high heat conductivity, the air is raised to 2000° F. at the point where it leaves the tile and enters the duct 6. Passing upward through this duct, the air arrives at the tip of the burner at a constant temperature throughout the entire oven. The advantage of having the air reach the burner at a uniform temperature at all times and throughout the entire battery cannot be over-rated as uniformity of heating can only be obtained when conditions governing combustion are uniform. The falling off in air temperature because of the cooling of a regenerator is well known; its effect has been measured and is admitted to be a factor in the heat effect of a reversing oven. The uniformity of preheating the air in the Roberts oven is but one of the points attained by the designers in the effort to eliminate "average conditions."

After the byproducts have been extracted, the fuel gas returns to the battery and then enters a large gas header, which is supported on steelwork attached to the buckstays on the coke side of the battery, occupying on this side approximately the same position as the collecting mains on the pusher side. From this header, the gas is distributed to the individual gas headers 11 which are embedded in the top brickwork so that the top of the battery is smooth and unbroken

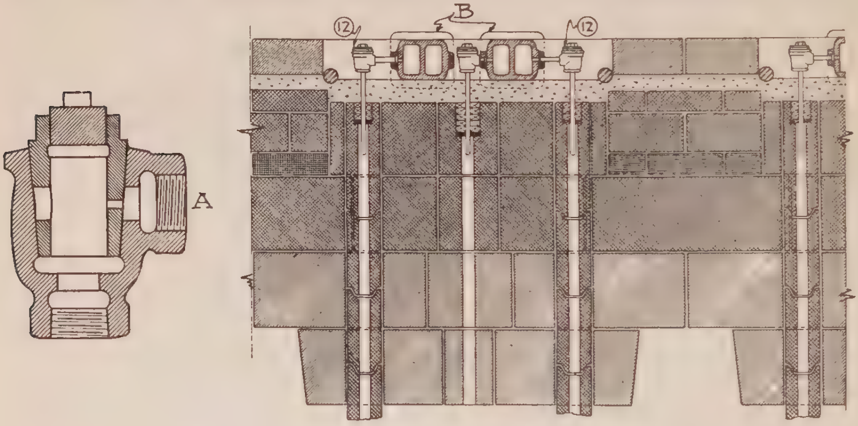

Fig. 5.—Gas cocks for individual burners.

These headers *B*, Fig. 5, are of square cross-section and are divided into two pipes of equal capacity. One side carries the primary gas and the other side carries the secondary gas.

The delivery of the gas to the individual headers is through a manifold, Fig. 6, equipped with shut-off cocks; from the rectangular headers the gas passes to the burners through the burner cock 12. Thus the gas may be shut off from the entire oven or from any individual burner cock. All the cocks mentioned are only for shutting off the supply and not for the regulation, as this regulation is by means of orifices in the manifolds supplying the burner headers and in the body of the burner cocks.

The regulation system of the Roberts oven is based on the fact that the flow of gas through an orifice is proportional to the pressure of the gas. A disk is inserted in each manifold supplying primary and secondary gas just below the shut-off cock. This disk has an accurately drilled hole of such size as to pass the required quantity of the primary and secondary gases. If the amount of secondary gas is to be 20 per cent. greater than that of primary gas, the disk for the secondary gas will have an orifice 20 per cent. larger than the orifice through which the primary gas passes. The proportions of the two gases are accurately known and all the disks are drilled before the ovens are heated and when once set in place need never be changed. To change the coking time, it is only necessary to raise or lower the pressure on the main battery header, and the primary and secondary gas orifices will pass the exact proportions of gas required. The supply of gas to the individual headers is thus correctly proportioned at all times and is independent of setting a valve or cock to what must be only an approximate position.

The gas supply through the individual burner cocks is cared for in a similar manner; a cross-section of the gas cock is shown in Fig. 5. The core of this cock is drilled with two holes set at 180° apart; as the core may be turned through 360°, either hole may be made to register with the gas inlet *A*. If the core is turned to a position midway between the two openings, the gas supply will be entirely closed. In practice, one of the holes is made the full size of the gas inlet but the other is drilled, with extreme accuracy, to the size necessary to compensate for the taper of the oven.

The oven chamber is wider at the coke end than at the pusher end, so that the coke will readily push from the oven. In the Roberts oven, this taper is generally 2 in.; that is, the oven will be 2 in. wider at the coke end than at the pusher end but this taper is uniform throughout the oven as the walls are built without offsets. It is obvious, therefore, that the thickness of coal at the coke end will be greater than at the pusher end; and this thickness will vary uniformly between the ends of the oven, so that it is necessary to burn more gas at the wider end if the entire mass of coal is to be coked in the same number of hours. Also the quantity of gas burned in any part of the oven should be graduated to this increasing taper of the oven from end to end.

The small holes drilled in the core of the cocks gradually increase in size from the pusher side to the coke side of the oven. This increase in size is accurately proportioned to the increase in the amount of the coal charged. As the cores are iron, they may be drilled to the thousandth of an inch and the areas of the holes graduated with great exactness. In the case of a 2-in. taper and a charge of 15 tons per oven, the Roberts oven will have approximately 15 per cent. more coal at the coke end than at the pusher end. The gas burned at the coke end will, therefore, be 15 per cent. greater than at the pusher end of the oven and each intermediate burner will supply the exact amount necessary to coke the coal in the portion of the oven heated by that particular burner.

As the increase due to taper is known, these cocks are drilled before the oven is put in operation and are not changed; for, as is the case with the supply to the headers, all that is necessary to change the coking time is to change the pressure. As the pressure is changed, each burner cock will carry the correct proportion of gas necessary to do the work at that point in the oven. The result is uniformity of heating and the coking of the entire charge in exactly the same time. Such uniformity is impossible without accurate regulation, such as is attained in the Roberts oven.

The vertical-flue reversing type of oven cannot accurately allow for this taper in the oven for it can only be regulated for average conditions. There is an attempt at regulation by increasing the pressure on the coke-side gas header, but as the individual gas nozzles cannot be regulated, the supply to each nozzle is not under close control.

In the Roberts oven, the fuel gas is introduced at numerous points in small, accurately measured quantities, so that the relative temperature of one part of the wall compared to the other is under control, and no point is subject to overheating from the rapid combustion of a large quantity of gas introduced at one point as in other ovens.

Combustion and Flow of Combustion Products

After passing the accurately graduated gas cock 12, the gas is conducted to the burner nozzle 10 where it meets the air flowing through the air ports 9 and combustion starts in the short mixing chamber 13. The gas introduced at this point is called the *primary gas* and is approximately one half of the total supplied to the heating wall. The rest of the gas, called *secondary gas,* is introduced into the secondary-gas ducts 31 through the same type of graduated cock as the primary gas. The secondary-gas ducts pass downward through the center wall to a point 32 about midway to the bottom of the oven and there enter the heating wall.

The initial combustion of primary gas is in the mixing chamber 13, as it is introduced with the total amount of air necessary for combustion of the total gas supplied to that heating wall; that is, this gas will meet twice the amount of air required for its combustion. This large excess of air acts as a depressent to the flame temperature. As the air is not preheated to flame temperature, there will be a reduction in the temperature of the flame corresponding to the amount of heat required to raise the excess air to the average temperature of the mixture of burning gas and air.

The gas and air are also introduced at a neutral pressure so that they mix very slowly, with the result that combustion takes place quietly and evenly. The mixing chamber aids in extracting the small amount of radiant heat in the burning gas. The gas used for heating a coke oven usually contains but a small percentage of the illuminants, therefore, the radiant heat generated is proportionately small.

At the bottom of the mixing chamber 13, the burning gas meets the standard checkered-wall typical of the Roberts construction. The stream of gas in each mixing chamber will here be broken into three parts by the wall brick. As there are 24 primary burners in each heating wall, there will be 72 streams of burning gases in the wall from the bottom of the mixing chambers downward. Such an extremely uniform distribution of the heating gases is not found in any other type of construction.

By impinging on the brick at the bottom of the mixing chamber, the gas and air are more intimately mixed and the extraction of heat generated by this combustion is increased. Sweeping around the brick in the top row of checkers, the burning gas flows downward directly on top of the brick in the next tier below, moving in this way to the bottom of the wall. The result is complete combustion; and as the brick are entirely surrounded by these gases and also present the maximum surface for the absorption of the heat, a high degree of heat extraction is attained.

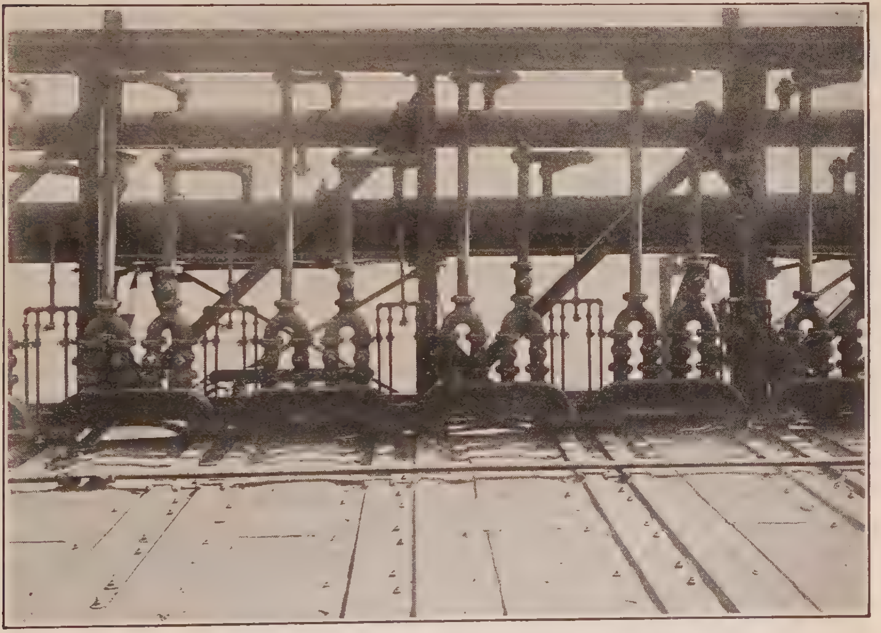

FIG. 6.—GAS AND AIR MAINS.

As a matter of fact, the extraction of heat is so complete that at a point 32 about midway down the wall the primary gases cannot heat the wall to coking temperature. At this point the secondary gas is admitted in the proper proportion to continue the generation of heat in the lower portion of the wall. Sufficient air is introduced with the primary gas to support also the combustion of the secondary gas; the oxygen in this air will, therefore, be available for combustion with the secondary gas at its point of introduction. At this point the temperature of the secondary gas is high, also the temperature of the waste gases from the primary combustion. The combustion of the secondary gas, however, will be subdued by the high proportion of inert gases present. These inert gases are the carbon dioxide and water vapor from the primary combustion and the nitrogen present in the air introduced with the primary gas. The combustion of the secondary gas will, therefore, be quiet and even. As this combustion takes place directly in the same checker brick of the wall, it is complete, and the extraction of the heat is as perfect as in the case of the primary gas.

As nearly all the available heat is extracted from the waste gases by the time they reach the bottom of the walls, the two streams are combined by the ducts 14 and this combination produces sufficient heat to maintain the sole of the oven at coking temperature. The coal in the sole of a Roberts oven is coked in a vertical direction by this method. The heat is thus extracted from the burning gases to such an extent that on reaching the lower sole flue 15 the gases contain only sufficient heat to be used for preheating air.

The upper sole flues 16 are connected to the lower sole flue 15 by six openings, each of which is controlled by dampers 17. By these dampers, the differential in the heating wall is maintained uniformly from end to end. These dampers are readily reached through openings left for this purpose.

From the lower sole flue 15, the waste gases pass down through 18 and through the top pass of the recuperators to the downcomer 19, then through the lower pass into 20 and offtakes 21 to the waste-gas tunnel 22. The waste-gas offtakes 21 are equipped with butterfly dampers 23 by which the draft for the oven is regulated; these dampers are readily set from the exterior of the offtake.

The ideal sought in any coke oven is the completion of the coking of the entire charge at one time; this can only be attained when there is uniform distribution of heat from the top to the bottom of the oven and a gradual distribution of heat from end to end to compensate for the taper of the oven and the increased thickness of the charge toward the coke side because of this taper.

The progress of heat, and consequent heating in an oven, in the vertical planes should be parallel to the walls of the coking chamber and the rate of this progress should be proportional to the thickness of the coal charge between the wall and the center of the chamber. The walls of the chamber are perpendicular, therefore, the thickness of the charge will be practically the same throughout its height, but the thickness will vary from the pusher end to the coke end in proportion to the taper of the oven.

Temperature in the Oven Top

The temperature of the space above the coal charge should be as low as possible, for it is through this space that the gases distilled from the coal pass on the way to the collecting mains. These gases are composed of hydrocarbons, some of which are readily broken down by heat; this breaking down results in free carbon and the destruction of valuable byproducts. The initial application of heat in the Roberts oven is at a point well below the top of the coal charge. The combustion is then downward, so that the tendency to overheat the top is eliminated. The temperature of the upper zone of the chamber may be changed by changing the amount of primary gas burned; this is done by changing the size of the orifice or the gas pressure.

GENERATION OF THE HEAT

The heat necessary for coking the coal is generated by burning gas in the walls of the ovens. Usually this is a lean coke-oven gas of about 525 B.t.u. per cu. ft. Sometimes producer gas is used, but then it is necessary to preheat the gas as well as the air in order to maintain the necessary flame temperature.

The principal factors governing the burning of a gas are the ignition temperature and the rate at which the particles of combustible matter in the gas combine with the oxygen present. This rate is dependent on the quantity of oxygen present and its dilution by inert gases, and the velocity of the gas and air streams during combustion.

Ignition Temperature

All coke ovens operate at a temperature sufficiently high to ignite the gas. Combustion will, therefore, be sustained as long as air and gas are admitted to the walls. The temperature at the point of admission will, however, have considerable effect on the rapidity of the combustion. If the temperature is high and there is just sufficient oxygen to combine with all the combustibles, a high flame temperature will result and the evolution of heat will be rapid.

Rate of Combination of Gas and Oxygen

The lean coke-oven gas generally used to heat up an oven ordinarily contains a high percentage of hydrogen. There is also a high percentage of methane, which, on burning, decomposes, producing more hydrogen. The calorific power of hydrogen is not great but it is rapidly combustible and produces a high flame temperature.

The introduction of this gas into a highly heated chamber, such as the heating wall of a coke oven, with the theoretical amount of highly heated air necessary to burn it, produces a high local temperature. If, in addition, the gas and air have a comparatively low velocity, the mixing will proceed at a high rate with the evolution of large volumes of heat.

The rate at which the gas is introduced (the quantity per unit of time), sufficient air for combustion being present, will determine the rate at which heat is generated. In those ovens, both vertical- and horizontal-flue types, having but a few points for the introduction of the gas, the quantity of gas burned at each point will be far greater than in an oven using many points of introduction. In the Roberts oven, there are 96 points of gas introduction while most ovens of the flue types introduce gas at not more than 16 points. It is obvious that, in the same coking time and using the same quantity of gas per ton of coal carbonized, the oven using but 16 points for the introduction of gas will burn six times as much gas at each point as the Roberts oven will burn at each of the 96 points. The danger from local overheating in the latter oven is further lessened by the method of transmitting the heat to the coal.

The temperature produced by the combustion of the gases sets up a heat flow that tends to balance the temperature difference between the heating gases and the oven wall with which they are in contact. This balance is influenced by the area of the wall in contact with the heating gases and the rapidity with which the heat is transmitted through the wall to the coking mass. In the Roberts wall, the area for the absorption of the heat is two and one-half times as great as the area that distributes this heat to the coking mass. It is believed that the Roberts oven will transmit the heat generated by the combustion of the heating gases almost twice as rapidly as is done by other types.

Air Control

The air supply in the Roberts oven is controlled by the damper 3 at the entrance to the recuperators and by the slide brick 8 over the air port 9. By means of the damper 3, the flow of air through the recuperator is equalized so that all parts will receive the same amount of air to preheat; this assures uniform temperatures in the recuperator and so reduces movement of the brick as the result of expansion and contraction. These dampers also regulate the quantity of air supplied to each oven. In other types of ovens, the air port is used first for the passage of air and, on the reverse, for the passage of waste gases. The quantities of air and waste gas are not the same, therefore, the conditions under which they should be governed are different.

In the Roberts oven, all the air is introduced at the top of the heating wall with the primary gas. As the latter is approximately only one half the gas required in the oven, there is no question of there being sufficient air for perfect combustion of the primary gas. The control of the heat generated by the primary gas is, therefore, dependent on the adjustment of the primary burners alone.

After leaving the mixing chambers, the burning gases enter the zone of checked brick typical of the Roberts oven. As this portion of the wall is practically one single chamber with inter-communicating passages, the air and gas are free to move from one part to the other so that not only is combustion of the primary gas complete but the secondary gas will come into contact with sufficient air to make combustion in the secondary zone complete. The control of the heat generated by the secondary gas is, therefore, dependent only on the amount of secondary gas and this control is consummated with the same accuracy as in the case of the primary gas.

Control of the Heating Gases

The control of the heating gases in the heating walls plays an important part in the distribution of the air, and this movement is controlled by the distribution of the draft. The regulation of the draft or differential (that is, difference in pressure between the point of admission of the gas and air and the pressure of the outgoing waste gases) through the various parts of the heating walls of the ovens is of great importance, because, in conjunction with the air slide regulation, it determines the volume of air introduced also the velocity of flow and distribution of the heating gases through various parts of the combustion chambers.

The Roberts oven is clear of controls and obstructions from the point of admission of the gas and air to the point where the waste gases leave the recuperators. The controls are located at the points of admission of the gas and air and can be accurately adjusted. The gases are unobstructed in their flow after their admission, except for the accurately designed motion through the wall. The products of combustion then pass through the damper-controlled openings that regulate the distribution of draft. These dampers 17 are easily accessible and may be readily set to obtain the proper distribution of draft. The essential feature of this regulation is the maintenance of the proper differential from end to end of the oven necessary to move the graded quantities of heating gases; the maximum differential is maintained at the coke end and the minimum at the pusher end.

Gas Control

The method of controlling the gas supply to the headers and individual burners has been described. The effect of this control and the distribution of the gas through many burners may be summed up as follows:

1. The number and arrangement of the points of introduction gives accurate control of the heat at all points in the oven walls.

2. The quantity of fuel gas introduced at any one point is reduced to a minimum, and the amount of heat generated in a restricted space is proportionately small.

3. The arrangement of these burners is such that a relatively large area of brickwork is exposed to the gases for the absorption of heat as it is generated.

4. There is 100 per cent. excess air on the basis of the air required for the fuel gas introduced at the top or primary burners, which serves to temper the flame temperature, the entire body of gases in the wall absorbing heat and tending to attain the same temperature.

5. The combustion of the secondary gas takes place in an atmosphere containing a high percentage of inerts (produced by the combustion of the primary gas) with the production of a low flame temperature at the point of introduction of the secondary gas and a progressive combustion with a sustained heating effect as the gases pass down through the wall.

6. The introduction of the gas from the top, by means of the evenly distributed ducts, serves to reduce the temperature of the brickwork in the top of the ovens protecting the products of the distillation from decomposition during their travel through this portion of the oven.

Extraction of Heat from the Heating Gases

One result of introducing the gas in small quantities at many points is the production of a very small amount of radiant heat. The gas used is a lean coke-oven gas with a low percentage of illuminants. This radiant heat is extracted in the short duct at the top of the wall in which the initial combustion of the primary gas takes place.

The transmission of the sensible heat of combustion is by conduction, which will take place more rapidly if the particles of the burning gases come directly in contact with the brick. In the flue structure, only part of the gases come directly in contact with the brick, the rest sweep past the flue walls some distance from the surface of the brick so that their heat can reach the brick only by conduction through the outer layers of gas. Practically, all gases are poor conductors of sensible heat so that there is poor extraction of the heat generated by the combustion.

In the Roberts oven, the heating gases are brought into actual contact with the brick in the heating chambers. They impinge directly on top of the wall brick, slide off at a slight angle, and drop to the tier below where they again impinge directly on top of the brick in this tier. By this method, the brick is entirely surrounded by the burning gases and each part of the brick will receive a uniform amount from them. The effect is the same as the baffling of the tubes in a boiler or the checker brick in a regenerative chamber. The passages for these streams of gases are but 3 in. wide, whereas in the flue-type oven the flues are often 15 in. or more in width.

Transmission of Heat to the Coal

There are two principal factors in heat transmission through a solid medium: The temperature differential and the conductivity of the transmitting medium.

The generation and extraction of heat in the Roberts oven has been so well worked out that the heat is supplied to the heating wall progressively and it is transmitted to the coal at the same rate, thereby maintaining a constant temperature differential between the combustion chamber and the coking mass without any tendency toward excessive temperature in any zone.

The production of heat is uniform from top to bottom of the wall and from end to end of the oven. As the wall is of uniform thickness, the transmission of heat will be uniform in all parts. The absorption of heat and its transmission to the coking mass will also be proportional to the surface exposed to the heating gases and the ratio of this surface to the surface in contact with the coal. From the shape of the brick and the method of laying them, a far greater surface is exposed to the heating gases than is possible with a flue construction.

Recovery of Heat from the Waste Gases

The recovery of the heat from the waste gases, after leaving the heating walls of the oven proper, is accomplished in three ways: (1) By waste-heat boilers. (2) By regeneration, (*a*) common regenerators; (*b*) individual regenerators. (3) By recuperation, (*a*) common recuperators; (*b*) individual recuperators.

The amount of heat reclaimed from the products of combustion is dependent entirely on the temperature to which the gases going to the stack can be reduced, and at the same time have them sufficiently hot to eliminate them from the oven system without the aid of auxiliary power. The choice between regenerative and recuperative settings is a matter of individual opinion.

Heat-effect Curves

The object of heating a coke oven is the effect on the coking mass and the more uniform the application of heat the more uniformly will the coking be carried on. To produce coke with uniform cell structure, the heat must be applied evenly and continuously throughout the entire oven. Heat-effect curves of the Roberts oven are shown in Fig. 7.

A section of the oven showing the points of admission for air and primary and secondary gas is illustrated at (*a*). Curve *AA* is the theoretically perfect curve for the application of heat to the coal charge; curve *BB* is the curve produced in the Roberts oven. Curve *GG* represents what would occur if all the gas and air were admitted at the top of the combustion space.

Curve *DDEE* is what would be expected if the gas and air were introduced at two points with just sufficient air at each point for complete combustion and this combustion took place in a flue. The dotted portion of curve *BB* represents the rise in temperature as the gas passes through the top of the oven to the mixing chamber. At this point, the gas meets the highly heated air, the volume of which is sufficient for the combustion of both the primary and the secondary gas. There is, then, a great excess of air over that required for the primary combustion, as a result the flame temperature is depressed and the curve flattens out as shown from *L* to *M*. At the point *M*, the burning gases meet the checkered construction, and the curve continues in a practically straight line to the point where the secondary gas is introduced.

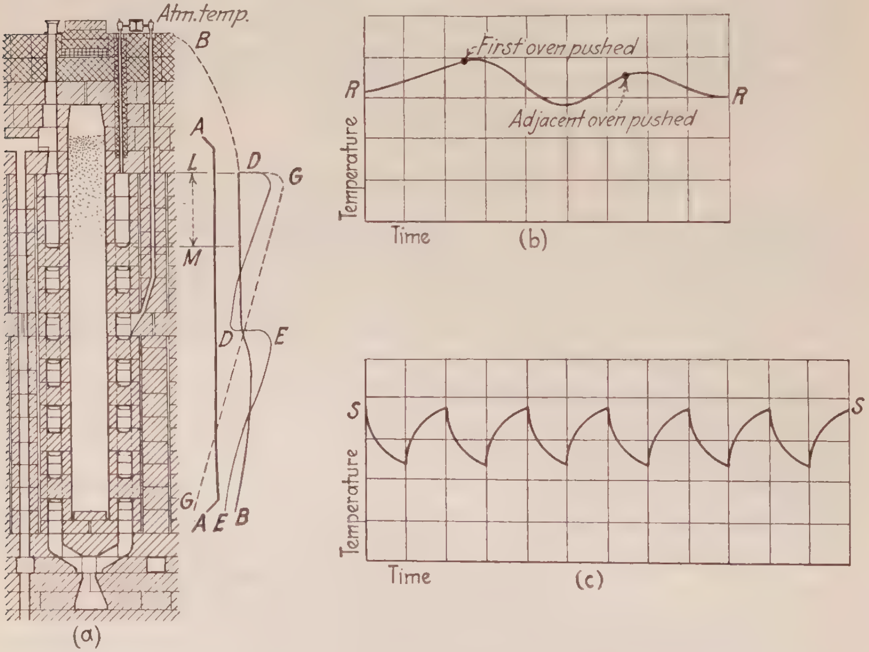

FIG. 7.—HEAT-EFFECT CURVES.

At this point, it might be supposed that the introduction of the secondary gas would make the curve take the form *EE* but, as this gas is introduced into an atmosphere containing a high percentage of inert gases resulting from the primary combustion, the reaction between the combustibles and the oxygen present is subdued and the generation of heat is continued along the line of curve *BB*. A tendency to follow *EE* is prevented by the baffled construction.

There is, then, in the case of the primary gas, a reduction of the heat effect of the initial combustion because of the large excess of air present. Beyond the point *M*, the heat effect is strengthened by the baffled-wall structure. The initial combustion of the secondary gas overlaps the final combustion of the primary gas but is subdued by the inert gases present, and the heat effect of the latter part of the secondary combustion is strengthened by the baffled-wall structure. The curve *BB*, which has been accurately checked by thermocouples, therefore, follows closely the ideal curve for heat effect. It must be remembered that this heat effect is applied continuously, for there are no reversals with the attendant fluctuations in temperature as shown at *RR* and *SS* (*b*) and (*c*), which are curves taken on reversing ovens. The effect of the reversals is evident.

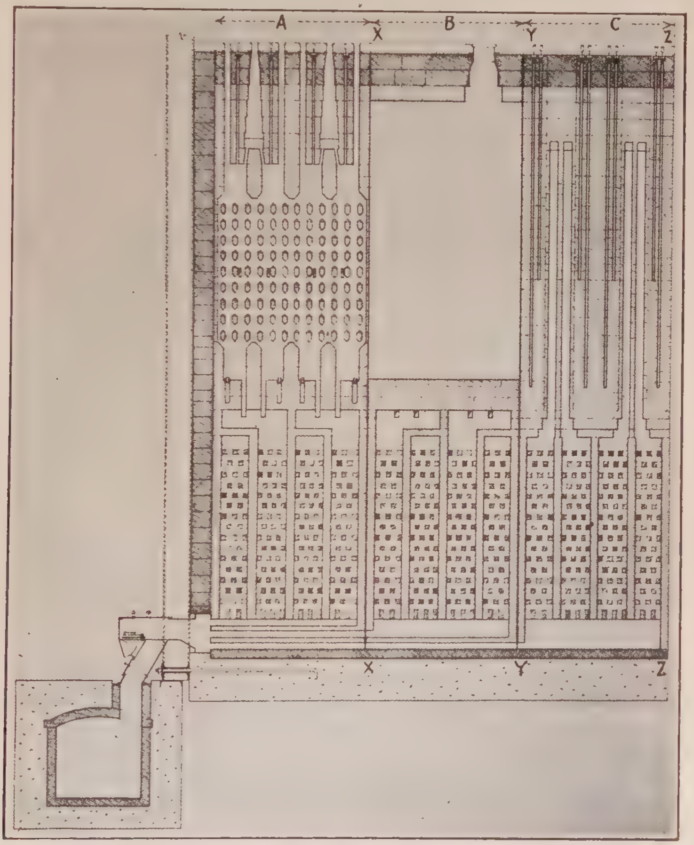

Fig. 8.—Construction of sectional furnace. *A*, section through combustion wall *XX*; *B*, section through oven *YY*; *C*, section through center wall.

Roberts Individual Regenerator

The adaption of individual regenerators to the Roberts oven is possible in most forms of common practice. According to Fig. 8, which is the longitudinal section of the oven illustrated in Fig. 9, a new principle in design is incorporated in this oven regardless of whether the oven is recuperative or regenerative. This plan makes it possible to cool the

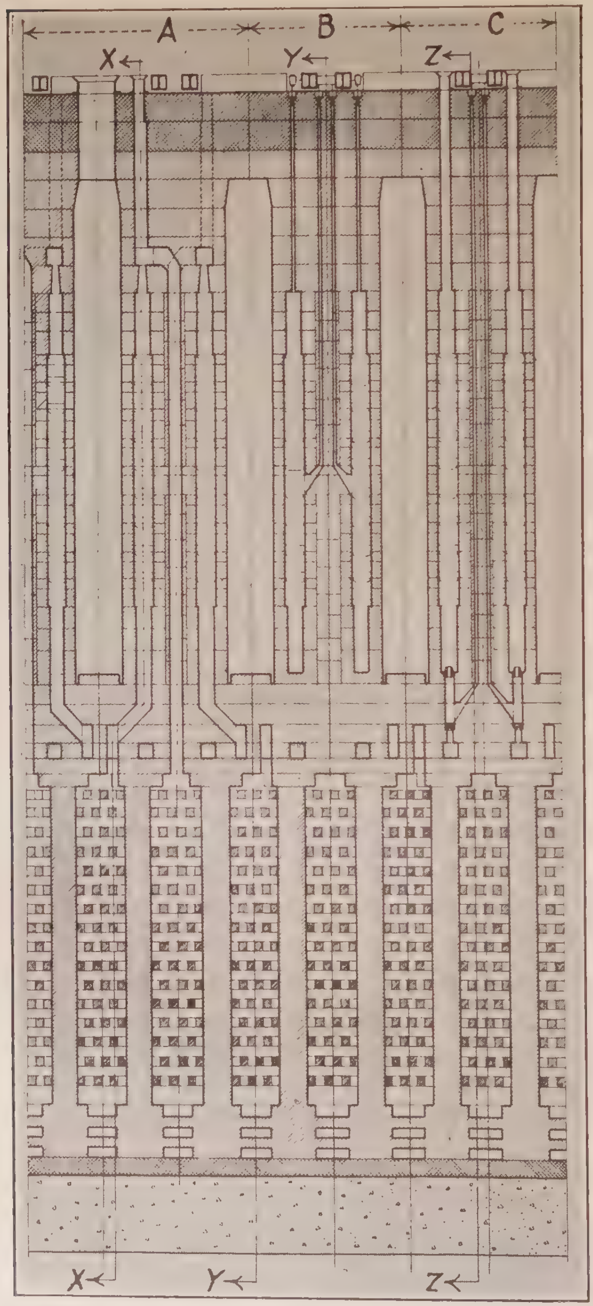

FIG. 9.—ROBERTS INDIVIDUAL REGENERATOR. *A*, SECTION THROUGH PASSAGE FROM REGENERATORS; *B*, SECTION THROUGH SECONDARY AND UPPER PRIMARY BURNERS; *C*, SECTION THROUGH LOWER BURNERS.

ovens to atmospheric temperature without rupturing the brickwork after it has been at working temperature.

The expansion of silica brick, such as is used in American coke-oven practice, is about ⅛ in. per lin. ft., when heated to 2600° or 2700° F.; therefore, a coke-oven wall 40 ft. long, when heated from atmospheric to working temperature expands 5 inches.

If it is necessary, because of the desirability of closing down the plant, to have the oven go back to atmospheric temperature, this wall must contract approximately 5 in. As the brick is not strong enough to withstand this contraction, shrinkage cracks result; and as there is no way of controlling the ruptures thus caused, they are irregular and form passages between the air and gas flues and between the coke-oven chamber and gas flues.

To overcome this fault, one type of Roberts oven is so designed that it can be built in sections; that is, it would be equivalent to a number of short ovens placed end to end for the required length of a complete unit. Each section can contain three or four burners and be complete within itself. As these sections are about 7 ft. long the total expansion of each section will be ⅞ in.; and as the brick will expand equally in all directions, the expansion from the center to either end of a section will be 7⁄16 in. Experience has shown that a coke-oven wall can expand and contract to this extent without damage, so that with this sectional construction a wall may be repeatedly expanded and contracted without developing cracks that result in operating difficulties.

This method of construction is illustrated in Fig. 8. The expansion joints are indicated at *XX*, *YY* and *ZZ*. These joints are intercommunicating between adjacent ovens, but are bulkheaded off from the sections in which are incorporated the combustion space and also from the center wall.

While the utilization of expansion joints in this manner permits circulation back and forth between adjacent ovens, when the ovens are first charged, this condition exists only a short time because the expansion of the brickwork will tightly close the joints and after the oven has been charged a few times any small leaks will be hermetically sealed by carbon deposits. The coking chambers will then be completely isolated from one another.

The other features of this oven are not different from those of the recuperative type. The primary difference, of course, is the admittance of gas into the heating wall at three points, the top, middle, and bottom. The fuel gas is admitted from the top and carried to intermediate and bottom burners through ducts, the same as the secondary gas is put into the wall in the recuperative oven.

In the reversal of the gas flow, this oven utilizes the dividing wall for eliminating the waste gas from the heating wall when the oven is burning in an upward direction; and, vice versa, the waste-gas ducts become air ducts when the gases are burning downward in the heating wall.

The arrangement of the checker brick in these regenerators is optional. They can be divided into zones or made two pass or single pass in a horizontal direction.

The possibility of keeping the gas ducts, particularly the secondary-gas ducts, free of carbon from the decomposition of the fuel gas passing through them is often questioned. Generally, these ducts are of such diameter that it would take many hours continuous flow of rich hydrocarbon gas to decompose sufficient hydrocarbon to cause a stoppage, and it has been found that by shutting off the gas supply in these secondary ducts at regular intervals and running air through in them for 15 to 20 min. all carbon accumulations on the walls are burned out. The common practice is pass air through these ducts once every 8 hr. When using lean, debenzolized coke-oven gas, once every 24 hr. is sufficient to keep the ducts free of carbon. The ovens are provided with an auxiliary air header (the upper pipe, Fig. 6), which contains air at atmospheric temperature at 1 lb. pressure, supplied by a rotary blower.

RESULTS OBTAINED WITH ROBERTS RECUPERATIVE OVEN

A plant of eighty ovens of the recuperative type has been in operation since January, 1921. During this period, these ovens have run continuously with more than satisfactory results. The plant was designed to operate on 15-hr. coking time, but when the market would absorb the products it has been operated continuously on 12-hr. gross coking time, using either Illinois or Indiana coal exclusively, or mixtures of the two coals. The resultant coke has been used for the usual purposes, such as the operation of a 500-ton blast furnace, lead smelting furnaces, water-gas practice, and foundry cupola work.

It is believed that this character of coal is not used in any other by-product plant for the production of metallurgical coke. These coals have been used in other plants, when mixed with coking coals in which the coking coal in the mixture predominated. In the Roberts ovens, the Illinois coal has always formed the larger part of the mixture and when the Illinois coals low enough in ash and sulfur could be secured, they have been used successfully without the mixture of any coking coals.

The average operating results in this plant are as follows:

Average quantity of coal per charge per oven, tons	14.5
Average coking time, hours	12
Average coke yield, per cent	67.25
Average breeze yield, per cent	3.7
Average tar yield, gallons	8.5
Average ammonium sulfate yield, pounds	27.6
Average total gas yield, cubic feet	10,284
Average surplus gas yield, cubic feet	4200
Average B.t.u. rich gas	593
Average B.t.u. lean gas	508
Average quantity of coal per oven per day, tons	29

The benzol plant was put in operation on Oct. 1, 1922, and has produced an average of 2.9 gal. of motor fuel per ton of coal.

The blast furnace operated in conjunction with this plant has produced approximately 260,000 tons of pig iron on an average coke consumption of 1806 lb. The ovens have been operated with the normal crew that would be used on any type of coke oven and have shown no operating difficulties and, after practically two years of continual operation, show no inherent weaknesses from a structural or operating standpoint. There has been no charge for repairs or maintenance on the ovens and, as far as it is possible to judge, the brickwork is in as good condition as when the oven was started.

It is thought by the owners of this plant that the only replacement necessary on the ovens will be the replacement of the false bottom in the sole of the oven; this work is easily done. These false bottoms are 3 in. thick, made of silica slabs, and are held in place in the oven by the curbs on each end.

There has been no difficulty in keeping the gas ducts clean and clear of obstructions, including carbon.

The relative value of recuperators and regenerators is a matter of individual opinion. In the operation of the plant mentioned, it is still a question whether the recuperators are as economical of gas in the operation of the ovens as regenerators would be. It is impossible at the present time to get a comparison of the amount of heat required to carbonize this character of coal as compared to the use of 100 per cent. coking coals. However, if the question of costs is eliminated, recuperators made of silica brick can be constructed that will be as efficient in heat recovery as regenerators. The men operating this plant favor the recuperator because of the elimination of the reversing feature and continuous flow of heat in one direction. As most of these men have had considerable experience in the operation of the reversing type of oven, considerable weight should be given their opinion.

While it is true that more brick is used in the construction of this type of oven than in the single-wall type, the added cost is offset by the greater capacity of the oven. Several demonstrations have been made with these ovens in which it has been proved that a 14-in. oven of this construction can be operated continuously on 10.5-hr. gross coking time. As the tendency in the United States for the last few years has been to build ovens with greater cubical content and operate them at faster coking times, based on the results obtained by the use of the Roberts oven, it is believed that it will be only a short time when ovens that will carbonize 20-ton charges of coal in from 10 to 12 hr. will be in use.

DISCUSSION

M. W. Ditto.—Since this paper was prepared, the plant at Granite City has been operated to approximately its maximum capacity. As a result the gross coking time has been reduced from 11 hr. and 50 min. to 11 hr. and 30 min. During November, December, and January, one of the batteries has been operated exclusively on 100 per cent. of Illinois coal, and the blast furnace has been operated with the resultant coke, its average production having been about 500 tons per day.

Since the superintendent of the blast furnace has learned more about the characteristics of coke made from Illinois coal, the use of this coke does not seem to have affected the operation or capacity of the blast furnace in any way. The principal objection he has had to this coke has been that we have been unable to control the ash and sulfur content. This is true principally when using coals produced outside of Franklin County, Ill. From coal produced in this and contiguous counties, it is possible to keep the coke under 1.35 per cent. sulfur and about 12 per cent. in ash so that there has been no difficulty in producing basic iron.

Probably the most important factor, in the Roberts oven, affecting the ability to make coke out of the Illinois type of coal, is the design of the wall structure. While it is difficult to prove, we have reason to believe that there is more heat available per unit of time for the coal in contact with the oven wall than there has been in the flue type of oven. Men in the coke-oven industry have been inclined to attribute the results obtained in the Roberts oven to the width of the chamber rather than the form of the brick used in the heating wall. The coking chamber has an average width of 14 in., its maximum width being 15½ in. The only way we have been able to prove that the width of the oven does not affect the results, is by coking the coal by the heating of one wall during the coking period; that is, the heat has been shut off from one wall. This is equivalent to coking an oven from 26 to 27 in. wide. The residual heat in the wall not operating is sufficient to coke the coal to a depth of about 2 in. so that the dividing line that normally is in the center of the chamber is on the side about that distance from the unheated wall. The only apparent difference in the coke is that the farther the coal is from the heated wall the larger and weaker is the coke cell structure. This result is also apparent in some of the extremely wide ovens that have been installed at various points in the United States; and a spongy coke structure is more characteristic of a wide oven than of a narrow one.

The other interesting feature of this installation of ovens is the heat-recovery element in the form of a special type of recuperator. At the time the installation was made, it was thought that there should be a continuous flow of heat in one direction; that is, there would be no fluctuation in the temperature of the walls during the coking period. Since the construction and operation of this plant, we are convinced that as good results could have been obtained with regenerators.

As this was the first installation of its kind, naturally there were some unknown factors. The principal one was the temperature of the gases leaving the recuperators. It was estimated that to carry the gases through the oven system it was necessary to discharge the gases from the recuperators at 750° F., when there would be between 4200 and 4500 cu. ft. of surplus gas per ton of coal carbonized. When operating on a 15-hr. coking period, we have been able to obtain more than 4200 cu. ft. of surplus gas per ton of coal treated, when using Illinois coal. We are convinced that this coal requires more heat for its conversion into coke aside from the fact that it averages about 10 per cent. moisture. The surplus gas recovery is also affected by the fact that the plant has used from 80 to 90 per cent. slack and screenings throughout a large part of its operating period, and they do not produce as much gas as fresh run-of-mine coal.

Before the Illinois type of coal is widely used for the production of metallurgical coke, greater care must be used by the producers to eliminate part of the ash and sulfur at the mine. Although we operate one mine in Williamson County, we buy a great deal of coal from other producers and are compelled to use considerable care in grading and mixing in order to get an average ash and sulfur content that will come below the blast-furnace requirements. This plant produces about 1500 tons of coke per day. That which is not used in the blast furnace is screened and the largest sizes are sold in the smelter and foundry trade and the smaller sizes in the domestic trade of both Chicago and St. Louis.

There is more or less leakage in a recuperator regardless of its type or design. In this instance, the Roberts recuperator was built entirely of silica brick on the assumption that this material, having the same coefficient of expansion and contraction as the upper part of the oven, would lessen the tendency to leak. This has proved true. As the temperature of the off gases in the last pass of the recuperators is kept above 500° F., the variation in the expansion of the upper part of the recuperator compared to the lower part is very small. As this recuperator is installed absolutely independent of the oven structure, the recuperator tile can be entirely replaced without affecting the brickwork of the oven proper.

This plant was designed to operate on 15 hr. coking time and produce from 4200 to 4500 cu. ft. of surplus gas per ton of coal; this we have practically accomplished. When operating with a shorter coking time, the amount of surplus gas decreases, for more coal is being treated and more fuel gas is being used. As a result, the velocity of the gases and air through the recuperator is considerably increased and there is a relatively lower total heat recovery. To offset this, if very wide ranges of coking time were to be used it would be necessary to design the recuperator for the shortest coking time; this might be excessively large for the longer coking time.

The result of the work done so far in the use of recuperators would influence us against their further use if costs of construction are taken into consideration. However if ease of operation only is considered, the recuperator is entitled to favorable consideration. As the one-way flow of gas through the heating walls is not essential in the coking of Illinois coal, the regenerative construction would be cheaper and result in higher gas economy.

Another interesting feature of the work with Illinois coal is that the temperature of the coke-oven wall apparently has nothing to do with the coking of this type of coal. These ovens have been operated at anywhere from 48 hr. to 11 hr. coking time; and while there is a change in the characteristics of the coke, the temperature of the brickwork in the wall, as measured with pyrometric instruments, does not vary in the same ratio as the coking time. In other words, so long as there is uncoked coal in the center of the oven to maintain a high potential difference in temperature, there is little change in the temperature of the wall between the slow coking time and the fast coking time. The principal difference noticeable is that the temperature of the wall increases much more rapidly when the coking is finished on a fast coking time, than it does on a slow coking time. When operating on 11-hr. coking time, if the ovens are pushed promptly when the carbonization is complete, the temperature of the wall seldom rises above 1850° F.; there is an increase, however, in the temperature of the combustion chamber. Whether the checkerlike structure of the heating wall has anything to do with this or not, I am not prepared to state. A few years ago Mr. Blauvelt of the Semet-Solvay Co. stated that by increasing the velocity of the gas in the horizontal flue the efficiency of the heat transmission was increased. This same thing is true in the Roberts wall, but instead of having high velocity of gas through the heating wall, the interruption of the flow of these gases gives the same effect on account of the impingement, which permits the rapid transference to the brick of a larger amount of heat by conduction and convection.

R. H. Sweetser, Columbus, Ohio.—From the data on page 506, if the average width of the oven 14 in. is divided by 12, 1.16 in. is coked per hour, is that the average coking rate?

M. W. Ditto.—I would not say that was the average. The average was taken over various periods of coking. In December and January, the average coking time was 11 hr. 30 min. gross; that is, from the time the oven was discharged until it was charged back in line again. On this basis the coking rate would be considerably higher. On a 14½-in. oven, it is possible to reduce the coking time to 9 hr. We do not consider this good practice, for if there are delays in pushing the wall temperatures may go above the critical point and be destroyed.

The general term "coking time" does not mean a great deal. This time is dependent on many conditions: such as thickness of charge, thickness of heating wall, prompt discharging of ovens when coal is completely devolatilized, etc. It does not mean that fast coking time is dependent solely on the temperature at which the heating wall is maintained.

J. V. Freeman, Joliet, Ill.—From what mine, county, and seam was this coal obtained? What percentage of water did it contain?

M. W. Ditto.—We have made no effort to procure the coal from any one mine, but have tried to stay inside of what is known as the low-sulfur zone of southern Illinois. Probably we buy more coal from the Old Ben Corporation than anyone else. We operate the mine known as the Black Briar property in Williamson County; but, since the strike was settled, we have been able to buy coal screenings cheaper than we can produce run of mine and can sell our lump coal at a higher price than we pay for screenings. We have had coal from about all of the larger producers throughout Franklin County and contiguous territory.

William H. Blauvelt, New York, N. Y.—Is there any difference in the physical structure of coke made from straight Illinois coal, outside of the chemical composition, compared with coke made from coking coals we are accustomed to regard as standard? What is the practicability of closing down and cooling off the Roberts oven?

M. W. Ditto.—I do not consider myself an authority upon relative value of coke from a structure standpoint, but as a result of the blast-furnace practice of using this coke, we feel convinced it is different from the so-called standard cokes. Most of this difference seems to be due to its free-burning qualities; some tests run by the Bureau of Mines seem to confirm this conclusion. Apparently, the reason for this is the fact that the cell structure of coke made from Illinois coal is more open and uniform while retaining the burden-bearing qualities. The furnace superintendent states that his experience with this coke in the furnace indicates that it can be burned at a faster rate than the cokes he used in some of the eastern furnaces. This is confirmed by the fact that, in the last two years of operation, the temperature of the gases from the top of the furnace have seldom gone above 300° F. This indicates that the zone of combustion is confined below the mantle of the furnace.

The fact that this coke is lighter per cubic foot than that made from coking coals from West Virginia, Kentucky, and Pennsylvania, also indicates that there is a difference in structure. The faster the coking time on this character of coal the smaller the size of the coke. It breaks down to from 1 to 2½ in. in size and is noticeably free of braize and coke dust. Over 200,000 tons of iron have been produced with coke consumption under 1900 lb. per gross ton of metal.

The furnace using this coke operates on an average of about 14 lb. pressure and from 33,000 to 34,000 cu. ft. of air per minute. The cubical content of the furnace is 19,500.

Our experience at the Dover plant indicates that after an oven of this type has been cooled down a couple of times, the cracks in the brickwork are of such nature that it is practically impossible to expand back tight. This we believe is characteristic of silica brick regardless of the design of the oven.

We are now planning to build at Granite City some test ovens. A 42-ft. oven will be composed of six 7-ft. sections, each section of which will expand and contract only about 7/8 in., instead of from 5 to 7 in. as when the wall is built in one mass. By this construction we believe it will be possible to cool and heat ovens rapidly without much risk of rupture; and if some of the sections should rupture it will be necessary to replace only that particular section.

Carl A. Wendell, Washington, D. C.—If the efficiency of an oven depends entirely on the brickwork (that is, the size of the flues and thickness of the brickwork) would not a construction where the flues are made entirely in the coal mass itself be more efficient for making coke?

M. W. Ditto.—I would not attribute the overall efficiency of a coke oven to the thickness of the brick in the combustion chamber. But applying the rule of resistance of heat through a mass, the thinner the wall the higher is the transmitting efficiency of a given material. In this wall, the head of the brick is only 3½ in. thick.

Carl A. Wendell.—That answers the question: the thinner the brickwork the better is the efficiency. If there was no brickwork, so that the flues were made in the coal itself, that construction would be more efficient.

Heat Distribution in New Type Koppers Coke Oven

By Jos. Van Ackeren,* Pittsburgh, Pa.

(New York Meeting, February, 1923)

Although the Siemens regenerator principle was introduced into byproduct coke-oven design about 40 years ago, many problems of construction, and particularly of heat distribution and pressure conditions, have not been fully solved.

Heat should be applied to the walls of a coke oven in such a way that the coking of the charge of coal will be completed everywhere at the same time, and at approximately the same temperature. This means that the application of heat should be uniform from the top to the bottom of the coal mass (except for the thin layer at the top that is purposely kept at a somewhat lower temperature to reduce cracking of the byproducts); also that a little more heat must be applied to the coke end than to the pusher end, because the coal mass is a little thicker at that end. As a modern coke oven is about 40 ft. long and 12 ft. high, a real problem is involved in the proper application of heat to the oven walls so as to secure quick coking time (without endangering the walls through local overheating) and produce coke made at a uniform temperature.

Design of Ovens

In all regenerative ovens, the points of combustion are arranged in duplicate sets, one set operating in one-half the area of the heating wall and the other set alternately operating in the other half. Because of the reversal of the flow of gases, the points of maximum and minimum temperatures are alternately located at the same places, thus preventing dangerous local overheating.

The practicability of the principle of reversal of flow seems to have been recognized at an early date, and has been applied with every conceivable flue arrangement, combustion chamber, and means of air and gas supply. To summarize the various principles that have proved useful for the control of oven-wall temperature, the best practice seems to have resulted from the acceptance of the use of vertical flues; the separate supply of air and gas to the base of these flues, with the resulting flame burning upward; the reversal of flow. This last principle involves the problem of distribution, which is of a basic nature.

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*Superintendent, The Koppers Co., Pittsburgh, Pa.

DISTRIBUTION OF GASES IN HEATING WALL

When a gas flowing through a duct toward a closed end discharges through uniform side outlets, the quantity discharged by individual openings varies from the inlet end, where the discharge is least, to the opening nearest the closed end, where the quantity discharged is greatest. This difference in discharge from point to point may be closely calculated by Bernoulli's theorem, the mathematical expression of which, for this case, is as follows:

$$ph_1 + vh_1 = ph_2 + vh_2 + Lh \tag{1}$$

where
ph_1 = initial pressure head;
ph_2 = final pressure head;
vh_1 = initial velocity head;
vh_2 = final velocity head;
Lh = loss of head (due to friction, etc.)

This law, which is extremely simple, is based on the fact that energy is indestructible but can be converted into various forms.

From this law, if the velocity of a gas flowing through a tube changes at a given point, whether from variation in cross-section or from the influx or egress of gas, there will be a corresponding change in pressure. As the velocity increases, the pressure decreases, and vice versa. If the velocity at any time reaches zero, the pressure head reaches its maximum value; that maximum is called total pressure. If the velocity has a definite value, the existing pressure head is something less than the total, and is called static pressure to distinguish it from the total pressure. That part of the total pressure which has been converted into kinetic energy, because of the velocity, is called velocity head or dynamic pressure.

The numerical value of velocity head may be calculated from the expression $\frac{v^2}{2g}$. Inserting this expression into equation (1), we have

$$ph_1 + \frac{{v_1}^2}{2g} = ph_2 + \frac{{v_2}^2}{2g} + Lh \tag{2}$$

where
v_1 = initial velocity;
v_2 = final velocity;
g = acceleration due to gravity.

This formula may be used as a basis when analyzing the distribution of gases in the heating systems of byproduct coke ovens. Because of the large cross-section of the horizontal flues of the ovens and the relatively low velocity of the gases passing through them, it will be permissible to neglect friction losses. Theoretical considerations, as well as attempts to measure the friction losses, have shown that their effect is negligible.

THE KOPPERS COKE OVEN

The design of the Koppers oven is well known; Figs. 1 and 2 show cross and longitudinal sections of it. Fuel gas is supplied through a duct and distributed into individual gas nozzles below each vertical flue. Air for the combustion is drawn in through sole flues and is preheated as it rises through the regenerators. The preheated air enters the vertical flues through air ports and, meeting the fuel gas, burns upward. The products of combustion pass into the horizontal flue through circular ports, the effective openings of which are regulated by sliding bricks. At a given time, combustion takes place in only one-half of the vertical

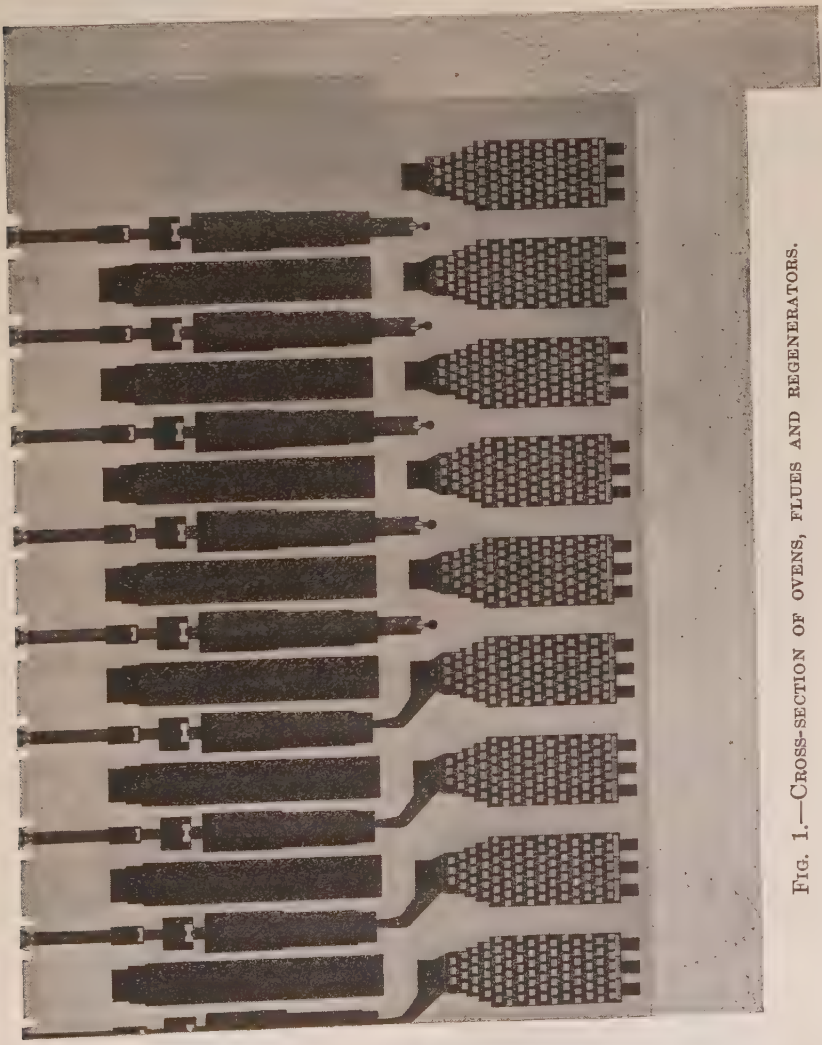
FIG. 1.—CROSS-SECTION OF OVENS, FLUES AND REGENERATORS.

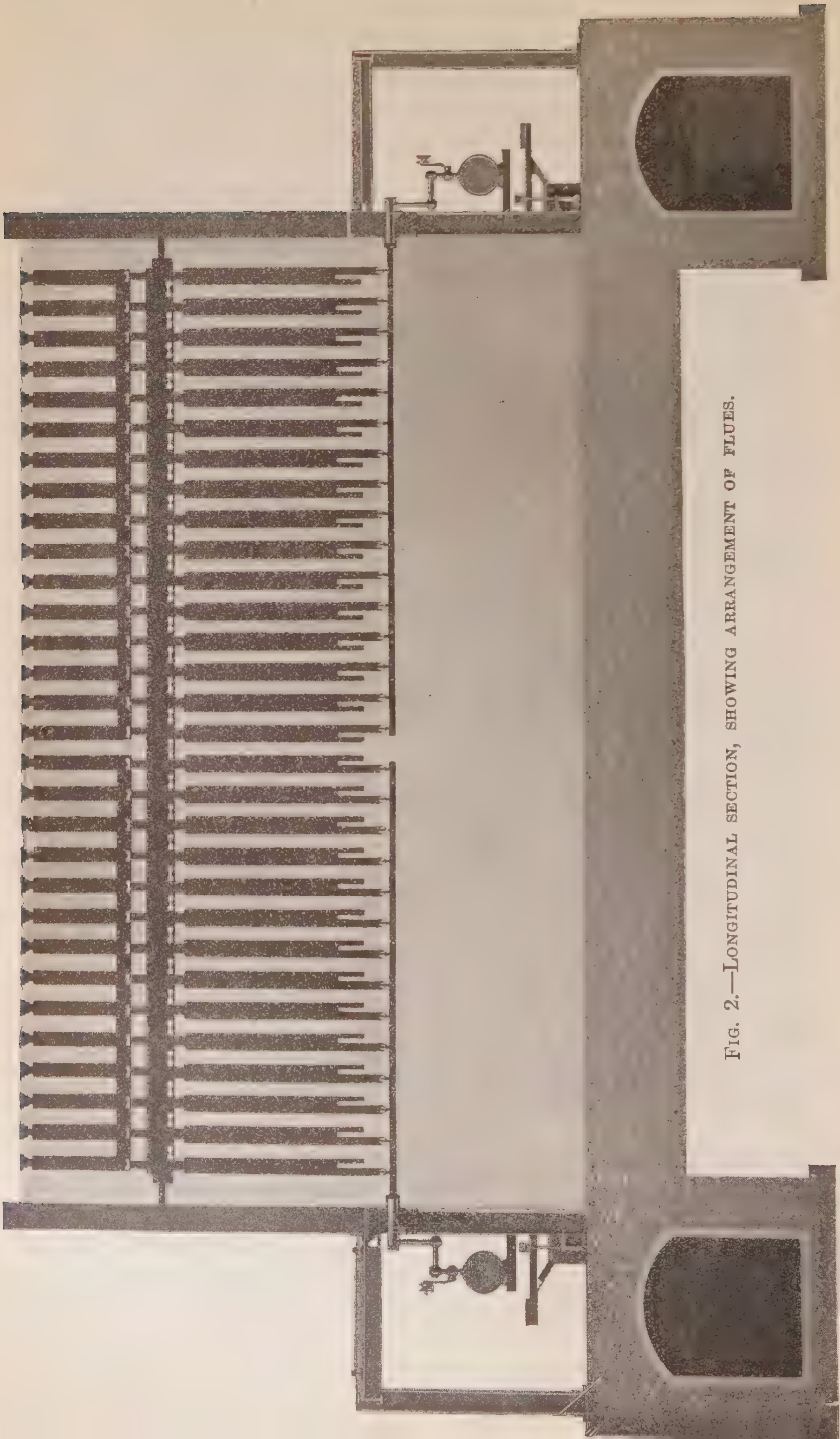

FIG. 2.—LONGITUDINAL SECTION, SHOWING ARRANGEMENT OF FLUES.

flues of a heating wall. The products of combustion collect in the horizontal flue, pass along this flue, and distribute downwards through the vertical flues of the far end of the same heating wall.

Study of Gas Distribution

The static pressure at the top of the regenerators is so nearly uniform that for our purpose it may be considered constant and represented by ph_r, the pressure head at top of regenerators.

The total pressure in the horizontal flue may be measured near the opening of the first vertical flue, where the gas velocity in the horizontal flue is so low that the velocity head is negligible, as could be shown by calculation.

ph_1, the pressure head over first vertical flue = total pressure in horizontal flue.

The velocity of the products of combustion in the horizontal flue increases as they move toward the axis of the oven, until their velocity is at a maximum above the fourteenth vertical flue. The static pressure decreases as the velocity head increases, until at the fourteenth flue,

$$ph_{14} = ph_1 - \frac{v_{14}{}^2 d}{2g} \tag{3}$$

where d = density of gases at prevailing temperature;

v_{14} = velocity of gases in horizontal flue above fourteenth vertical.

The motive pressure that determines the amount of flow in the first vertical flue is $ph_r - ph_1$; and the motive pressure in the fourteenth vertical flue is $ph_r - ph_1 + \frac{v_{14}{}^2 d}{2g}$.

That is, the motive pressure in the fourteenth flue is greater than that in the first flue by $\frac{v_{14}{}^2 d}{2g}$.

If an oven carbonizing 18 tons of coal per 24 hr. burns gas of 550 B.t.u., to meet a requirement of 1200 B.t.u. to coke each pound of coal there will be required 0.9 cu. ft. of fuel gas per oven per second. Assuming that five volumes of air are required for the combustion of one volume of gas and neglecting the slight change in volume due to the formation of products of combustion, there is $6 \times 0.9 = 5.4$ cu. ft. of waste gas per second.

If the area of the horizontal flue is 189.9 sq. in. and the temperature of the products of combustion is 2510° F., the velocity above the fourteenth flue will be 24.7 ft. per sec. Inserting this velocity into the expression $\frac{v_{14}{}^2 d}{2g}$ and using the value 1.3 for the density of the products of combustion when measured under standard conditions, gives 0.617 mm. water for the drop in pressure head in the horizontal flue due to the increase in velocity from the first to the fourteenth vertical flue.

If for the moment it is assumed that there is no loss in pressure head when the gases pass through the vertical flues, it is possible to calculate by the formula of Lecocq[1] the setting of the sliding bricks that will permit the passage of equal amounts of gas in all of the vertical flues.

$$\text{Log}_e \frac{V_n}{V_{n-1}} = \frac{\text{area vertical flue port}}{\text{area horizontal flue cross-section}} \qquad (4)$$

Where V_n = velocity in horizontal flue above vertical flue n;
V_{n-1} = velocity in horizontal flue above next vertical flue $n-1$.

If the area of the vertical flue port is 11 sq. in., $\frac{V_n}{V_{n-1}} = 1.06$; therefore, as the axis of the oven is approached each vertical flue port should have 1/1.06 the area of the preceding one, so that the setting would be approximately as follows:

Flue	Area, Square Decimeters	Width of Opening, Millimeters	Flue	Area, Square Decimeters	Width of Opening, Millimeters
1st	0.712	95	8th	0.474	60
2d	0.672	85	9th	0.447	57
3d	0.634	79	10th	0.421	54
4th	0.598	74	11th	0.398	52
5th	0.564	70	12th	0.375	49
6th	0.532	66	13th	0.354	47
7th	0.503	63	14th	0.334	45

This calculation does not include the friction loss in the horizontal flue, which is negligible, nor the friction loss in the ducts of the vertical flues, which is a measurable amount. This latter friction loss will have the effect of somewhat reducing the variation in setting as calculated above. The settings here calculated closely approximate those used at a plant where extremely satisfactory heating is being obtained. It is not necessary, of course, to have a different setting for each port, as the regulation will be close enough when the setting is altered for each four or five ports. From the Chart, Fig. 3, the sliding-brick setting necessary to give a desired area is readily calculated. When the proper width of opening has been determined, the sliding brick is pushed aside to this extent by thrusting a taper gage, graduated in millimeters, into the opening.

Down Flow of Products of Combustion

By analyzing the down flow of the products of combustion, it is found that as the gases travel along the horizontal flue away from the axis of the oven, the static pressure increases as the velocity decreases, so that increasing quantities of burnt gases pass downward through individual vertical flues as the end of the heating wall is approached. This defect in distribution is increased by the setting of the sliding bricks, which have been placed to give the correct distribution of the upward-flowing burning gases.

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[1] Eugene et Louis Lecocq: "Les Fours à Coke," 308.

This unevenness of down-flow distribution, however, has little practical effect on the heat application: Careful measurements have shown that the following is a typical distribution of heat in a Koppers oven.

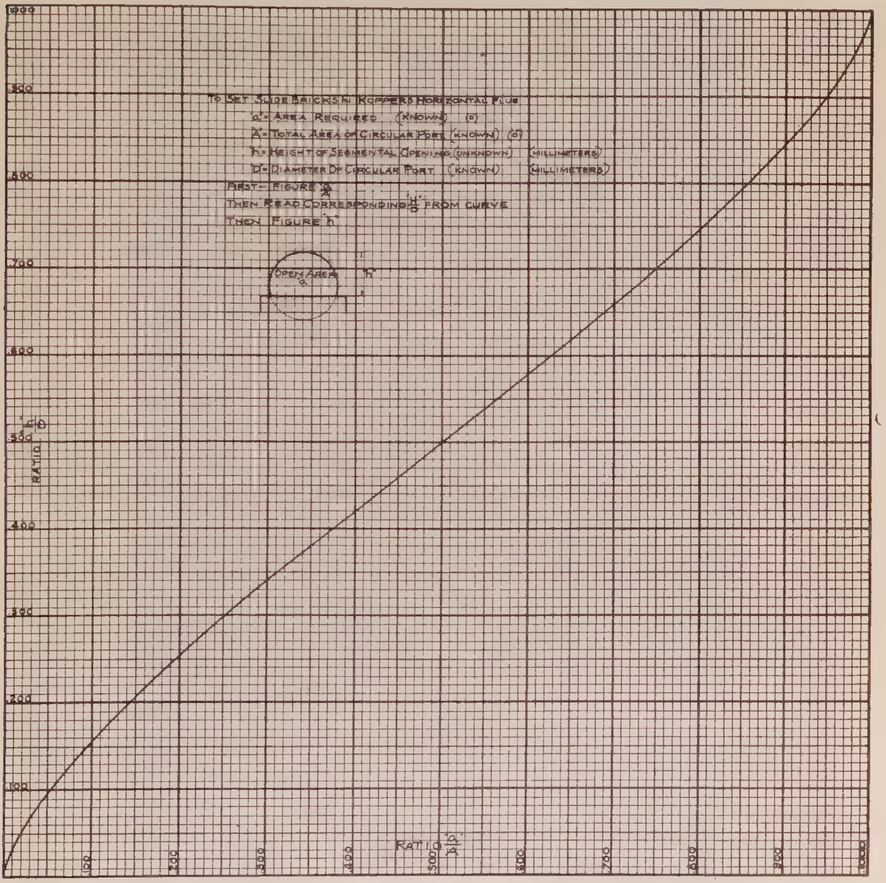

FIG. 3.—CHART FOR CALCULATION OF SLIDING-BRICK SETTING NECESSARY TO GIVE A REQUIRED AREA.

	PER CENT.
Total heat entering vertical flues	100.0
Absorbed upstream	54.7
Absorbed downstream	2.1
Entering top of regenerators	43.2

or

Heat absorbed by oven	100.0
Absorbed upstream	96.4
Absorbed downstream	3.6

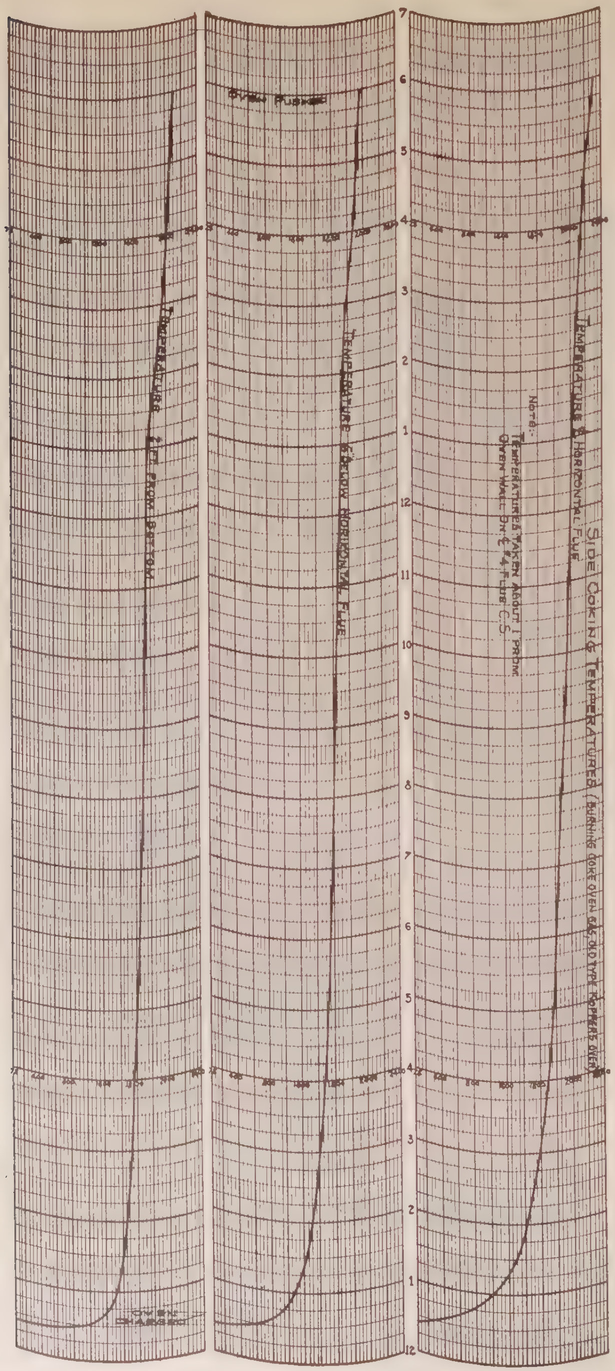

Fig. 4.—Continuous record of temperature 1 in. from heating wall, at three locations.

Inasmuch as nearly all the heat absorbed by the oven is absorbed during the upflow, the non-uniform distribution of gases on down flow is not important.

At a given time, one-half of the oven is receiving only 3.6 per cent. of the heat that is being absorbed, and this condition is maintained for 30 min., when reversal of flow takes place and gas again burns in this half. But the flow of heat into the coal mass is not intermittent; it progresses steadily, and is not affected by the reversals. Perhaps the most striking proof is as follows: The weight of silica brick in the heating wall of a coke oven is approximately 66,000 lb. As the mean specific heat of silica brick for the temperature interval under consideration is about 0.375, the heat capacity of a wall is $66,000 \times 0.375 = 24,750$ B.t.u. for each change in temperature of 1° F. The coal charge of 12 tons will require the application of about 1000 B.t.u. per lb. of coal to carry out the coking process, not considering stack and radiation losses in the heating system, or a total of $12 \times 2000 \times 1000 = 24,000,000$ B.t.u. for the entire charge.

If this charge of coal is placed in an oven having a wall temperature of 2600° F. and no gas is burned in the heating walls at any time during the coking period, the wall temperature will drop $\frac{24,000,000}{24,750} = 969°$ F. That is, the finishing temperature will be about 1630° F., which is a temperature sufficient to produce good metallurgical coke. As the heat capacity of the walls is sufficient to coke completely a charge of coal at the required temperature, it is apparent that the effect of 30-min. reversal period during a coking time of 15 to 20 hr. will be negligible. The most direct proof of this fact is that no continuous record of temperature in the coal charge, taken very near the wall, has shown the slightest evidence of temperature fluctuations due to reversal periods. Fig. 4 shows typical records of temperature in the coal mass 1 in. from the heating wall, at various levels in a Koppers oven operating on a 30-min. reversing schedule.

Regeneration

As about 43 per cent. of the total heat entering the heating wall is present in the down-flowing products of combustion when they enter the top of the regenerators, this heat can be utilized in the most efficient manner by alternately passing the products of combustion in strict counterflow with the requisite amount of air. Measurements at the top of the regenerators in Koppers ovens have shown a pressure difference of not more than 0.02 mm. water underneath the first and fourteenth flues. Any inequality of distribution of down-flowing gases as they come from the vertical flues is, therefore, rectified almost instantaneously. The efficiency of regeneration, however, is not entirely satisfactory for the following reason.

Owing to the longitudinal taper of a coke oven, it is necessary to supply a larger amount of fuel gas to the coke end than to the narrower pusher end. In the Koppers type oven, products of combustion from the coke end of the oven exchange their heat with the air, or air and gas, going to the narrower pusher end. As the latter gases are required in smaller volume, the efficiency of heat exchange varies with reversals, as is shown by fluctuations of the stack temperature. (See Fig. 5.) This inequality of heat exchange may amount to 2 per cent. of the calorific value of the fuel gas.

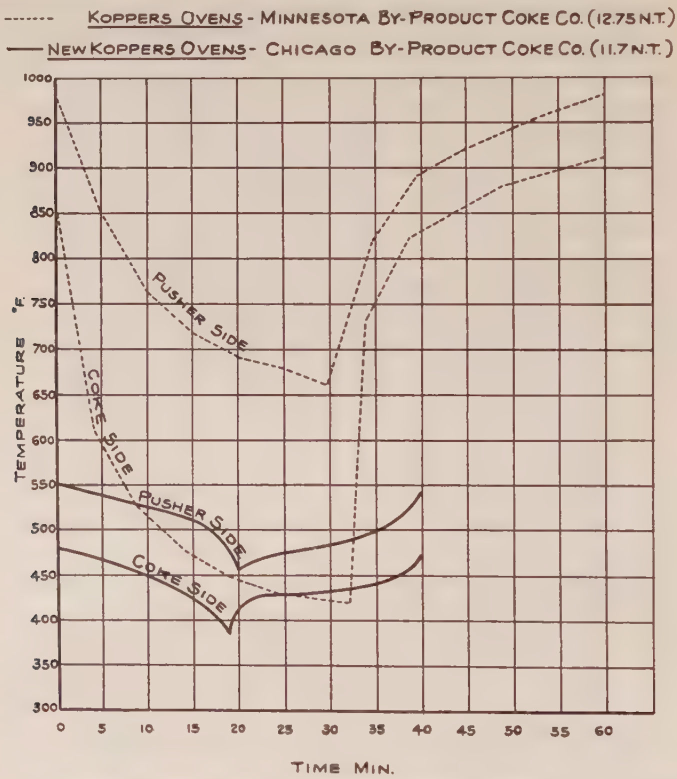

FIG. 5.—TEMPERATURE AT BASE OF REGENERATORS DURING CYCLE OF REVERSALS; NEW TYPE OVEN OPERATING ON 20-MIN. REVERSING SCHEDULE, KOPPERS OVEN ON 30-MIN. SCHEDULE.

NEW TYPE KOPPERS CO. COKE OVEN[2]

The remarkable efficiency of this oven is made possible by the crossover flue principle. As shown by Fig. 6, the vertical flues of a heating wall are divided into two or more groups, each group having its own horizontal flue, and a crossover flue centrally located with respect to the vertical flues it serves. Fig. 6 shows a heating wall divided into five such units. At a given time, gas burns in all the vertical flues of a heating wall. The products of combustion collect in the five horizontal flues and pass over the top of the oven through the respective crossover flues and distribute downwards through the flues in that wall of the oven. It is apparent that reversal of flow of gases in the new type oven takes place over the top of the oven and down the opposite wall, instead of longitudinally in the same heating wall, as in the earlier types.

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[2] Read before a meeting of the Eastern States Blast Furnace and Coke Oven Association in Buffalo, N. Y., Oct. 5, 1922, and reprinted in whole or in part in *Iron Age* (1922) **110,** 1275–1279; *Iron Trade Review* (1922) **71,** 1055–1062; *Gas Age Record* (1922) **50,** 653–656; 671–675; *Blast Furn. and Steel Plant* (1922) **10,** 575–83; *Chem. & Met. Eng.* (1922) **27,** 875–881; *Coal Ind.* (1922) **8,** 489–494.

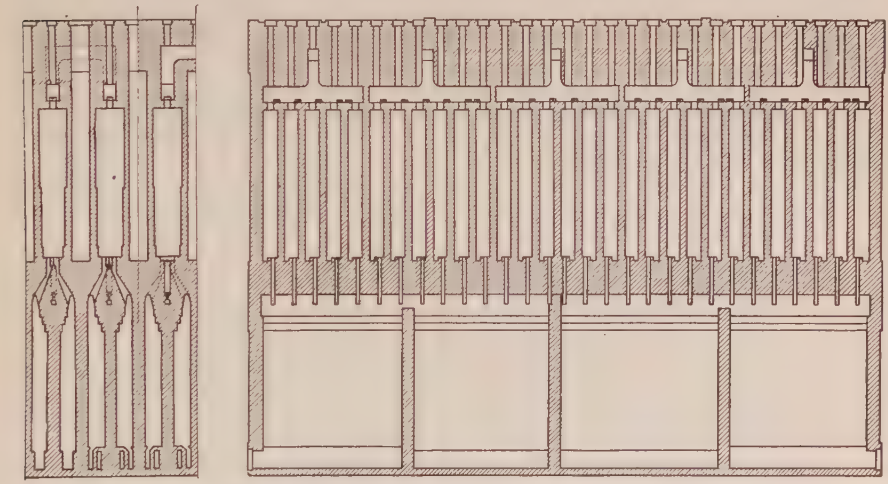

FIG. 6.—CROSS-SECTION AND LONGITUDINAL SECTION; NEW TYPE OVENS.

The new design permits a great reduction in the cross-sectional area of the horizontal flues, which may be located higher in the heating walls, so that the vertical heating flues can extend upwards as far as necessary for the uniform heating of the top of the coal charge. This feature makes possible a considerable shortening of the coking time, as carbonization proceeds at a practically uniform rate at all points in the oven, and it is not necessary to hold a charge in the oven after the bottom is coked, in order to finish the top of the charge. Temperature measurements made at the center line of the new oven show that a temperature difference of not more than 100° F. exists between the top and bottom of the coke mass. Combustibility experiments and shatter tests have shown a distinct improvement in the quality of the coke produced.

The reduced size of the horizontal flues permits the strengthening of the oven wall at this point. In fact, the walls adjacent to the horizontal flues may now be made so thick that they insulate sufficiently to prevent overheating of the gas space above the coal charge. As the size of the horizontal flues is now under almost complete control of the designer, these flues are no longer the main factor governing the practical design and operation of a coke oven.

A battery of five of these ovens having two crossover flues has been in successful operation at the plant of the Chicago By-Product Coke Co. during the past year. Operating on coal gas, producer gas, or producer gas diluted to resemble blast-furnace gas, these ovens have continuously demonstrated that their performance far surpasses that of any other type of coke oven. The new oven will carbonize 27 tons of coal per 24 hr., where the older Koppers oven of the same dimensions operating at the same temperature carbonizes 18 tons. Under similar conditions, a former minimum coking time of 16 hr. has been reduced to 11 hr. In addition, there has been a most gratifying and important improvement in the uniformity of quality of the coke produced. All of these results follow more or less directly from the fact that the crossover flue design has made possible new standards of heat distribution and pressure conditions throughout the entire system.

Heat Distribution and Pressure Conditions

If an oven has a carbonizing capacity of 30 net tons coal per 24 hr., when burning fuel gas of 550 B.t.u., to meet a requirement of 1160 B.t.u. to carbonize each pound of coal, there will be required 1.46 cu. ft. of fuel gas per oven per second. Then, if each volume of gas produces six volumes of burnt gas, there will be 8.76 cu. ft. of products of combustion passing through an entire heating wall per second.

In a new type oven, having in a heating wall five groups of six vertical flues each, each crossover flue will carry one-fifth of the total products of combustion, and each branch of the horizontal flues will carry one-half of that amount, or only one-tenth of the total products of combustion. (This contrasts strongly with the condition in the older Koppers oven, where the horizontal flue must be large enough to carry the entire volume of products of combustion from a heating wall.) Therefore, the volume of burnt gases, passing in the horizontal flues above the third vertical flues of any unit, will be $\frac{8.76}{10} = 0.876$ cu. ft., and the volume passing over the first or end flues will be slightly less than one-third this amount, or less than $\frac{0.876}{3} = 0.292$ cubic feet.

If these gas volumes are corrected to a temperature of 2510° F. and the equation $\frac{v^2d}{2g}$ solved when a horizontal flue of 116.4 sq. in. is used, the drop in static pressure in the horizontal flue above the opening of the third vertical flue will be 0.044 mm. water; and above the first vertical flue less than 0.005 mm. Thus, the maximum difference of motive pressure in the vertical flues of the new oven caused by the velocity of gases in the horizontal flue will be 0.044 − 0.005 = 0.039 mm. water.

This difference in motive pressure is only one-sixteenth as great as the value 0.617 mm., which was the calculated difference in pressure existing in the horizontal flues of the Koppers oven above the first and fourteenth flues. It can be readily understood that proper heat distribution in the new oven is accomplished with remarkable ease. This design of oven operates with as low a suction condition and pressure differential as is possible to attain in practice. This is important. In fact, it will probably be possible to dispense with sliding-brick regulation entirely, thus eliminating a pressure difference of 3 to 4 mm. water, as found in Koppers ovens.

Regeneration

In the new type oven, the products of combustion always exchange their heat with an exactly equivalent amount of air or of air and gas. Thus, the efficiency of regeneration approaches a maximum, and the products of combustion as they leave the regenerator chambers always have a uniform and very low temperature, as is shown by Fig. 5.

Oven Firing with Low B.t.u. Gas

The use of fuel gas of low B.t.u. value, such as producer gas, for underfiring ovens is of increasing importance, for the time is not far distant when economic conditions will force most coke plants to use producer gas or blast-furnace gas for oven firing, so that the entire production of high B.t.u. coal gas will be available for those purposes which require such a gas. Many batteries of Koppers ovens are being regularly under fired with producer gas. However, the large proportion of inerts in this gas increases considerably the volume of products of combustion that must be handled, and oven capacity has been limited by the necessity of designing horizontal flues of sufficient size to handle properly the increased volume of gases. The new type oven has entirely overcome this limiting condition.

Combustion of Blast-furnace Cokes in Fuel Beds

BY RALPH A. SHERMAN* AND JOHN BLIZARD,† PITTSBURGH, PA.

(New York Meeting, February, 1923)

THE experimental investigation described in this paper was conducted to determine the relative combustibilities of different samples of blast-furnace coke when burned in a current of air in a small furnace. It was undertaken in connection with the work done by Perrott[1] and Kinney[2] on the process of combustion in the blast-furnace hearth.[3]

Reference is often made to the "combustibility" of coke by blast-furnace men, but exactly what is meant by the term "combustibility" is somewhat vague. Brassert,[4] in a paper before a meeting of blast-furnace men in 1914, refers to combustibility as follows:

> The value of the coke in relation to the heat reactions in the furnace is determined not only by the carbon content but even more so by its combustibility. This, in turn, is governed by its size, the physical and chemical composition of the carbonaceous substance forming the cell walls, and the interior and surface structure of the coke pieces. These properties are, in turn, influenced by the chemical and physical properties of the coals used for coking, the density of the cake of coal in the oven, the coking time, temperature and heat regulation of the ovens, and last, but not least, by the method of quenching, screening and handling the finished coke.
>
> What principally concerns the blast furnace is the rate of progression of the combustion, which depends not so much on the chemical analysis as on the physical qualities of the coke. It is this rate of progression that we term combustibility, which is the speed at which the carbon molecules in the coke combine with oxygen under given conditions. In general, what the blast furnace requires is a coke which will burn rapidly and with intensity at the tuyeres, yet which has a sufficiently strong structure and hard surface to resist abrasion and solution by CO_2 in the upper part of the stack.

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* Assistant Physicist, Bureau of Mines.

† Fuel Engineer, Bureau of Mines.

[1] Associate Physical Chemist, Bureau of Mines.

[2] Assistant Metallurgical Chemist, Bureau of Mines.

[3] Combustion of Coke in Blast-furnace Hearth; see p. 543.

[4] Herman A. Brassert: "Notes on Blast-furnace Coke with Particular Reference to Combustibility." 1914. See Johnson: "Principles, Operation and Products of the Blast Furnace," 171–178.

This definition of combustibility as "the speed at which carbon molecules combine with oxygen under given conditions" renders it necessary, when comparing the combustibilities of various kinds of coke, to determine the rate of chemical reaction between the carbon and oxygen when the size of the pieces of coke, the rate of flow of the air over the coke, and other factors are the same in all experiments. Thus, if two kinds of coke were burned in two furnaces in fuel beds consisting of pieces of coke of the same size, in blasts of air of exactly the same intensity, the rates of gasification would be a measure of what might be termed the mean combustibility of the whole mass of the fuel bed. If the two kinds of coke be considered as consisting of pure carbon and it be remembered that the oxygen is practically all combined with carbon to form CO_2 or CO in passing through the fuel bed, it is easily seen, as the same quantity of oxygen passes through each fuel bed in the same time, that the combustibility of the coke is greater, the greater the quantity of CO present in the gas leaving the top of the fuel bed. That is to say, if all the carbon in the gas from one coke were in the form of CO and in the other all in the form of CO_2, the mean combustibility of the former coke would be exactly twice that of the latter.

However, if the gases leaving the two fuel beds consisted of CO only, it need not follow that the cokes were of the same combustibility, for one might have reduced the CO_2 to CO in one-half the length of gas travel through the fuel bed, whereas the other might have completely reduced the CO_2 to CO only near the top of the fuel bed. In other words, what has been called the mean combustibility would be the same for both cokes, though the real combustibility would differ. In combustibility experiments in which air is forced through coke, it is therefore necessary to take samples of gas from various parts of the fuel bed along the travel of the gas, and then by determining the composition of these samples to show the mean combustibility of the fuel bed between the points of air admission and sampling.

These combustibilities may be estimated from the initial rate of decrease of the oxygen of the air and the CO and CO_2 contents of the gas samples, or if a more exact measure be desired it may be calculated as a percentage of the maximum attainable combustibility. The method of calculating the percentage combustibility in this paper neglects the hydrogen and hydrocarbon contents of the gas and is calculated as follows:

Consider a gas containing x molecules of CO_2, y molecules of CO, and z molecules of O_2; then if the x molecules of CO_2 be reduced to $2x$ molecules of CO and the z molecules of O_2 form $2z$ molecules of CO by contact with carbon, we shall have $2x + y + 2z$ molecules of CO and so have obtained from the original gas one containing the largest possible amount of carbon. The expression $\frac{x + y}{2x + y + 2z}$ thus represents the ratio of the carbon actually present in a sample of gas to the maximum that could have been obtained, and

$$\frac{CO_2 + CO}{2CO_2 + CO + 2O_2} \times 100$$

may be termed the mean combustibility, in per cent., for the bed of coke between the grate and the point of sampling.

In this paper are presented the results of twenty-five tests on four kinds of coke when burned in fuel beds of uniform depth with a uniform rate of air supply during each test.

ACKNOWLEDGMENT

Acknowledgment is made of the valuable coöperation of A. C. Fieldner, supervising chemist, G. St. J. Perrott, associate physical chemist, and S. P. Kinney, assistant metallurgical chemist, of the Bureau of Mines; also to the companies that furnished the coke used in the tests. The coke analyses were made in the coal laboratory of the Bureau of Mines under the direction of H. M. Cooper, chemist; and the physical tests in the analytical laboratory under the direction of W. A. Selvig, assistant analytical chemist.

COKES USED IN THE TESTS

Four cokes were selected for the tests and ample supplies were furnished by the manufacturers.

1. Benham coke, Wisconsin Steel Co., Wilputte oven, Kentucky coal, coking time 29 hours.
2. St. Louis coke, St. Louis Coke and Chemical Co., Roberts oven, about 90 per cent. Illinois coal and 10 per cent. Pocahontas coal, coking time 15 hours.
3. Clairton overheated coke, Clairton By-Product Coke Co., Koppers oven, Connellsville coal; average temperature of coking 1850° F., time 19 hours, 56 minutes.
4. Clairton underheated coke, average temperature of coking 1600° F., time 16 hours, 17 minutes.

The cokes as received were crushed and screened, that which went through a 1½-in. mesh and over a 1-in. mesh being used in the tests. The average analyses together with data on physical tests are given in Table 1.

METHOD OF TESTING THE COKES

In the first series of seventeen tests, eleven of which are reported here, the cokes were burned at rates approximating 25, 50, 75, and 100 lb. per sq. ft. per hr., firing at frequent intervals and maintaining a constant thickness of fuel bed and a constant rate of air supply during each test.

TABLE 1.—*Analyses and Physical Properties of Coke*

	Coke Analysis														
Coke	Moisture, Per Cent.	Volatile Matter, Per Cent.	Fixed Carbon, Per Cent.	Ash, Per Cent.	Hydrogen, Per Cent.	Carbon, Per Cent.	Nitrogen, Per Cent.	Oxygen, Per Cent.	Sulfur, Per Cent.	Calorific Value, B.t.u. Per Lb.	Specific Gravity		Porosity, Per Cent. by Volume of Cell Space	Weight Per Cu. Ft., Lb.	Softening Temperature of Ash, ° F.
											Apparent	True			
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Benham	0.4	1.0	89.5	9.1	0.5	89.0	0.6	0.2	0.6	13,185	0.95	1.90	50.0	25.9	2,130
Clairton overheated	0.1	0.8	87.0	12.1	0.4	85.2	0.7	0.9	0.7	12,520	1.09	1.95	44.1	30.0	2,180
Clairton underheated	0.5	1.7	85.4	12.4	0.7	83.9	1.4	0.6	1.0	12,535	1.03	1.97	47.7	28.6	2,305
St. Louis	1.3	1.6	81.9	15.2	0.7	81.2	0.8	1.1	1.0	12,060	0.87	1.84	52.7	22.6	2,040

The gases arising from the fuel bed were sampled continuously throughout the test. These tests were made to study the effect of the rate of combustion and the difference in the cokes on the final products of combustion.

In the second series of eight tests, approximately 25 lb. of each coke were burned per square foot per hour, each coke being run in duplicate. Three sets of gas samples and temperature readings were taken in the fuel bed during each test. These tests were made to study the effect of the differences in the cokes on the process of combustion in the fuel bed.

DESCRIPTION OF APPARATUS

The tests were conducted in an experimental hand-fired furnace of 1 sq. ft. grate area which is described in detail in *Technical Paper* 137.[5] A cross-section of the furnace is shown in Fig. 1. It consists essentially of a steel shell, lined with firebrick, set on an ashbox fitted with an air-tight door, for the removal of ashes, and connected to a fan by a square air duct in which is placed an orifice for the measurement of the air supply. A series of ⅝-in. holes through the wall at 1½-in. intervals, in a vertical

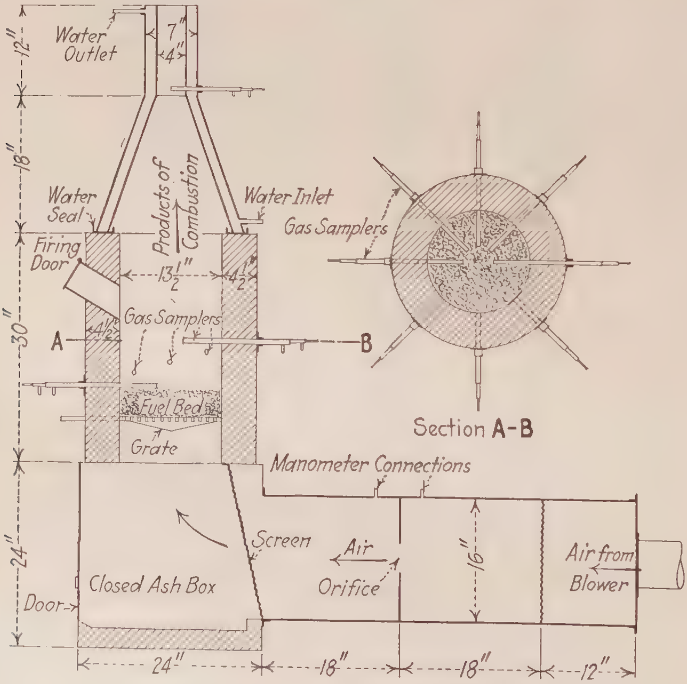

FIG. 1.—CROSS-SECTION OF EXPERIMENTAL FURNACE.

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[5] Henry Kreisinger, F. K. Ovitz and C. E. Augustine: Combustion in the Fuel Bed of Hand-Fired Furnaces. Bureau of Mines *Tech. Paper* 137 (1917) 15–16.

direction, are distributed radially around the furnace for the sampling of gases and measuring the temperatures in the fuel bed.

On the top of the furnace, a conical sheet-iron hood, which was cooled by water flowing through it, was placed in a water seal. This hood maintained a pressure in the furnace, thus preventing the infiltration of air, partly cooled the gases from the fuel bed, and, by reducing the cross-sectional area of the gas stream at the point of sampling, enabled a representative sample of gas to be collected.

A small firing door, placed about 14 in. above the grate, was connected to a gate between the fan and ash pit so that the air supply was cut off when the door was opened for firing.

The gas samples were drawn from the fuel bed or from the gases leaving the furnace through water-cooled gas samplers and collected, over mercury, in glass receptacles. The composition of the gas was determined, by analysis, in a modified Orsat, using mercury as the displacing fluid. The gases were analyzed for their CO_2, O_2, CO, H_2, and CH_4 content.

The temperature of the gases leaving the furnace was determined by means of a platinum, platinum-rhodium thermocouple placed in the gas stream. The temperatures in the fuel bed were determined by means of a disappearing-filament type of optical pyrometer sighted through the holes used for sampling the gases.

METHOD OF RUNNING THE TESTS

All tests were run with a 12-in. fuel bed. On starting the test, the fuel bed was built up to this height at approximately the rate of combustion to be used during the test and this rate maintained until the furnace was heated through. The fire was then cleaned by poking and by shaking the grates and the ashes removed from the ashpit.

The coke was fired in small quantities at the intervals required to maintain the level constant, the quantity fired varying from 10 lb. every 25 min. to 10 lb. every 6 minutes.

The air blast intensity for each test was maintained at a constant rate by keeping constant the drop of pressure across an orifice through which it flowed on its way to the furnace. The air supply was calculated from values determined by Durley.[6]

The pressure drop across the orifice, through the fuel bed, the temperature of stack gases, the air temperature, and the temperature and amount of water entering and leaving the water-cooled hood were observed and recorded at regular intervals during a test.

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[6] R. J. Durley: On the Measurement of Air Flowing into the Atmosphere through Circular Orifices in Thin Plates and under Small Differences of Pressure. *Trans.* Am. Soc. Mech. Engrs. (1906) **27**, 193.

TABLE 2.—*Averages of Results of Tests*

Test Number	Duration, Hours and Minutes	Total Coke Fired, Pounds	Mean Coke Fired per Hour, Pounds	Ash Removed, Pounds	Combustible in Coke, Per Cent.	Combustible in Ash, Per Cent.	Combustible Burned per Hour, Pounds	Diameter of Orifice, Inches	Pressure Drop, Inches of Water		Air Supplied, Pounds Per Hour	Temperature, Degrees F.		Humidity, Grains per Cubic Foot	Barometric Pressure, Inches of Mercury	Position of Gas Sampler, Inches Above Grate	Average Gas Composition by Volume						Fuel Bed Temperature, Degrees F.
									Through Orifice	Through Fuel Bed		Air	Stack Gases				CO_2, Per Cent.	O_2, Per Cent.	CO, Per Cent.	H_2, Per Cent.	CH_4, Per Cent.	N_2, Per Cent.	
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24
Benham coke																							
1	2 30	50.0	20.0	1.75	92.8	49.9	18.2	2.0	0.52	0.16	172	79	1063			Stack	7.2	0.5	23.0	1.8	0.3	67.2	2534[a] 2248[b]
5	2 05	92.0	44.0	10.4	90.6	37.4	38.1	3.0	0.59	0.89	408	80	1450	4.48	29.39	Stack	5.3	0.3	25.6	1.4	0.1	67.3	2830[a] 2450[b]
6	1 24	75.0	53.6	9.1	90.6	47.8	45.5	3.0	1.06	1.17	555	87	1700	6.63		Stack	5.5	0.5	25.0	1.5	0.4	67.1	2853[a] 2434[b]
7	1 28	100.0	68.0	15.3	92.4	27.7	60.0	3.5	0.77	1.77	640	87	1466			Stack	4.9	0.3	26.6	2.0	0.4	65.8	2713[a] 2275[b]
Clairton Overheated																							
8	2 11	50.0	22.9	5.6	86.7	10.9	19.6	2.0	0.52	0.19	172	88	1170			Stack	5.9	0.4	25.5	1.8	0.2	66.2	2656[a] 2220[b]
10	1 40	75.0	45.0	15.5	86.9	16.8	37.5	3.5	0.28	0.25	340	84	1580	5.69	29.29	Stack	6.6	0.4	24.2	1.4	0.2	67.2	2787[a] 2400[b]
11	1 34	97.5	62.0	27.0	86.2	46.4	45.3	3.5	0.55	1.10	533	81	1865	6.74	29.27	Stack	7.6	0.3	22.6	1.1	0.2	68.2	2726[a] 2445[b]
15	1 43	108.5	63.0	14.4	86.5	33.0	51.8	3.5	0.60	1.97	557	91	1763			Stack	7.5	0.7	22.2	2.0	0.2	67.4	2747[a] 2532[b]
Clairton Underheated																							
13	1 33	40.0	25.8	5.5	85.3	29.0	21.0	2.0	0.54	0.21	175	81	1438	3.42	29.32	Stack	6.4	0.3	22.8	1.6	0.2	68.7	2753[a] 2236
16	1 30	70.0	46.6	6.6	85.5	19.0	39.0	3.5	0.31	1.23	370	83	1558	4.52	29.28	Stack	7.2	0.7	20.9	2.0	0.0	69.2	2817[a] 2459[b]
17	1 36	90.0	56.3	7.0	87.1	19.3	48.2	3.5	0.47	1.44	485	85	1490	8.77	29.12	Stack	8.3	0.6	20.2	2.1	0.3	68.5	2698[a] 2510[b]

St. Louis Coke

18	1	40	60.0	35.9	7.3	83.9	6.2	29.9	2.0	0.47	0.44	163	74	1347	3.74	29.42	1.5	1.2	18.9	0.0	0.0	0.0	79.9	2116
																	3.0	8.0	12.3	0.0	0.0	0.0	79.7	2598
																	4.5	13.0	6.8	1.1	0.0	0.0	79.1	2683
																	6.0	13.0	0.5	11.2	0.0	0.1	75.2	2758
																	7.5	8.6	0.8	18.7	0.3	0.2	71.4	2507
																	9.0	9.3	0.4	18.9	0.4	0.2	70.8	2200
																	10.5	7.3	0.5	20.8	0.6	0.1	70.7	2320
																	12.0	7.4	0.6	21.6	1.2	0.0	69.2	
																	Stack	7.2	1.1	21.3	0.9	0.1	69.4	
19	1	56	50.0	25.9	7.6	83.7	28.5	20.6	2.0	0.48	0.26	165	76	1001	3.69	29.20	1.5	5.5	15.2	0.0	0.0	0.0	79.3	2175
																	3.0	11.4	9.4	1.5	0.1	0.1	77.5	2509
																	4.5	16.7	3.3	2.1	0.0	0.1	77.8	2579
																	6.0	13.7	1.0	9.8	0.1	0.2	75.2	2562
																	7.5	6.5	0.4	22.4	0.4	0.2	70.1	2227
																	9.0	8.8	0.4	19.2	0.5	0.1	71.0	2125
																	10.5	7.2	0.3	21.7	0.2	0.5	70.1	2033
																	12.0	6.0	0.3	23.9	0.9	0.2	68.7	
																	Stack	7.3	0.2	21.9	1.1	0.1	69.4	

Clairton Underheated

20	2	2	50.0	24.6	6.3	87.7	33.2	20.6	2.0	0.48	0.33	165	76	984	5.75	29.26	1.5	6.5	14.0	0.2	0.2	0.0	79.1	2506
																	3.0	14.7	5.1	1.2	0.2	0.0	78.8	2676
																	4.5	13.9	0.6	9.6	0.8	0.1	75.0	2618
																	6.0	12.8	0.5	12.4	1.1	0.0	73.2	2472
																	7.5	6.3	0.8	23.1	0.8	0.0	61.5	2241
																	9.0	7.1	0.2	21.5	0.8	0.0	70.4	2256
																	10.5	6.6	0.5	23.0	0.8	0.0	69.1	2242
																	12.0	6.1	0.4	23.2	0.8	0.2	69.3	
																	Stack	8.3	0.2	20.1	1.8	0.0	69.6	
21	1	59	50.0	25.3	6.0	87.7	15.3	21.7	2.0	0.52	0.28	173	78	1041	4.25	29.40	1.5	12.5	6.8	1.0	0.1	0.0	79.6	2549
																	3.0	6.9	11.8	0.8	0.2	0.0	80.3	2712
																	4.5	11.3	2.6	10.8	0.4	0.1	74.8	2736
																	6.0	12.5	0.7	12.7	0.5	0.1	73.5	2662
																	7.5	6.8	0.2	22.4	0.5	0.2	69.9	2644
																	9.0	5.7	0.3	23.9	0.7	0.2	69.2	2538
																	10.5	5.5	0.2	25.9	0.7	0.0	67.7	2361
																	12.0	6.3	0.2	24.0	0.9	0.2	68.4	
																	Stack	8.0	0.0	20.7	1.5	0.1	69.7	

[a] 4.5 in. above grate.
[b] 10.5 in. above grate.

TABLE 2.—*Averages of Results of Tests* (*Continued*)

Test Number	Duration, Hours and Minutes	Total Coke Fired, Pounds	Mean Coke Fired per Hour, Pounds	Ash Removed, Pounds	Combustible in Coke, Per Cent.	Combustible in Ash, Per Cent.	Combustible Burned per Hour, Pounds	Diameter of Orifice, Inches	Pressure Drop, Inches of Water		Air Supplied, Pounds per Hour	Temperature, Degree F.		Humidity, Grains per Cubic Foot	Barometric Pressure, Inches of Mercury	Position of Gas Sampler, Inches Above Grate	Average Gas Composition by Volume						Fuel Bed Temperature, Degrees F.
									Through Orifice	Through Fuel Bed		Air	Stack Gases				CO_2, Per Cent.	O_2, Per Cent.	CO, Per Cent.	H_2, Per Cent.	CH_4, Per Cent.	N_2, Per Cent.	
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24
Clairton Overheated																							
22	2 05	50.0	24.0	5.9	88.7	25.7	20.6	2.0	0.49	0.25	166	79	982	5.53	29.30	1.5	6.3	13.9	0.8	0.1	0.0	78.9	2562
																3.0	16.9	0.7	5.0	0.3	0.0	77.1	2735
																4.5	13.0	0.3	11.8	0.2	0.1	74.6	2660
																6.0	15.2	0.4	8.4	0.1	0.1	75.8	2529
																7.5	7.6	0.9	19.8	0.5	0.0	71.2	2454
																9.0	10.8	0.6	14 9	0.6	0.0	73.1	2355
																10.5	7.2	0.6	21.9	0.4	0.1	69.8	2183
																12.0	5.8	0.2	23.9	1.4	0.1	68.6	
																Stack	8.4	0.5	20.4	0.8	0.0	69.9	
23	2 06	50.0	23.8	5.5	88.7	16.4	20.7	2.0	0.51	0.19	170	76	1016	5.53	29.46	1.5	2.4	18.6	0.0	0.0	0.0	79.0	2737
																3.0	13.0	4.9	4.9	0.1	0.0	77.1	2825
																4.5	13.9	3.1	6.2	0.2	0.0	76.6	2782
																6.0	13.6	0.6	11.7	0.3	0.1	73.7	2677
																7.5	7.0	0.7	21.6	0.7	0.1	69.9	2588
																9.0	8.2	1.3	18.9	0.6	0.0	71.0	2486
																10.5	5.1	1.5	23.8	0.7	0.1	68.8	2323
																12.0	6.2	0.9	23.3	0.8	0.1	68.7	
																Stack	7.7	0.6	20.6	1.0	0.1	70.0	

Benham Coke

24	1	58	50.0	25.4	8.3	89.2	71.9	19.6	2.0	0.49		166	77	932	2.70		1.5	12.5	8.1	0.0	0.0	0.0	79.4	2500
																	3.0	14.6	0.7	8.4	0.2	0.1	76.0	2580
																	4.5	15.8	0.8	6.2	0.3	0.1	76.8	2527
																	6.0	13.9	0.5	10.6	0.2	0.0	74.8	2527
																	7.5	9.0	0.9	17.4	0.9	0.0	71.8	2353
																	9.0	7.1	1.0	20.2	0.8	0.1	70.8	2251
																	10.5	9.2	0.9	17.3	0.9	0.0	71.7	2188
																	12.0	8.0	0.9	20.1	1.5	0.0	69.5	
																	Stack	8.7	0.8	18.7	1.4	0.0	70.4	
25	1	58	50.0	25.4	7.6	89.3	67.5	20.1	2.0	0.49	0.26	166	73	852	3.71	29.25	1.5	13.7	5.7	1.8	0.5	0.0	78.3	2586
																	3.0	14.8	5.2	0.9	0.1	0.0	79.0	2600
																	4.5	14.1	1.1	9.8	0.3	0.0	74.7	2544
																	6.0	15.9	1.4	6.2	0.3	0.0	76.2	2502
																	7.5	9.7	1.0	16.6	0.6	0.0	72.1	2350
																	9.0	8.1	0.9	19.3	0.5	0.1	71.1	2265
																	10.5	6.0	0.2	24.4	0.6	0.1	68.7	2134
																	12.0	7.0	0.5	21.8	1.1	0.0	69.6	
																	Stack	8.4	0.6	18.4	1.3	0.2	71.1	

When the required number of gas samples and temperature readings had been taken the test was closed with the same depth of fuel bed as at the start, the grate was shaken, the ashes removed and the fire dumped. The clinkers were removed from the dumped coals, added to the ash, and charged to the test.

RESULTS OF TESTS

The averages of the data collected during the tests are given in Table 2.

DISCUSSION OF RESULTS

Tests at Varying Rates of Combustion

In Fig. 2 are shown the variations in the composition of the stack gases and fuel-bed temperatures for the Benham and two Clairton cokes at varying rates of combustion. For the Benham coke, the CO content of the gases increases with the rate of combustion, while with the two Clairton cokes, there is a decrease. The maximum difference between the CO content of the gases for two cokes at the same rate or for one coke at different rates is about 6 per cent. The fuel-bed temperatures do not vary greatly, although all increase somewhat at first with the rate of combustion.

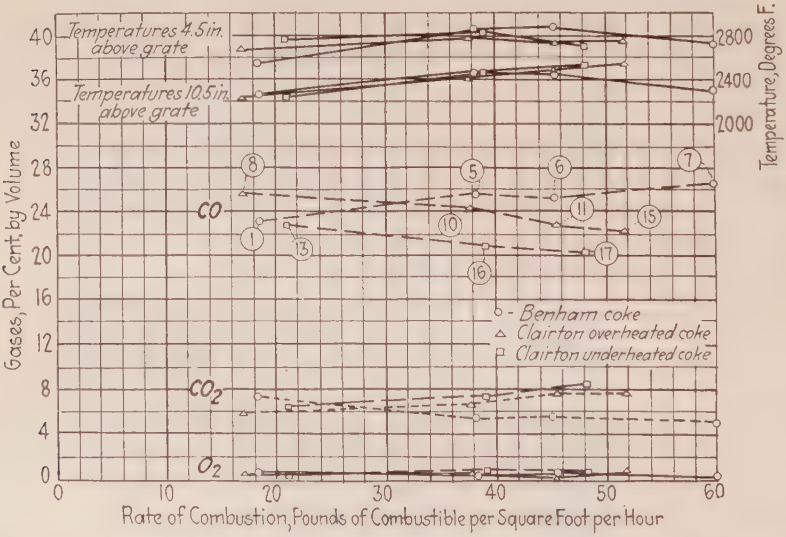

FIG. 2.—COMPOSITION OF STACK GASES AND TEMPERATURES FROM VARIOUS COKES AT VARYING RATES OF COMBUSTION. TEST NOS. 1, 5, 6, 7, 8, 10, 11, 13, 15, 16, 17.

The curves in Fig. 2 show that neither the rate of combustion nor difference in coke materially affects the final products of combustion from the fuel bed. This effect of intensity of air blast on the gas composition is in accord with the conclusion reached in *Technical Paper* 137.[7] "Within wide limits, the rate of feeding air has practically no effect on the composition of the gases within the fuel bed. This is particularly true of the gases near the surface of the fuel bed." This has been proved within the limits of 3 to 180 lb. of fuel per square foot per hour.

Combustion in Fuel Bed

In the study of the process of combustion in a fuel bed by the sampling of gases and reading temperatures, many sets of samples and readings are necessary to secure values that may be taken as representing the average conditions at a point. In any horizontal plane in a fuel bed, the gases will show varying composition at various points, owing to the varying resistance to the flow of the gases. If, however, sufficient samples are taken during a test, the average of the analyses will closely represent the average composition of the gases in that plane. Another point to be noted when studying the composition of gas samples is that the gas samples were not taken instantaneously but during a period of 5 to 10 min. When a gas sample contains a particular relative quantity of CO

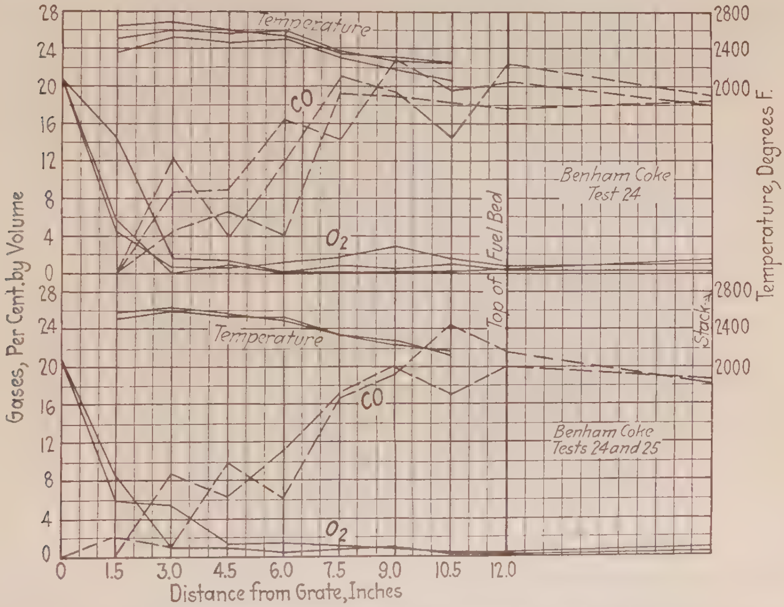

FIG. 3.—O_2 AND CO CONTENT OF GASES AND TEMPERATURES IN FUEL BED SHOWING VARIATION AT DIFFERENT PERIODS DURING A TEST AND FOR DIFFERENT TESTS.

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[7] *Loc. cit.*

and O_2, this does not necessarily mean that they were coexistent at all times in the fuel bed. At one moment the gas stream may have contained a large percentage of O_2 and at another moment a large percentage of CO.

In the upper half of Fig. 3, the O_2 and CO content of individual gas samples and fuel-bed temperatures at different periods of test 24 on Benham coke are plotted on a base showing the distance of the sampling point from the grate; the CO_2 is omitted to avoid confusion in reading the curves. There are, as would be expected from the foregoing, considerable variations in the composition of the gases. The general tendency of the curves is, however, the same. In the lower half of the figure are plotted the averages of the O_2 and CO content and temperatures from tests 24 and

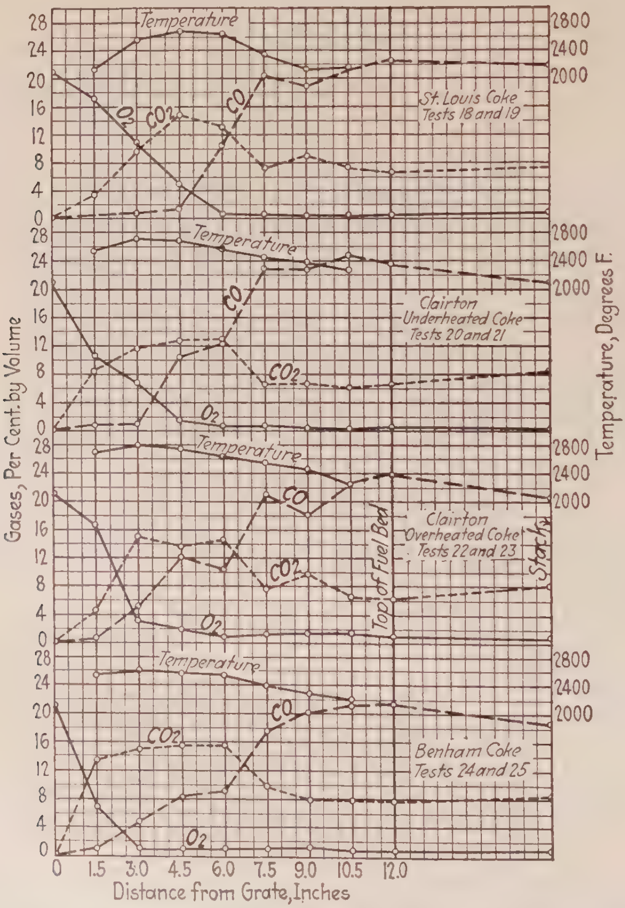

FIG. 4.—AVERAGE COMPOSITION OF GASES AND TEMPERATURES IN FUEL BEDS OF ST. LOUIS, CLAIRTON, AND BENHAM COKES.

25, which were duplicate tests on the same coke. These curves do not differ greatly, but show the necessity of running several tests before a true value of the combustibility of the coke may be obtained.

The averages of the gas composition and temperature from all samples and readings taken on the duplicate tests of the four cokes when burned at approximately 25 lb. per sq. ft. per hr. are shown in Fig. 4. The curves are much alike for the Benham and the two Clairton cokes. The CO_2 remains at a maximum value for several inches in the fuel bed in all tests with these three cokes. The initial rate of decrease of O_2 and increase of CO_2 are greater for the Benham than for the Clairton cokes. The maximum CO_2 is somewhat less, and therefore the corresponding CO greater, for the Clairton cokes than for the Benham coke and the maximum temperature attained with the Clairton cokes is higher than that with the Benham coke; also the percentage of CO and temperature at 10½ in. are higher for the Clairton cokes than the others.

The St. Louis coke gives somewhat different curves. The initial rate of decrease of O_2 and increase of CO_2 are much less than for the other cokes and the temperature at 1½ in. is lower by 400°. The peak of the CO_2 curve with St. Louis coke is quite sharp compared with that for the Benham and Clairton cokes.

Between the top of the fuel bed and the stack, the CO decreases and the CO_2 increases for all the cokes. This probably results from the combustion of the CO with some O_2 that has reached the top of the fuel bed by coming up between the fuel bed and the furnace wall because of the difference in the resistance between this point and the fuel bed, or it may result from the reaction

$$CO + H_2O \rightarrow CO_2 + H_2$$

In the lower half of Fig. 5, the mean O_2, CO, and temperature curves are assembled for more convenient comparison, and the mean combustibility of the cokes from the grate to about 2 in. above may be given from inspection of the O_2 curves as being in the following order: (1) Benham, (2) Clairton underheated, (3) Clairton overheated, (4) St. Louis. On the other hand, the mean combustibility of the cokes in the fuel bed as a whole, measured by the CO contents at the top of the fuel bed, would be in the following order: (1) Clairton underheated, (2) St. Louis, (3) Clairton overheated, (4) Benham. Thus the relative mean combustibilities of the various cokes vary in order, depending on whether the measurement is taken over the whole fuel bed or for the first few inches above the grate. The most striking example of the difference is for Benham coke, which reduces the O_2 content of the blast near the grate much more rapidly than the other cokes but which, higher in the fuel bed, reduces the CO_2 less rapidly than the other cokes.

In the upper half of this figure are given curves drawn from data in

Technical Papers 137 and 207 on lignite char in a 6-in. fuel bed, Pittsburgh coal, anthracite, and a metallurgical coke in 12-in. fuel beds.[8, 9] These fuels in order of their combustibility are: (1) Lignite char, (2) Pittsburgh coal, (3) anthracite, (4) coke.

The variations between the widely different Pittsburgh and anthracite coals is no more marked than that between the cokes used in the present study and the particular coke used follows the St. Louis coke quite closely. The lignite char, however, is markedly different, the O_2 having completely disappeared at but 1½ in. above the grate, and the CO reaching a maximum of 32 per cent. in a 6-in. fuel bed. The great activity of lignite char has again been shown in some recent tests at the U. S. Bureau of Mines on the use of lignite char in a base-burner house-heating stove, where the gases immediately above a 2-in. fuel bed contained 27 per cent. of CO. From what we know of charcoal, it is probable that its combustibility would be comparable to that of lignite char. It should be noted that the lignite char was smaller than the other

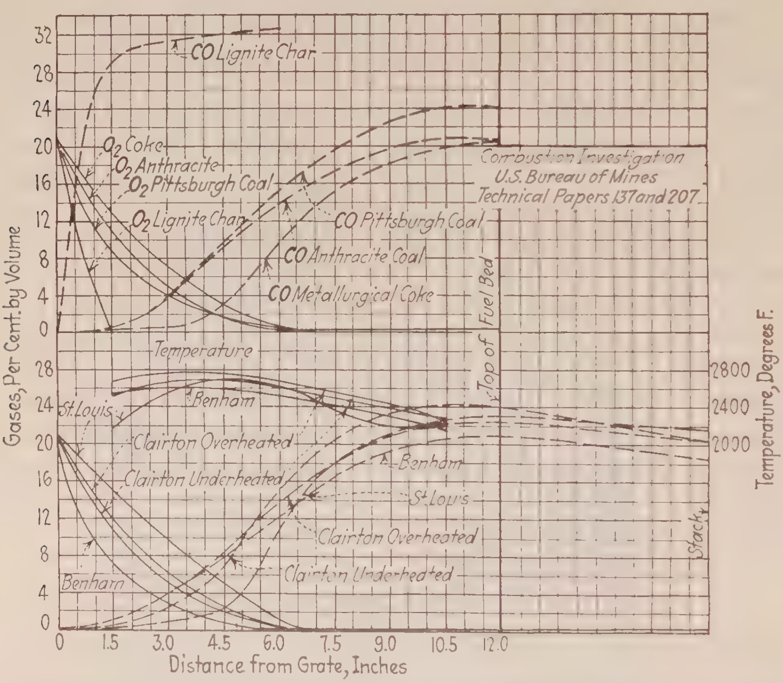

FIG. 5.—MEAN O_2 AND CO CONTENT OF GASES AND TEMPERATURES IN FUEL BEDS OF DIFFERENT FUELS.

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[8] *Loc. cit.*

[9] Henry Kreisinger, C. E. Augustine, and W. C. Harpster: Combustion Experiments with North Dakota Lignite. Bureau of Mines *Tech. Paper* 207 (1919) 19.

fuels, 100 per cent. passing a ½-in. screen. In order to determine what the effect of a similar size of coke would be, tests with this size of coke are planned and will be run.

Fig. 6 shows values of the expression $\frac{CO_2 + CO}{2CO_2 + CO + 2O_2} \times 100$,

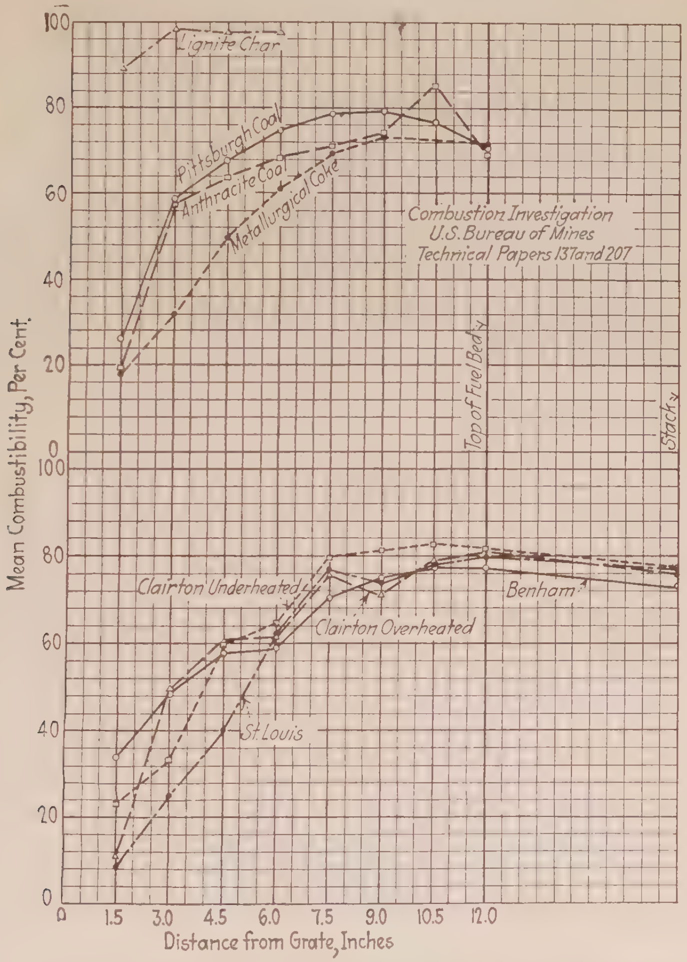

Fig. 6.—Mean combustibility of various fuels at varying distances from the grate.

calculated from the composition of the gases at the various points of sampling for the various cokes, lignite char, Pittsburgh coal, and anthracite. This expression gives an approximate measure of the mean combustibility, expressed as a percentage of the maximum attainable. The curves show that near the grate the combustibility varies with the different cokes, but that about 6 in. above the grate the mean combustibility is about the same. Between this point and the top of the fuel bed, the combustibility curves again separate somewhat but approach one another at the top of the fuel bed.

General Summary of Results

This investigation of the process of combustion in fuel beds of blast-furnace cokes shows:

1. That the rate of combustion, within wide limits, has no great effect on the final products of combustion.
2. That the relative mean combustibilities of the various cokes are different in different parts of the fuel bed.
3. That the mean combustibilities of the cokes for the entire fuel bed, calculated from the stack gases, when burned at a rate of approximately 25 lb. per sq. ft. per hr. were:

Clairton underheated	77 per cent.	St. Louis	77 per cent.
Clairton overheated	76 per cent.	Benham	73 per cent.

(*Discussion of this paper begins on page* 584)

Combustion of Coke in Blast-furnace Hearth*

By G. St. J. Perrott† and S. P. Kinney,‡ Tuscaloosa, Ala.

(New York Meeting, February, 1923)

Consumers of metallurgical coke are agreed that the quality of their fuel plays an important part in the performance of the furnace. Less unanimous agreement is evident when the properties of a desirable coke are discussed or when consideration is made of suitable testing methods by which the worth of a coke for metallurgical purposes may be determined quantitatively.

Much has been written of the importance of combustibility as a factor in determining the performance of coke in the blast furnace and it is generally believed that cokes vary widely in this respect. The usual operating data of the furnace afford little evidence as to the combustibility of the fuel.

As a part of a general investigation of the properties of metallurgical coke, the Bureau of Mines has explored the hearth combustion zone in a number of blast furnaces by means of water-cooled gas sampling tubes driven into the hearth through the tuyeres. The attempt has been made to determine the difference in combustibility of cokes varying widely in physical properties. Data have also been obtained in an experimental combustion furnace at the Bureau laboratories in Pittsburgh.

This paper gives the results of experiments, made at eleven blast furnaces, in which the composition of the gases was determined at various points in the hearth in the plane of the tuyeres. Further work is in progress in which the zone above the tuyeres will be explored.

Factors Affecting Coke Combustibility

Brassert[1] has defined combustibility as the rate of progression of combustion, or, in other words, the speed at which the carbon molecules combine with oxygen under given conditions. He states that combustibility is governed by the size of the coke pieces, the physical and chemical composition of the carbonaceous material forming the cell walls, and the interior and surface structure of the coke pieces. A coke for blast-furnace use must burn rapidly at the tuyeres and yet not be too vulnerable to the stack gases. A coke that is porous and light but with hard cell walls and surfaces will give best results; it will offer more carbon surface per unit weight to the oxygen of the blast than a denser, heavier coke and hence will be more combustible.

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* Published by permission of the Director, Bureau of Mines.

† Associate Physical Chemist, Southern Experiment Station, Bureau of Mines.

‡ Assistant Metallurgical Chemist, Southern Experiment Station, Bureau of Mines.

[1] J. E. Johnson, Jr.: "Principles, Operation, and Products of the Blast Furnace," 172. N. Y., 1918. McGraw-Hill Book Co.

Koppers[2] states that highly combustible coke may be made by carbonizing at temperatures between 550° and 800° C. He believes that the ideal metallurgical coke should burn rapidly to carbon monoxide at the tuyeres and states that oxidation of metallic iron occurs in the combustion zone at the tuyeres, necessitating the consumption of carbon in the hearth for the reduction of the iron oxide so formed and tending to cool the hearth.[3] A combustion zone of small extent is therefore desirable. Oxidation (or solution) of carbon in the stack is believed, by Koppers, to be desirable because the heat absorbed by this reaction tends to keep the coke cool and in its original highly combustible state until it reaches the tuyeres. He believes that the combustibility of coke is influenced by the amount of volatile matter it contains and, to some extent, by the porosity. Heating above 700° C. reduces the oxygen and hydrogen content and produces a less combustible coke of higher true specific gravity.

Sutcliffe and Evans[4] agree with Koppers as to the necessity for combustible coke in blast-furnace operation but do not consider his method of carbonization below 800° C., to be a satisfactory solution. They lay special stress on the importance of porosity and believe that a fuel containing interconnected pores is more combustible than one in which the pores are not connected.

Other workers have defined combustibility in terms of empirical laboratory tests, as for example, the relative weights of fuel oxidized in a given time when equal weights of various cokes are placed in an experimental furnace and burned by natural draft or by a measured forced draft. It is generally believed that a light porous coke will burn "faster" than a dense coke of low porosity, but opinions differ as to what constitutes faster burning. The statement is often made that the blast furnace "travels faster" with a free burning coke: often this means merely that a greater number of skip loads of a porous coke are used daily than when dense coke is employed, a phenomenon quite independent of combustibility and explained simply on the basis of difference in apparent specific gravity. The implication is often made, however, that the weight of carbon burned per minute, under conditions where the volume of blast per minute is constant, increases with the combustibility of the coke.

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[2] H. Koppers: Fortschritte auf dem Gebiete der Kokserzeugung, der Einflusz der Koksbeschaffenheit auf den Hochofenbetrieb und Vorschläge fur die Verbesserung des letzteren. *Stahl und Eisen* (1921) **41,** 1173–81, 1254–62; Vorschläge zur Prüfung des Kokses fur Hochofen- und Gieszereizwecke. *Stahl und Eisen* (1922) **42,** 569–73.

[3] Obviously the net heat effect is identical whether iron is burned by air and carbon burned by the iron oxide so formed, or carbon burned directly by air.

[4] E. R. Sutcliffe and Edgar C. Evans: Influence of Structure on Combustibility and Other Properties of Solid Fuels. *Jnl.* Soc. Chem. Ind. (1922) **41,** 196–208T.

Various investigators have shown that the primary product[5] of combustion in a fuel bed is carbon dioxide, which then reacts with the incandescent carbon to form carbon monoxide. The aim of burning fuel at the tuyeres in the blast furnace, besides producing the necessary heat for the reactions, is to form "bosh gas" for reducing the ore in the stack. This bosh gas consists of carbon monoxide, nitrogen, and small amounts of hydrogen. Discussion must be limited to the combustion of that part of the fuel which actually reaches the tuyeres and is consumed by the blast, neglecting the variable amount oxidized by direct reduction or by premature combustion (or solution) in the stack gases.

It is essential that a clear definition of the term combustibility be made before the subject can be discussed intelligently. For our purposes the term must be considered from the standpoint of the blast furnace, a furnace in which air for combustion is supplied at a constant and very high velocity, in which the temperature of the combustion zone is 1650–1800° C. and in which, irrespective of the fuel employed, the final gases of combustion contain carbon monoxide and only traces of carbon dioxide. Thus a given weight of blast will burn the same weight of carbon at the tuyeres, irrespective of the physical properties of the coke. Analogies drawn from combustion at low rates in hand-fired furnace or from combustion tests in experimental furnaces burning by natural draft are apt to be misleading.

In terms of blast-furnace operation, we may therefore formulate the following measure of combustibility: The combustibility of fuel burned at the tuyeres is inversely proportional to the distance from the tuyeres at which oxygen and carbon dioxide have disappeared or, in other words, directly proportional to the space rate of disappearance of oxygen and carbon dioxide. As combustion takes place at the tuyeres in a zone of three dimensions, the term may be defined (assuming a tuyere of given dimensions and a given weight of air per minute) as the mean rate of gasification of fuel per unit volume of the combustion zone.

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[5] It is not definitely known whether CO_2 is the initial product of combustion or whether CO is first formed and immediately oxidized by excess oxygen in the gases. Rhead and Wheeler [*Jnl.* Chem. Soc. (1912) **101,** 846–56 and (1913) **103,** 461–89] conclude that both CO and CO_2 are primarily formed. For practical purposes, however, CO_2 may be taken as the primary product of combustion.

The extent to which combustion has proceeded at any point in a fuel bed may be conveniently expressed as the ratio between the percentage of carbon in the gases at that point and the maximum percentage the gases can hold, *i.e.*, when they consist of CO and N_2 only. From the equations

$$O_2 + C = CO_2$$
$$CO_2 + C = 2CO$$

it appears that when combustion has proceeded completely to the formation of CO, the volume of CO appearing in the gases is equal to $2O_2$. Hence the expression

$$\frac{CO_2 + CO}{2O_2}$$

where CO_2, CO, and O_2 are in molecules, represents the percentage saturation of the gases with carbon. From the gas analyses, this percentage may be determined as follows (where $20.9 \times \frac{N_2}{79.1}$ = oxygen in unit volume of combustion gases):

$$\frac{CO_2 + CO}{2\left(20.9 \times \frac{N_2}{79.1}\right)} \times 100 \qquad (1)$$

or

$$189 \times \frac{CO_2 + CO}{N_2}$$

where CO_2, CO, and N_2 are in volume per cent.

If the blast is passing through a fuel bed of pure carbon, and hence all the oxygen appears in the gases, the expression $20.9 \times \frac{N_2}{79.1}$ is obviously equal to $(O_2 + CO_2 + \frac{1}{2}\ CO)$ and equation (1) may be written

$$\frac{CO_2 + CO}{2(O_2 + CO_2) + CO} \times 100 \qquad (2)$$

In the blast-furnace hearth, the gases may contain a greater percentage of CO than could be obtained by burning carbon with air, due to the introduction of CO produced by direct reduction. Equation (1) will indicate this condition by values higher than 100 per cent. if the gases contain little CO_2, and is to be preferred to equation (2) for our purposes. The hydrogen, oxygen, and nitrogen present in the fuel and the blast are assumed to be dry. Results are to be considered merely as approximate indications of the extent of combustion at any point.

Before considering the experimental data on combustion in the blast-furnace hearth, obtained by the writers, certain conclusions may be made as to the probable effect of the physical properties of coke on its combustibility:

1. Combustibility of coke does not increase with decreasing apparent density or with increasing porosity. The extent of the zone of combustion, other factors being constant, depends on the total effective surface exposed to the blast per unit volume of coke. Neglecting the effect of porosity, the total surface per unit volume is independent of the apparent density of the coke. The total effective surface is probably independent of the number and size of the pores, over the range of porosity in ordinary metallurgical coke, because the voids between the separate coke pieces are so large, in comparison to the size of the pores, and the velocity of the blast of such magnitude that the percentage of the blast passing in and out of the pore depressions or passing through the coke pieces by way of connected pore spaces is infinitesimal.

P. H. Royster[6] has suggested that a porous coke might be expected to be less combustible than a dense coke because combustion at the blast velocity obtaining in the furnace probably takes place on the exposed peaks of the cell walls, making a porous coke present less effective surface for combustion than a dense coke. In any event, it is obvious that the surface exposed per unit weight of coke is no measure of its combustibility because the exposed surface in a combustion zone of given dimensions is independent of apparent gravity *per se*, provided, of course, that the size of the coke pieces is constant. Differences in apparent density may have a certain mechanical effect, because for a given weight of carbon burned per minute, the number of pieces of coke falling into the combustion zone will increase as the apparent density decreases. Thus, with a coke of low apparent density and the fuel in the combustion zone in a greater state of flux, the effective carbon surface exposed in unit volume of the combustion zone may be lower than when dense coke is employed. This would mean decreased combustibility, as measured by the extent of the combustion zone at the tuyeres.

2. Combustibility probably increases within certain limits as the size of the coke pieces diminishes, because the smaller coke presents greater surface per unit volume to the impact of oxygen molecules of the blast. This is evidenced by the fact that diminishing the size of coke increases the pressure necessary to force a given amount of air through a fuel bed of given dimensions, the increased pressure being due to increased number of impacts of air molecules with the fuel. Practical operating considerations limit the range of sizes that may be employed successfully.

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[6] Private communication.

3. The rapidity of combustion may depend on the nature of the carbonaceous material composing the cell walls of the coke. Clements, Adams, and Haskins[7] have shown that the rate of reduction of carbon dioxide by charcoal is considerably greater than the rate of reduction by coke over the temperature range 900°–1100° C. As the extent of the combustion zone is affected by the rate of oxidation of coke by carbon dioxide formed as the primary product of combustion, it is possible that the character of the carbonaceous material may, as Koppers believes, influence the rate of combustion. It must be remembered, however, that the temperature of the combustion zone at the tuyeres (1650–1800° C.) is considerably higher than the range investigated by Clements, Adams, and Haskins. Differences in the rate of reduction of CO_2 by different forms of carbonaceous material may be considerably less at these high temperatures.

To sum up: The combustibility, or rate of gasification, of fuel per unit volume of the combustion zone does not increase with decreasing apparent density (considered apart from porosity), it probably does not increase with increasing porosity, but probably increases to some extent as the size of the coke pieces diminishes, and may depend on the nature of the carbonaceous material.

HISTORICAL

Beginning with Bunsen, in 1843, a number of investigators[8] have endeavored to explain the chemical reactions taking place in the blast furnace by analyzing the gases at different levels. Peter Tunner, in 1860, concluded that combustion at the tuyeres extends in the direction of the air current to a distance of 1½ ft., the hottest part of the focus, about 6 in. in extent, being in the middle. The space of combustion extends upwards about 1 ft. Increased pressure, increased blast velocity, or change of tuyere dimensions only slightly modifies the extension of the space of combustion.

In 1893, Van Vloten[9] published the results of a series of experiments in which gas samples were taken by means of a water-cooled sampling tube, both through the tuyere and through a hole 25 in. above the tuyere. He found that oxygen had disappeared 24 in. from the tuyere, that CO_2 had disappeared 40 in. from the tuyere, and that none of the samples taken 25 in. above the tuyere contained any oxygen. His results showed considerable deficiency of oxygen in the vicinity of the tuyeres, which he concluded was caused by the burning of other elements than carbon, *i.e.*, silicon, manganese, iron, and phosphorus. He divides the hearth into an oxidizing zone of small extent around the tuyeres where oxygen and carbon monoxide are present, and a reducing zone where only carbon monoxide is present.

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[7] J. K. Clements, L. H. Adams, and C. N. Haskins: Essential Factors in the Formation of Producer Gas. Bureau of Mines, *Bull.* 7 (1911).

[8] The reader will find the data of Bunsen and others completely set forth in Wedding's *Handbuch der Eisenhüttenkunde* (1906) **3**, 203–241.

[9] W. Van Vloten: Die Verbrennung in Gestell des Hochofens. *Stahl und Eisen* (1893) **13**, 26–30.

Table 1.—*Data Obtained Near Tuyeres by Other Investigators*

Furnace	Distance above Tuyeres, Feet	Analysis of Gases					Fuel Used	Investigator	Date
		CO_2	CO	H_2	CH_4	N_2			
Clerval, France	0.0	0.00	51.35	1.25		47.40	Charcoal	Ebelmen	1841
	1.5	0.31	41.59	1.42		56.68	Charcoal	Ebelmen	1841
	0.0	0.93	39.86	0.79	0.25	58.17	Charcoal	Ebelmen	1848
	3.5	0.00	37.55	1.13	0.10	61.22	Charcoal	Ebelmen	1848
Eisenerz, Austria	0.3	11.60	22.06			66.34	Charcoal	Tunner and Richter	1860
Hammarby, Sweden	0.2	0.51	37.6	1.7		60.2	Charcoal	Rinnman and Fernqvist	1862
Forssjo, Sweden	0.2	1.4	36.7	1.4		60.3	Charcoal	Rinnman and Fernqvist	1862
Hasselfors, Sweden	0.2	1.4	75.9	1.9		20.8	Charcoal	Rinnman and Fernqvist	1864
	3.1	4.1	31.4	1.6		62.9	Charcoal	Rinnman and Fernqvist	1864
Eisenerz, Styria	0.0	1.19	29.33	1.62	0.03	67.70	Charcoal	Kupelwieser and Schoffel	1873
	0.0	2.96	38.22	1.17	0.09	57.56	Charcoal	Kupelwieser and Schoffel	1873
Audicourt, France	0.0	0.0	48.52	0.90		50.58	Charcoal and bit. coal	Ebelmen	1841
Vienne, France	2.1	0.68	38.84	1.41		61.07	Coke	Ebelmen	1843
Pont-l'Eveque, France	0.8	8.11	16.53	0.26		75.10	Coke	Ebelmen	1843
	1.0	5.87	22.25	0.68		71.20	Coke	Ebelmen	1843
	2.2	0.16	36.15	0.99		62.70	Coke	Ebelmen	1843
Seraing, Belgium	2.4	0.00	45.05	0.25	0.07	54.63	Coke	Ebelmen	1848
Alfreton, England	2.1	0.00	37.43	3.18		58.05	Bit. coal	Bunsen and Playfair	1845
Clarence, England	tuyere[a]	1.7	35.0	0.3		63.0	Coke	Lowthian Bell	1884
Germany	3.0	0.3	34.0	0.9	0.2	64.5	Coke	Levin and Niedt	1911
	2.1	0.3	35.9	0.9	0.1	61.6	Coke	Levin and Niedt	1911
	1.2	0.3	42.6	0.9	0.2	54.9	Coke	Levin and Niedt	1911
	5.5	0.30	41.72	2.44	0.60	54.98	Coke	Metz	1913

[a] Obtained by shutting off blast from one tuyere and collecting gases blowing out of the closed tuyere.

In 1911, Levin and Niedt[10] and, in 1913, Norbert Metz[11] conducted an extensive series of experiments in which gas samples were taken for several months at different levels in the furnace. At 1.2 ft. above the tuyeres, Levin and Niedt found no oxygen, only a trace of carbon dioxide, and about 43 per cent. of carbon monoxide. Metz had no sampling hole nearer the tuyeres than 5.5 ft. The gases here contained no oxygen, traces of carbon dioxide, and about 42 per cent. carbon monoxide.

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[10] M. Levin and H. Niedt: Untersuchungen uber die Zusammensetzung des Gasstromes im Hochofen. *Metallurgie* (1911) **8**, 515–39.

[11] Norbert Metz: Studien über die Vorgange im Hochofen. *Stahl und Eisen* (1913) **33**, 93–104.

TABLE 2.—*Data of Van Vloten* (1893)

Distance from Nose of Tuyere, Inches	Gas Analysis				
	CO_2	O_2	CO	H_2	N_2
			Samples taken in tuyere plane		
14	8.25	9.75		4	78
14	6	13		0.75	80.25
24	15	1.5	1.5	1.75	80.25
24			32.5	1.75	65.75
24	7.75		19	0.75	72.50
24	3.25		25.5	1	70.75
24	11.75		9.25	0.5	78.5
32	2.5		28.25	2.25	67
32	2		28.5	4	65.5
36	2.25		30.25	3	64.5
41			39.75	3	57.25
48[a]			37	3	60
48	0.5		38	2	59.5
Midway between two tuyeres			33.75	1.75	64.5
			Plane 25 inches above tuyere		
4			34.25	0.75	65.0
11	3.0		27.25	2.0	67.75
19[b]	4.0		24.5	2.5	69.0
23.5	6.0		19.5	0.75	73.75
27	5.5		23.0	2.5	69.0
31	11.0		11.75	2.0	75.25
35	3.75		27.0	2.75	66.5
42.5	1.0		30.0	1.0	68.0
51[c]	0.5		33.0	2.0	64.5

[a] Center of hearth.
[b] Vertically above nose of tuyere.
[c] 17 in. from center of hearth.

Tables 1 and 2 summarize the analyses obtained near the tuyeres by these investigators.

The conclusion to be drawn from the work of previous investigators is that combustion at the tuyeres is rapid and that the area of the combustion zone where oxygen and carbon dioxide are present is of small extent compared to the total area of the hearth

PRESENT INVESTIGATION

In the present investigation, gas samples have been secured at various points in the hearth in the plane of the tuyeres. Data have been secured at six blast furnaces in the Birmingham district and at five northern furnaces.

In the procedure finally adopted, a water-cooled sampling tube, Fig. 1, is introduced into the hearth through a hole in the tuyere cap. The outer tube is 2-in. boiler tubing and the central gas-sampling tube is of ⅜-in. black pipe. By hammering with a dolly, the pipe is driven to the center of the furnace. Compressed air is blown through the ⅜-in. tube while the pipe is being driven into the furnace, to prevent stoppage of the tube with slag or coke. When the pipe has been driven in a sufficient distance, the air line is disconnected and the sampling begun.

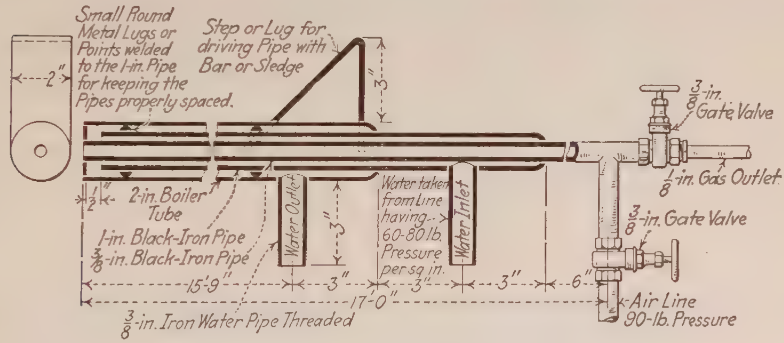
Small Round Metal Lugs or Points welded to the 1-in. Pipe for keeping the Pipes properly spaced.
Step or Lug for driving Pipe with Bar or Sledge
2"
3"
3/8-in. Gate Valve
1/8-in. Gas Outlet
3/8-in. Gate Valve
1/2"
2-in. Boiler Tube
1-in. Black-Iron Pipe
3/8-in. Black-Iron Pipe
Water Outlet
3"
Water taken from Line having 60-80 lb. Pressure per sq. in.
Water Inlet
15'9"
3"
3"
3"
6"
17'0"
Air Line 90-lb. Pressure
3/8-in. Iron Water Pipe Threaded

FIG. 1.—WATER-COOLED GAS SAMPLING TUBE.

As the gas comes out under considerable pressure, sampling is relatively simple. Samples are collected by displacement over mercury in 4-oz. wide-mouth bottles, which are closed with a rubber stopper after the sample is taken. A small amount of mercury is left in the bottles and they are stored stopper down until analyzed, the mercury covering the stopper and preventing solution of the gases in the rubber. Samples are collected at points 6 in. apart, from the center of the hearth to a distance of 4 ft. from the tuyere, when they are collected every 4 inches.

Analyses were made in a modified Orsat apparatus of the Burrell type. Two pipettes of the bubbling type containing acid cuprous-chloride solution were used for the determination of carbon monoxide. Hydrogen was determined in a slow-combustion pipette. Cuprous-chloride solutions were removed as soon as any trace of carbon dioxide appeared after combustion. Check analyses were made, from time to time, by the Bureau laboratory in Pittsburgh. In these latter analyses, carbon monoxide and hydrogen were determined by combustion over copper oxide, and methane was determined by slow combustion. No methane was found either in the hearth-gas or the stack-gas samples. While small amounts of methane might be expected in the stack gas of a furnace operating on high-volatile coke, it is probable that the methane reported in many published analyses is really carbon monoxide that escaped absorption in the cuprous-chloride solutions. It is essential that two cuprous-chloride pipettes be employed and solutions be renewed frequently.

The investigation was begun in the Birmingham district and data at many of the Southern furnaces are somewhat meager because of the mechanical difficulties at first experienced in obtaining samples. A considerable number of tubes were melted before the final design and procedure were developed. One of the most troublesome difficulties experienced was the stoppage of the gas line with slag. At times, several hours were taken in getting a few samples because just as soon as sampling was started the gas line became plugged with slag, necessitating removal of the pipe from the furnace and removing the obstruction from the pipe by a drill. This trouble was experienced when the nose of the pipe was in the region from 1 to 3 ft. from the tuyeres, rarely in the inner part of the hearth or close to the tuyeres.

The pipe can be easily pushed in 24 to 32 in. from the nose of the tuyere, the distance varying with the furnace, at which point it hits a fairly solid obstruction and must be driven farther with a dolly.

RESULTS OF INVESTIGATIONS

Table 3 gives operating data of the eleven furnaces. With the exception of the stack-gas analyses, these were taken from the records of the companies; stack-gas analyses were made by the writers. The figures in the table represent the average of a considerable number of samples taken at various times during the experiments.

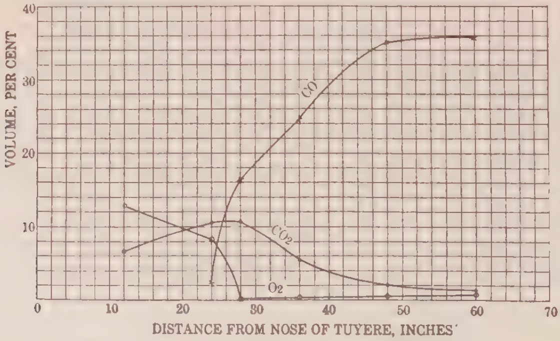

FIG. 2.—ALABAMA FURNACE NO. 1.

Table 4 gives chemical analysis and physical properties of the cokes used as fuel by the different furnaces. These determinations were made at the Bureau laboratories on samples of coke secured by the writers during the hearth-gas sampling.

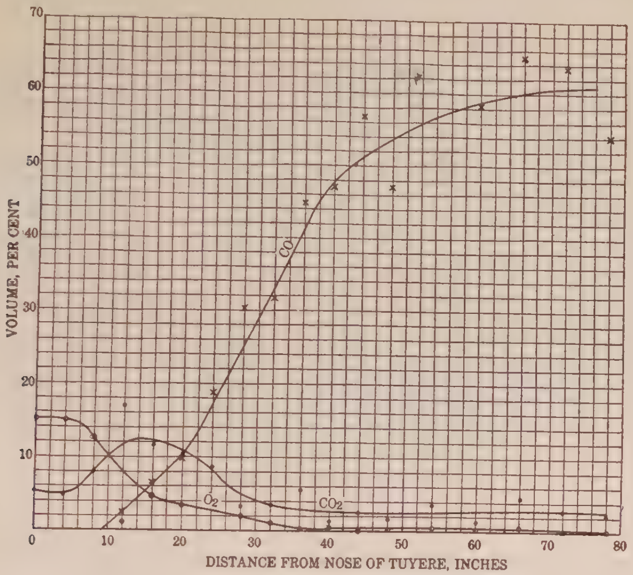

FIG. 3.—ALABAMA FURNACE No. 2.

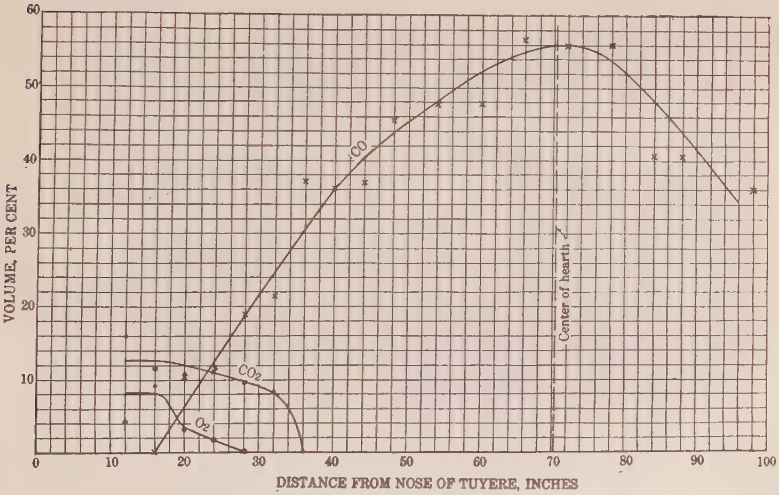

FIG. 4.—ALABAMA FURNACE No. 3.

In Table 5, the hearth-gas analyses obtained at the eleven furnaces are given in summarized form; Tables 6–16 give the detailed data. In Figs. 2 to 12, the average values for the several furnaces are plotted;

Fig. 13 is a plot of the average values for ten furnaces. Pennsylvania furnace No. 2 has not been included in this average because its curve was of different character from the other furnaces. In Fig. 14, the combustibility factor, equation (1), at different points in the hearth has been plotted for several furnaces.

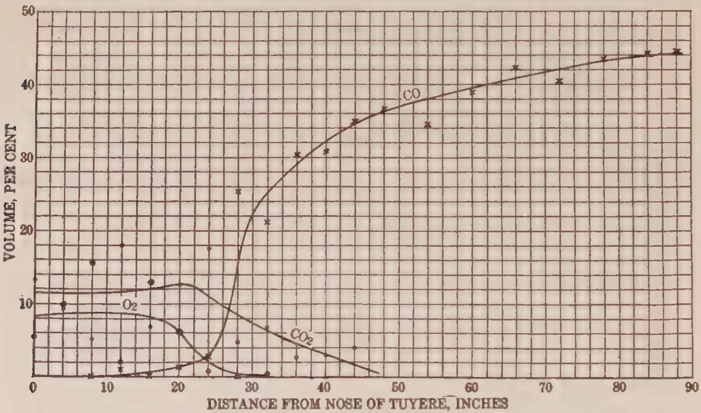

FIG. 5.—ALABAMA FURNACE NO. 4.

The tables are self-explanatory with the exception of the column headed "Parts oxygen per 1000 parts nitrogen." Dry air contains 264 parts oxygen per 1000 parts nitrogen by volume. When dry air combines with carbon to form mixtures of O_2, CO_2, CO, and N_2, the parts oxygen per 1000 parts nitrogen may be calculated as follows:

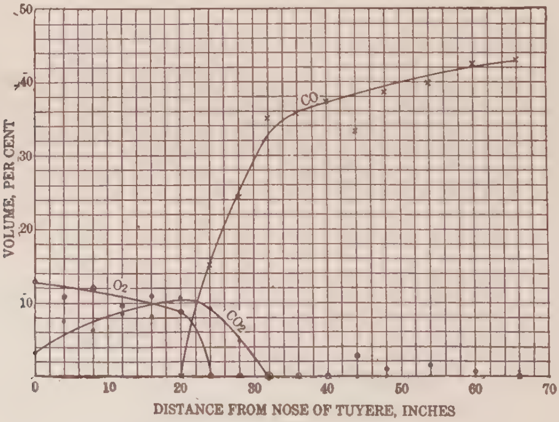

FIG. 6.—ALABAMA FURNACE NO. 5.

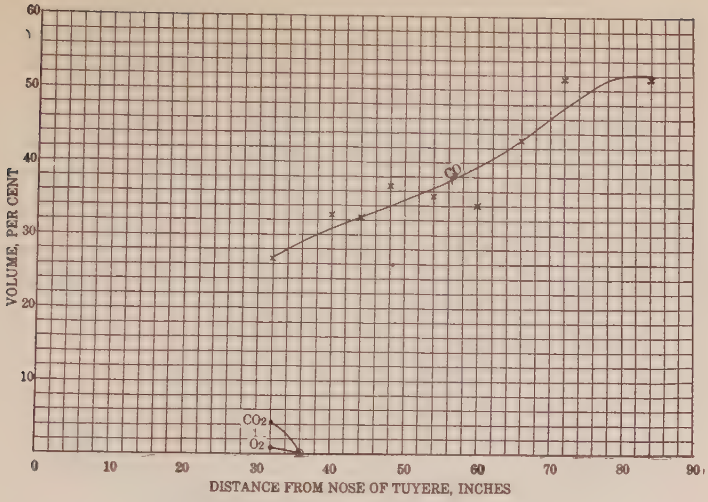

FIG. 7.—ALABAMA FURNACE No. 6.

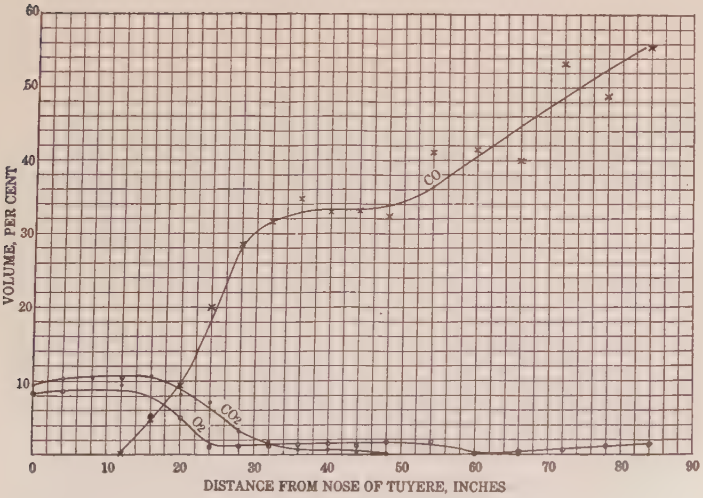

FIG. 8.—PENNSYLVANIA FURNACE No. 1.

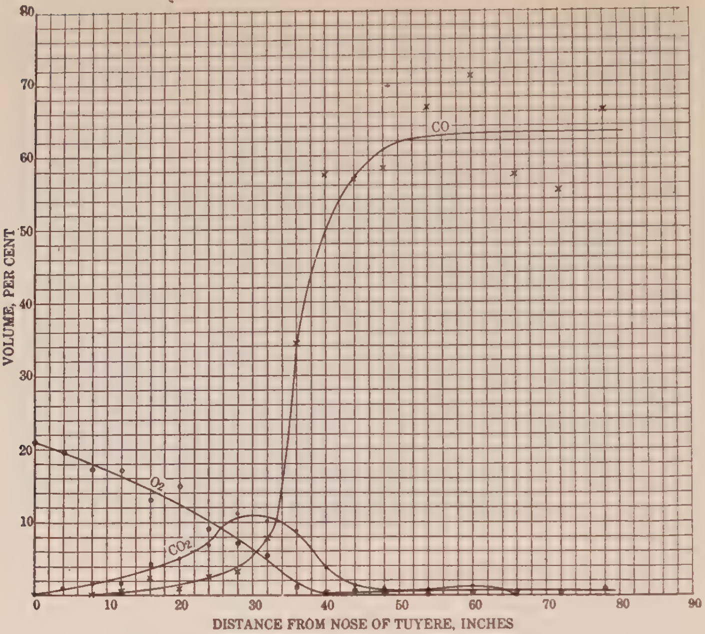

FIG. 9.—PENNSYLVANIA FURNACE NO. 2.

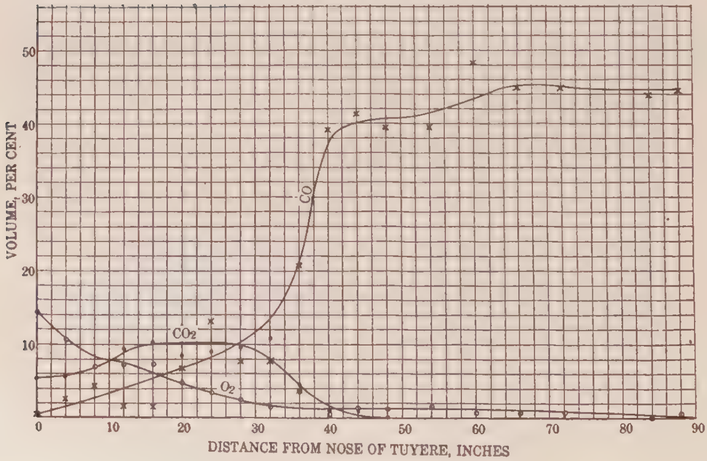

FIG. 10.—PENNSYLVANIA FURNACE NO. 3.

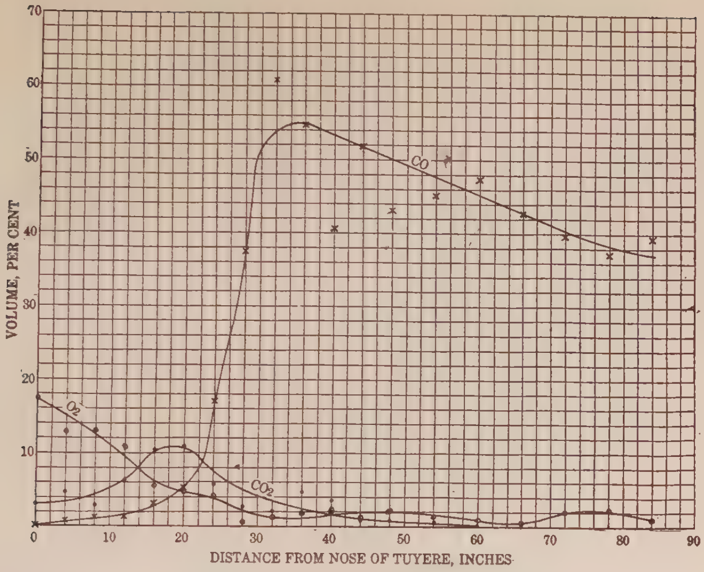

FIG. 11.—PENNSYLVANIA FURNACE NO. 4.

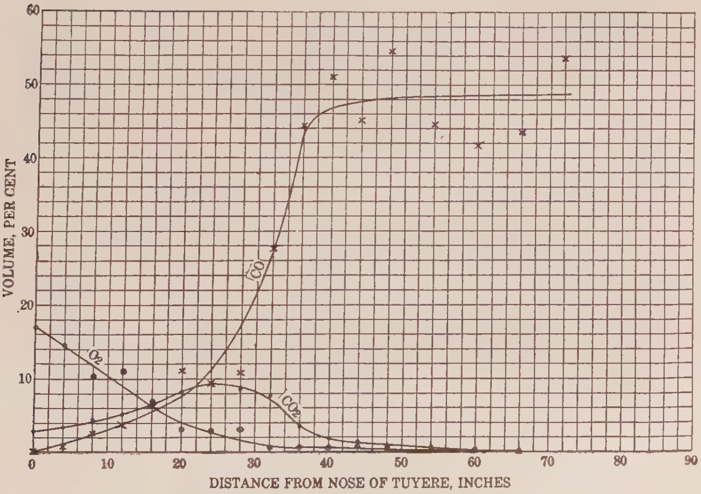

FIG. 12.—ILLINOIS FURNACE.

TABLE 3.—*Furnace Data*

Furnace	Month	Pounds Coke per Ton of Iron	Tons Iron per Day	Scrap[b] Charged, Per Cent. of Iron Made	Stone, Pounds per Ton of Iron	Flue Dust, Pounds Per Ton of Iron	Per Cent. Iron in Ore	Per Cent. Silicon in Iron	Per Cent. Carbon				Wind, Cu. Ft. Per Min. of Dry Air at 62° F. and 30 In. of Hg.		Moisture, Grains per Cu. Ft. of Dry Air	Blast Pressure, Lb. per Sq. In.	Lost Time, Min. per Day
									Stone	Ore	Flue Dust	Coke	From Engine Room Data	Calculated			
Alabama 1	March	3,160	258	0.5	1,700[e]	537	44.5	2.5	12.5	3.3	15.0	85.6	49,500	33,400	3.1	16.0	74
2	June	2,800	260	6.1	950	108	38.0	2.8	10.5	0.5	[f]	82.4	25,500	22,600	6.7	16.0	19
3	April	1,900	418	45.0	476[e]	26	38.0	1.1	12.4	3.3	12.3	87.3	41,600	29,000	4.6	13.0	30
4	March	2,645	355	6.0	0	36	35.0	3.0		3.1	[f]	87.2	41,700	33,600	3.3	15.0	20
5	May	2,765	356	5.0	134[e]	18	35.0	2.8	12.6	3.2	[f]	86.8	40,200	34,400	7.0	19.0	17
6	May	2,750	476	5.0	330[e]	22	36.0	1.1	12.4	3.2	24.0	87.5	51,400	47,600[c]	6.8	16.0	24
Penna. 1[a]	July	2,800	220		1,200		49.0		11.3	0.0		84.7	26,800	20,600	5.9	15.0	
2	August	1,700	575	0.0	1,060	58	53.0	1.3	11.2	0.0	6.0	85.8	41,700	33,500	4.9	18.0	60
3	Sept.	2,465	449	5.0	1,466	178	51.0	1.4	11.3	0.0	6.7	82.7	43,500	40,500	5.1	17.5	40
4	Sept.	2,300	514	3.3	1,358	204	51.0	1.2	11.3	0.0	6.7	81.3	43,200	39,200[d]	5.1	20.0	48
Illinois 1	Sept.	1,770	515	13.0	916	20	51.0	1.2	11.1	0.0	8.0	82.3	36,000	29,200	6.0	15.0	28

[a] Making 20 per cent. spiegeleisen; management did not wish to supply complete data. [b] Other than furnace scrap. [c] Assuming 78 per cent. carbon burned at tuyeres. [d] Assuming 75 per cent. carbon burned at tuyeres. [e] Dolomite. [f] Analyses not available.

Furnace	Blast Temp., Degrees F.	Top Temp., Degrees F.	Total Height, Feet	Diameter, Feet			Volume, Tuyeres to Stock Line, Cu. Ft.	Bosh Angle, Degrees	Tuyeres			Stack Gas Analysis, Per Cent.				Carbon Burned at Tuyeres per 100 Lb. Carbon Charged as Coke
				Hearth	Bosh	Stock Line			No.	Diam.	Length	CO_2	CO	H_2	N_2	
Alabama 1	1,100	450	86	14.5	19.7	13.0	13,500	79.0	12	6.0	12	8.5	30.1	2.8	58.6	87
2	1,180	450	79	12.3	17.8	12.0	10,386	78.0	10	5.5	10	9.3	32.4	2.8	55.5	72
3	1,100	600	66	12.8	19.6	14.0	11,974	78.0	16	5 & 6	12	7.7	32.7	2.1	57.5	78
4	1,150	500	84	16.0	20.0	13.0	16,521	80.0	12	6.0	12	8.2	32.1	3.0	56.7	77
5	1,200	365	85	13.0	20.0	13.3	15,477	74.0	12	6.0	12	7.5	33.4	2.6	56.5	77
6	1,175	500	83	17.0	22.0	17.0	19,306	80.0	12	6.0						
Penna. 1			88	15.0	15.0	15.0	17,025	78.0	10	6.0	10 & 15	8.0	32.6	3.6	55.8	75
2	1,100	400	95	19.0	22.5	17.0	23,363	80.0	10	6.0	15	16.0	25.0	3.0	56.0	74
3	1,000	355	100	18.0	22.0	16.0	23,635	81.2	15	6.0	12	10.8	27.7	3.5	58.0	82
4	900	430	100	18.0	22.0	16.0	21,358	78.0	12	6.0	12					
Illinois	1,000	230	85	17.0	20.5	15.3	19,583	80.0	9	7.0	12 & 15	15.1	25.8	3.3	55.8	73

TABLE 4.—*Source and Properties of Coke*

Furnace	Type Oven	Coking Time	Coal Seam	Moisture, Per Cent.	Vol. Matter, Per Cent.	Fixed Carbon, Per Cent.	Ash, Per Cent.	Sulfur, Per Cent.	Shatter Test,* Per Cent.	True Sp. Gr.	Apparent Sp. Gr.	Porosity, Per Cent.
Alabama 1	SS	18	Brookwood and Milldale	2.5	1.4	85.6	10.5	0.9	69	2.00	0.99	51
2	SS	20	Mary Lee	3.0	1.1	82.4	13.5	0.8	69	1.96	1.00	49
3	K	19	Pratt	3.6	1.1	87.3	8.0	1.1	70	2.04	1.14	44
4	BH	72	Pratt	0.5	1.0	87.2	11.3	1.3	64	2.13	1.15	46
5	BH	72	Pratt	0.5	1.0	86.8	11.7	1.1	66	2.08	1.13	46
6	K	19	Pratt	3.5	1.1	87.5	7.9	1.1	70	2.01	1.13	44
Penna. 1	BH	72	Pittsburgh	2.0	1.6	84.7	11.7	0.8	66	1.89	1.02	46
2	K	16	20 per cent. Pocohontas, 80 per cent. Pittsburgh	2.0	1.4	85.8	10.8	0.9		1.95	0.96	51
3	BH	48	Pittsburgh	2.0	1.1	82.7	14.2	1.1	67	1.90	0.98	48
4	K	20	10 per cent. Pocohontas, 90 per cent. Pittsburgh	4.7	1.7	81.3	12.3	1.1		1.99	1.04	48
Illinois	R	15	No. 7 Ill.	5.1	1.4	82.3	11.3	0.9	42	1.84	0.87	53

SS, Semet-Solvay byproduct ovens. K, Koppers byproduct ovens.
R, Roberts byproduct ovens. BH, beehive ovens.

* Per cent. of coke remaining on 2-in. square-mesh screen, after 50-lb. sample has been dropped four times from a height of 6 ft. upon a steel plate.

TABLE 5.—*Hearth-gas Analysis at Eleven Blast Furnaces, Summary Table**

	Analysis of Gas					Parts O_2 per 1000 Parts N_2	Per Cent. Excess Oxygen	Number of Samples	Remarks
	CO_2, Per Cent.	O_2, Per Cent.	CO, Per Cent.	H_2, Per Cent.	N_2, Per Cent.				
Distance from nose of tuyere, 0 in.									
Ala. 2	5.3	15.0	0.0	0.0	79.7	254	− 3.8	3	
4	13.4	5.6	0.0	0.0	81.0	235	− 11.0	1	
5	6.3	12.9	0.0	0.0	80.8	238	− 9.8	1	
Penna. 1	9.6	8.2	0.7	0.1	81.4	222	− 15.9	8	
2	0.0	20.9	0.0	0.0	79.1	264	0.0	7	
3	5.4	14.1	0.3	0.0	80.2	246	− 6.8	4	
4	3.3	17.5	0.0	0.0	79.2	263	− 0.4	2	
Illinois	2.9	16.8	0.1	0.0	80.2	246	− 6.8	6	
Average	6.6	12.9	0.2	0.0	80.3	244	− 7.6		
Distance from nose of tuyere, 4 in.									
Ala. 2	4.9	15.0	0.0	0.0	80.1	248	− 6.1	3	
4	9.4	10.0	0.0	0.0	80.6	240	− 9.1	1	6.5 in.
5	7.8	10.8	0.0	0.0	81.4	228	− 13.6	1	
Penna. 1	10.2	8.5	0.3	0.0	81.0	234	− 11.4	6	
2	0.8	19.6	0.0	0.0	79.6	256	− 3.0	8	
3	5.7	10.4	2.6	0.2	81.1	213	− 19.3	6	
4	4.6	12.8	0.6	0.0	82.0	216	− 18.2	3	
Illinois	3.5	14.5	0.4	0.2	81.4	222	− 15.9	9	
Average	6.6	11.7	0.6	0.1	81.0	230	− 12.9		
Distance from nose of tuyere, 8 in.									
Ala. 2	8.0	12.4	0.0	0.0	79.6	256	− 3.0	4	
4	5.0	15.8	0.0	0.0	79.2	262	− 0.8	1	10.5 in.
5	6.2	12.0	0.0	0.0	81.8	222	− 15.9	1	
Penna. 1	10.3	9.3	0.1	0.0	80.3	246	− 6.8	6	
2	1.8	17.3	0.2	0.0	80.7	238	− 9.9	8	
3	6.7	6.7	4.3	0.5	81.8	187	− 29.2	4	
4	2.9	13.0	1.1	0.2	82.8	198	− 25.0	3	
Illinois	4.4	9.5	2.8	0.2	83.1	183	− 30.6	7	
Average	6.2	11.2	1.1	0.1	81.4	220	− 16.7		
Distance from nose of tuyere, 12 in.									
Ala. 1	7.6	12.8	0.0	0.0	79.6	256	− 3.0	2	
2	16.9	1.0	2.6	1.2	78.3	238	− 9.9	2	
3	16.0	4.2	0.0	0.0	79.8	253	− 4.2	1	
4	18.0	2.0	1.0	0.8	78.2	257	− 2.7	1	
5	8.4	9.7	0.0	0.0	81.9	221	− 16.3	1	
Penna. 1	9.5	10.3	0.1	0.0	80.1	248	− 6.1	5	
2	1.7	17.2	0.4	0.1	80.6	237	− 10.2	6	
3	9.6	7.1	1.5	0.3	81.5	212	− 19.7	4	
4	6.2	10.9	1.2	0.2	81.5	216	− 18.2	6	
Illinois	5.0	11.0	3.9	0.2	79.9	224	− 15.1	6	
Average	10.8	7.7	1.1	0.3	80.1	236	− 10.6		
Distance from nose of tuyere, 16 in.									
Ala. 2	11.6	4.5	6.5	0.7	76.7	248	− 6.1	3	
3	9.2	11.7	0.0	0.0	79.1	264	0.0	2	18 in.
4	6.7	12.9	0.0	0.0	80.4	244	− 7.6	2	17 in.
5	8.1	10.9	0.0	0.0	81.0	234	− 11.4	1	
Penna. 1	10.4	5.3	5.0	0.5	78.8	228	− 13.6	7	
2	4.2	12.5	2.3	0.3	80.7	219	− 17.1	7	
3	10.1	7.2	1.3	0.1	81.3	220	− 16.7	4	
4	10.3	5.4	3.2	0.1	81.0	214	− 18.9	3	
Illinois	6.6	6.5	6.2	0.3	80.4	200	− 24.2	8	
Average	9.1	8.1	2.8	0.2	79.8	232	− 12.1		

* Penna. No. 2 not included in final average.

TABLE 5.—*Summary Table (Continued)*

	Analysis of Gas					Parts O_2 per 1000 Parts N_2	Per Cent. Excess Oxygen	Number of Samples	Remarks
	CO_2, Per Cent.	O_2, Per Cent.	CO, Per Cent.	H_2, Per Cent.	N_2, Per Cent.				
Distance from nose of tuyere, 20 in.									
Ala. 2	10.1	3.6	9.8	0.8	75.7	240	− 9.1	2	
3	10.7	3.2	10.5	0.6	75.0	252	− 4.5	5	
4	12.3	6.1	1.2	0.8	79.6	234	− 11.3	1	19 in.
5	10.5	8.7	0.0	0.0	80.8	238	− 9.8	1	
Penna. 1	8.1	4.9	9.4	0.7	76.9	226	− 14.4	8	
2	4.9	14.7	0.9	0.1	79.4	252	− 4.5	6	
3	8.2	4.5	6.7	0.4	80.2	198	− 25.0	5	
4	10.8	4.6	4.9	0.4	79.3	223	− 15.5	3	
Illinois	8.1	3.1	11.0	0.4	77.4	213	− 19.3	8	
Average	9.9	4.8	6.7	0.5	78.1	228	− 13.6		
Distance from nose of tuyere, 24 in.									
Ala. 1	10.3	8.1	2.2	0.2	79.2	245	− 7.2	10	
2	8.4	0.6	18.9	1.7	70.4	250	− 5.3	5	
3	11.9	1.8	11.1	1.5	73.7	251	− 4.9	3	
4	17.7	0.8	2.7	0.5	78.3	250	− 5.3	1	
5	9.3	0.1	15.2	1.2	74.2	221	− 16.3	1	
Penna. 1	7.0	1.0	20.0	1.4	70.6	241	− 8.7	9	
2	6.6	9.1	2.4	0.2	81.7	206	− 22.0	9	
3	9.0	3.4	13.0	0.6	74.0	252	− 4.5	5	
4	5.7	4.1	17.1	0.7	72.4	248	− 6.1	6	
Illinois	9.4	2.7	9.4	0.6	77.9	212	− 19.7	8	
Average	9.9	2.5	12.2	0.9	74.5	243	− 8.0		
Distance from nose of tuyere, 28 in.									
Ala. 1	10.5	0.1	16.2	1.4	71.8	250	− 5.3	10	30 in.
2	3.2	0.2	30.2	2.2	64.2	271	+ 2.7	5	
3	9.1	0.0	18.9	1.1	70.9	254	− 3.8	3	27 in.
4	4.7	0.0	25.4	1.6	68.3	243	− 7.9	1	27 in.
5	4.7	0.0	24.2	1.3	69.8	232	− 12.1	1	
Penna. 1	3.1	0.8	28.4	1.5	66.2	263	− 0.4	6	
2	11.0	7.0	3.2	0.0	78.8	249	− 5.7	5	
3	9.4	2.4	7.8	0.7	79.7	193	− 26.9	4	
4	2.5	0.5	37.6	1.5	57.9	364	+ 37.8	3	
Illinois	8.5	3.0	10.5	0.6	77.4	213	− 19.3	10	
Average	6.2	0.8	22.1	1.3	69.6	250	− 5.3		
Distance from nose of tuyere, 32 in.									
Ala. 2	3.3	0.8	31.9	2.2	61.8	307	+ 16.3	4	
3	8.2	0.1	21.4	1.5	70.3	260	− 1.5	4	33 in.
4	7.4	0.3	21.2	1.5	69.6	253	− 4.2	2	31 in.
5	0.0	0.0	35.0	1.9	63.1	262	− 0.8	1	33 in.
6	4.1	0.5	26.8	2.1	66.5	256	− 3.0	1	
Penna. 1	1.8	0.8	31.5	1.4	64.5	274	+ 3.8	7	
2	10.1	5.4	7.9	0.3	76.3	253	− 4.2	7	
3	10.7	1.6	7.8	0.4	79.5	201	− 23.8	4	
4	2.0	1.3	60.8	0.9	35.0	952	+260	2	
Illinois	7.7	0.4	27.8	0.8	63.3	341	+ 29.2	8	
Average	5.0	0.6	29.4	1.4	63.6	308	+ 16.7		
Distance from nose of tuyere, 36 in.									
Ala. 1	5.5	0.2	24.3	1.3	68.7	250	− 5.3	5	
2	1.5	0.1	45.9	2.1	50.4	466	+ 76.6	3	
3	0.0	0.0	37.1	2.6	60.3	287	+ 8.7	1	
4	2.4	0.1	30.3	1.7	65.5	256	− 3.0	2	
5	0.0	0.0	35.9	2.0	62.1	274	+ 3.8	1	
Penna. 1	0.3	1.2	34.7	1.6	62.2	291	+ 10.2	5	
2	8.7	1.0	34.3	0.9	55.1	479	+ 81.0	12	
3	4.6	3.8	20.7	0.5	70.4	262	− 0.8	6	
4	4.6	1.9	54.5	0.7	38.4	871	+230	4	
Illinois	3.4	0.4	44.2	0.7	51.3	499	+ 89.0	3	
Average	2.5	0.9	36.4	1.5	58.7	356	+ 34.8		

TABLE 5.—*Summary Table (Continued)*

	Analysis of Gas					Parts O_2 per 1000 Parts N_2	Per Cent. Excess Oxygen	Number of Samples	Remarks
	CO_2, Per Cent.	O_2, Per Cent.	CO, Per Cent.	H_2, Per Cent.	N_2, Per Cent.				
Distance from nose of tuyere, 40 in									
Ala. 2	1.1	0.5	47.1	1.9	49.4	492	+ 86.8	4	
3	0.0	0.0	36.2	2.3	61.5	276	+ 4.5	1	38 in.
4	2.7	0.0	30.5	1.9	66.8	254	− 3.8	2	
5	0.0	0.0	37.4	1.9	60.7	293	+ 11.0	1	
6	0.0	0.0	32.6	2.3	65.1	234	− 11.4	1	39 in.
Penna. 1	0.4	1.1	32.8	1.8	63.9	266	+ 4.5	4	
2	3.9	0.1	57.6	1.0	37.4	865	+228	2	
3	1.0	0.4	39.3	1.1	58.2	352	+ 33.4	3	
4	3.6	2.2	40.5	1.0	52.7	485	+ 83.6	4	
Illinois	1.9	0.4	51.0	0.6	46.1	596	+126	5	
Average	1.2	0.5	38.6	1.6	58.1	348	+ 31.8		
Distance from nose of tuyere, 44 in.									
Ala. 2	2.3	0.0	56.8	2.2	38.7	765	+190	2	
3	0.0	0.0	36.9	1.7	61.4	287	+ 8.7	1	46 in.
4	0.4	0.0	34.9	1.8	62.9	270	+ 2.3	3	42 in.
5	0.0	2.4	33.4	1.5	62.7	293	+ 11.0	1	
6	0.0	0.0	32.3	2.2	65.5	230	− 12.9	1	
Penna. 1	0.1	1.0	33.0	1.7	64.2	262	− 0.8	3	
2	1.1	0.5	56.8	1.0	40.6	726	+175	3	
3	0.9	1.1	41.4	1.9	54.7	398	+ 50.7	4	
4	0.9	1.0	51.9	1.1	45.1	605	+129	3	
Illinois	1.3	0.5	45.2	0.6	52.4	460	+ 74.1	5	
Average	0.7	0.7	40.6	1.6	56.4	370	+ 40.1		
Distance from nose of tuyere, 48 in.									
Ala. 1	0.2	0.3	35.0	1.5	63.0	274	+ 3.8	3	
2	1.5	0.1	47.1	1.6	49.7	491	+ 48.1	5	
3	0.0	0.0	45.5	1.8	52.7	416	+ 57.2	1	51 in.
4	0.0	0.0	36.5	2.1	61.4	278	+ 5.3	2	
5	0.0	0.8	38.5	1.0	59.7	328	+ 24.2	1	50 in.
6	0.0	0.0	36.7	2.9	60.4	280	+ 6.1	2	51 in.
Penna. 1	0.0	1.5	32.2	1.1	65.2	263	− 0.4	3	
2	0.7	0.2	58.2	1.2	39.7	742	+181	4	
3	0.0	1.0	39.6	1.1	58.3	348	+ 31.8	5	
4	0.6	2.0	43.0	0.9	53.5	443	+ 67.7	5	
Illinois	0.7	0.3	54.4	0.8	43.8	634	+140	7	
Average	0.3	0.6	40.9	1.5	56.7	363	+ 37.5		
Distance from nose of tuyere, 54 in.									
Ala. 2	3.6	0.2	43.8	1.7	50.7	491	+ 76.8	2	
3	0.0	0.0	47.7	1.8	50.5	455	+ 72.1	1	57 in.
4	0.0	0.0	34.4	2.1	63.5	255	− 3.4		
5	0.0	1.6	40.0	1.4	57.0	366	+ 38.6	1	56 in.
6	0.0	0.0	35.4	2.5	62.1	265	+ 0.4	1	55 in.
Penna. 1	0.0	1.5	41.1	1.7	55.7	381	+ 44.3	2	
2	0.5	0.0	66.5	1.1	31.9	1040	+294	2	
3	0.1	1.6	39.6	0.7	58.0	366	+ 38.6	6	
4	0.5	1.2	45.2	1.1	52.0	458	+ 73.5	1	56 in.
Illinois	0.5	0.2	44.5	0.9	53.9	418	+ 58.3	3	
Average	0.5	0.7	41.3	1.5	56.0	376	+ 42.4		
Distance from nose of tuyere, 60 in.									
Ala. 1	1.3	0.8	35.8	1.9	60.2	317	+ 20.1	1	
2	1.2	0.2	58.3	1.9	38.4	770	+192	2	
3	0.0	0.0	47.7	1.8	50.5	455	+ 72.1	1	57 in.
4	0.0	0.0	38.9	2.1	59.0	312	+ 18.2	1	
5	0.0	0.4	42.6	1.6	55.4	378	+ 43.1	1	62 in.
6	0.0	0.0	34.0	3.1	62.9	246	− 6.8	1	62 in.
Penna. 1	0.0	0.0	41.6	1.7	56.7	353	+ 33.7	2	
2	1.1	0.1	71.0	1.3	26.5	1360	+415	3	
3	0.4	0.4	48.3	1.1	49.8	490	+ 85.5	3	61 in.
4	0.0	0.8	47.5	1.2	50.5	475	+ 79.9	1	
Illinois	0.1	0.1	41.8	1.0	57.0	362	+ 37.1	5	
Average	0.3	0.3	43.7	1.7	54.0	400	+ 51.5		

TABLE 5.—*Summary Table (Continued)*

	Analysis of Gas					Parts O_2 per 1000 Parts N_2	Per Cent. Excess Oxygen	Number of Samples	Remarks
	CO_2 Per Cent.	O_2 Per Cent.	CO Per Cent.	H_2 Per Cent.	N_2 Per Cent.				
Distance from nose of tuyere, 66 in.									
Ala. 2	4.3	0.3	65.0	1.9	28.5	1270	+381	2	
3	0.0	0.0	56.9	1.4	41.7	666	+153	1	64 in.
4	0.0	0.0	42.2	2.2	55.6	360	+36.4	1	66 in.
5	0.5	0.0	43.1	1.8	54.6	388	+47.0	1	68 in.
6	0.0	0.0	43.1	2.0	54.9	375	+42.0	1	65 in.
Penna. 1	0.0	1.1	39.9	1.8	57.2	353	+33.7	2	
2	0.0	0.2	57.6	1.3	40.9	694	+163	5	
3	0.1	0.4	44.7	1.0	53.8	416	+57.5	3	69 in.
4	0.0	0.3	42.6	1.6	55.5	375	+42.0	1	68 in.
Illinois	0.3	0.1	43.6	0.7	55.3	396	+50.0	5	
Average	0.6	0.2	46.8	1.6	50.8	460	+74.1		
Distance from nose of tuyere, 72 in.									
Ala. 2	2.8	0.1	63.6	1.8	31.7	1070	+305	2	
3	0.0	0.0	56.0	1.9	42.1	645	+144	1	76 in.
4	0.0	0.0	40.3	2.1	57.6	332	+25.8	1	
6	0.0	0.0	51.5	4.3	44.2	534	+102	1	75 in.
Penna. 1	0.0	0.6	53.3	1.3	44.8	593	+125	2	
2	0.0	0.2	55.0	1.4	43.4	622	+136	3	
3	0.1	0.4	44.7	1.0	53.8	416	+57.5	3	69 in.
4	0.0	2.0	39.7	1.2	57.1	372	+40.9	2	74 in.
Illinois	0.2	0.2	53.8	0.8	45.0	598	+127	2	70 in.
Average	0.4	0.4	50.4	1.7	47.1	535	+103		
Distance from nose of tuyere, 78 in.									
Ala. 2	2.1	1.8	54.0	1.7	40.4	745	+182	2	
3	0.0	0.0	56.0	1.9	42.1	645	+144	1	76 in.
4	0.0	0.0	43.4	1.7	54.9	381	+43.5	3	
Penna. 1	0.0	1.0	48.7	1.9	48.4	505	+91.3	2	
2	0.3	0.6	66.0	1.3	31.8	1048	+297	2	
4	0.1	2.1	37.2	1.3	59.3	340	+28.8	1	
Average	0.4	1.0	47.9	1.7	49.0	521	+97.2		
Distance from nose of tuyere, 84 in.									
Ala. 3	0.0	0.0	40.6	2.1	57.3	336	+27.2	1	87 in.
4	0.0	0.0	44.1	2.1	53.8	390	+47.6	1	
6	0.0	0.0	51.6	2.5	45.9	535	+103	1	82 in.
Penna. 1	0.0	1.4	55.5	1.3	41.8	680	+157	1	
3	0.0	0.0	43.9	0.9	55.2	390	+47.7	1	82 in.
4	0.0	1.0	39.4	1.2	58.4	344	+30.3	2	82 in.
Average	0.0	0.4	45.9	1.7	52.0	433	+64		

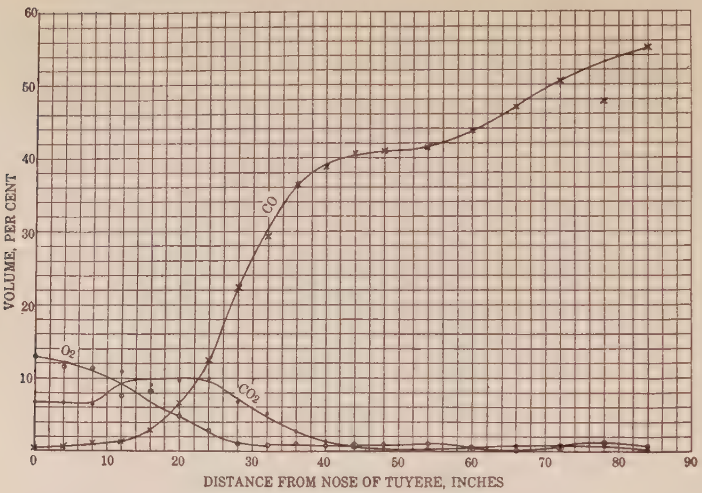

FIG. 13.—AVERAGE DATA FROM TEN FURNACES.

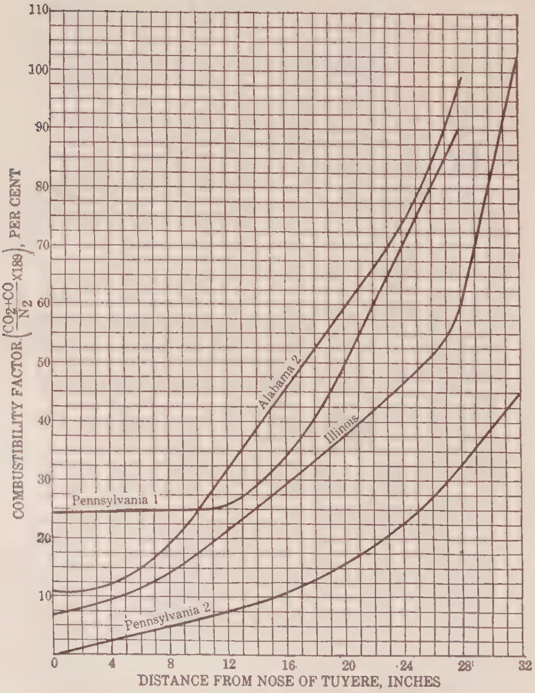

FIG. 14.—PERCENTAGE TO WHICH COMBUSTION IS COMPLETE AT DIFFERENT POINTS IN HEARTH.

TABLE 6.—*Alabama Furnace No. 1, Tuyere No. 7*

Distance from Nose of Tuyere, Inches	Analysis of Gas, Per Cent.					Time	Date 1922
	CO_2	O_2	CO	H_2	N_2		
12	7.6	13.0	0.0	0.0	79.4	1:40 p. m.	Mar. 14
	7.6	12.6	0.0	0.0	79.8	1:45	Mar. 14
	Av. 7.6	12.8	0.0	0.0	79.6		
24	7.0	11.6	0.0	0.0	81.4	12:38 p. m.	Feb. 11
	15.8	2.0	3.5	0.0	78.7	12:36	Feb. 11
	5.5	14.3	0.0	0.0	80.2	2:00	Mar. 9
	7.9	0.5	13.9	1.0	76.7	2:10	Mar. 9
	9.5	10.8	0.0	0.0	79.7	2:15	Mar. 9
	7.6	12.3	0.0	0.0	80.1	1:30	Mar. 14
	12.6	7.9	0.0	0.0	79.5	11:45 a. m.	Mar. 27
	11.7	8.7	0.0	0.0	79.6	11:46	Mar. 27
	10.7	8.9	1.3	0.5	78.6	2:00 p. m.	Apr. 18
	14.7	4.1	3.4	0.3	77.5	2:15	Apr. 18
	Av. 10.3	8.1	2.2	0.2	79.2		
28	7.9	0.1	18.0	1.0	73.0	2:16	Mar. 9
	17.8	0.4	2.6	0.2	79.0	2:20	Mar. 9
	Av. 12.9	0.3	10.3	0.6	75.9		
30	8.7	0.0	17.6	1.4	72.3	11:00 a. m.	Mar. 9
	12.6	0.2	12.1	0.8	74.3	11:10	Mar. 9
	14.3	0.2	8.9	0.6	76.0	11:50	Mar. 27
	14.0	0.2	9.1	2.1	74.6	11:51	Mar. 27
	11.8	0.1	14.8	1.3	72.0	2:00	Apr. 18
	10.2	0.1	17.5	1.1	71.1	2:02	Mar. 18
	10.2	0.0	17.7	1.2	70.9	2:05	Mar. 18
	2.5	0.0	32.0	2.7	62.8	2:30	Mar. 18
	Av. 10.5	0.1	16.2	1.4	71.8		
36	2.9	0.2	29.4	2.1	65.4	2:25	Mar. 9
	3.5	0.0	26.2	0.5	69.8	2:30	Mar. 9
	2.2	0.0	30.5	1.7	65.6	11:55	Mar. 27
	12.0	0.5	13.0	1.2	73.3	2:25	Apr. 18
	7.0	0.0	22.4	1.2	69.4	1:00	Apr. 14
	Av. 5.5	0.2	24.3	1.3	68.7		
48	0.0	0.0	34.6	1.1	64.3	1:40	Mar. 14
	0.5	0.6	33.8	1.5	63.6	1:43	Mar. 14
	0.0	0.3	36.6	1.9	61.2	12:01	Mar. 27
	Av. 0.2	0.3	35.0	1.5	63.0		
60	1.3	0.8	35.8	1.9	60.2	2:25	Mar. 28

TABLE 7.—*Alabama Furnace No. 2*

Distance from Nose of Tuyere, Inches	Analysis of Gas, Per Cent.					Time	Date 1922	Tuyere
	CO_2	O_2	CO	H_2	N_2			
0	4.4	15.9	0.0	0.0	79.7	11:00 a.m.	June 27	5
	8.7	11.6	0.0	0.0	79.7	11:00	June 27	5
	2.7	17.4	0.0	0.0	79.9	1:30 p.m.	June 28	5
	Av. 5.3	15.0	0.0	0.0	79.7			
4	4.2	16.8	0.0	0.0	79.0	11:15 a.m.	June 27	5
	8.4	12.3	0.0	0.0	79.3	11:00	June 27	5
	3.8	15.8	0.0	0.0	80.4	1:30 p.m.	June 28	5
	Av. 4.9	15.0	0.0	0.0	80.1			
8	8.7	11.4	0.0	0.0	79.9	11:15 a.m.	June 27	5
	9.0	11.9	0.0	0.0	79.1	11:00	June 27	5
	8.8	11.7	0.0	0.0	79.5	11:30	June 27	5
	5.5	14.7	0.0	0.0	79.8	1:30 p.m.	June 28	5
	Av. 8.0	12.4	0.0	0.0	79.6			
12	4.5	16.0	0.0	0.0	79.5	2:00 p.m.	Mar. 20	4
	3.5	16.8	0.0	0.0	79.7	2:00	Mar. 20	4
	8.2	12.3	0.0	0.0	79.5	11:00	Mar. 21	5
	7.5	12.8	0.0	0.0	79.7	11:00 a.m.	Mar. 21	5
	Av. Mar. 5.9	14.5	0.0	0.0	79.6			
12	18.6	1.3	1.8	0.5	79.8	11:00 a.m.	June 27	5
	15.2	0.7	3.4	1.9	78.8	1:30 p.m.	June 28	5
	Av. June 16.9	1.0	2.6	1.2	78.3			
16	13.7	0.8	9.7	0.9	74.9	11:00 a.m.	June 27	5
	14.6	1.6	6.2	0.7	76.9	11:30	June 27	5
	7.3	11.0	3.5	0.5	77.7	1:30 p.m	June 28	5
	Av. 11.6	4.5	6.5	0.7	76.7			
20	10.1	0.0	17.9	1.3	70.7	11:00 a.m.	June 27	5
	10.0	7.2	1.7	0.3	80.8	1:30 p.m.	June 28	5
	Av. 10.1	3.6	9.8	0.8	75.7			
24	5.9	0.5	24.0	1.6	69.0	2:00 p.m.	Mar. 20	4
	8.6	0.5	19.6	1.9	69.4	2:00	Mar. 20	4
	6.0	14.7	0.0	0.0	79.3	11:00 a.m.	Mar. 21	5
	7.1	13.4	0.0	0.0	79.5	11:00	Mar. 21	5
	5.4	15.1	0.0	0.0	79.5	2:00 p.m.	Mar. 22	3
	10.8	10.1	0.0	0.0	79.1	2:00	Mar. 22	3
	7.2	13.1	0.0	0.0	79.7	2:00	Mar. 22	3
	7.0	13.5	0.0	0.0	79.5	9:00 a.m.	Mar. 23	3
	Av. Mar. 7.3	10.1	5.5	0.4	76.7			
24	12.5	2.1	6.1	0.3	79.0	11:15 a.m	June 26	5
	10.7	0.4	16.5	1.0	71.4	11:00	June 27	5
	12.0	0.0	13.4	1.8	72.8	10:30	June 27	5
	5.8	0.3	25.3	1.9	66.7	11:30	June 27	5
	1.1	0.0	32.9	3.5	62.5	1:30 p.m.	June 28	5
	Av. June 8.4	0.6	18.9	1.7	70.4			
28	8.8	0.3	19.5	1.7	69.7	11:00 a.m.	Mar. 21	5
	1.5	0.2	31.4	1.7	65.2	9:00	Mar. 22	4
	4.7	0.4	26.2	1.1	67.6	9:00	Mar. 22	4
	8.0	0.2	21.0	1.0	69.8	2:00 p.m.	Mar. 22	4
	Av. Mar. 5.8	0.3	24.5	1.4	68.0			

TABLE 7.—*Alabama Furnace No. 2 (Continued)*

Distance from Nose of Tuyere, Inches	Analysis of Gas, Per Cent.					Time	Date 1922	Tuyere
	CO_2	O_2	CO	H_2	N_2			
28	2.1	0.3	31.6	1.9	64.1	11:15 a.m.	June 26	5
	1.8	0.4	31.6	2.4	63.8	11:00	June 27	5
	11.6	0.5	13.8	1.7	72.4	10:30	June 27	5
	0.3	0.0	37.0	2.1	60.6	11:30	June 27	5
	0.0	0.0	37.1	2.9	60.0	1:30 p.m.	June 28	5
	Av. June 3.2	0.2	30.2	2.2	64.2			
32	10.0	1.5	23.9	1.9	62.7	10:30	June 27	5
	1.9	0.2	32.9	2.4	62.6	11:00	June 27	5
	1.1	1.3	34.7	1.6	61.3	11:15	June 26	5
	0.0	0.0	36.2	2.8	61.0	1:30 p.m.	June 28	5
	Av. 3.3	0.8	31.9	2.2	61.8			
36	10.0	1.0	17.0	1.5	70.5	2:00 p.m.	Mar. 21	5
	3.2	0.0	29.0	1.2	66.6	11:00 a.m.	Mar. 22	4
	8.0	0.2	21.0	1.3	69.5	9:00	Mar. 22	3
	1.2	0.3	32.2	0.8	65.5	2:00 p.m.	Mar. 22	3
	0.5	0.2	33.0	1.1	65.2	2:00	Mar. 22	3
	1.5	0.3	31.9	1.1	65.2	9:00 a.m.	Mar. 23	3
	Av. Mar. 4.1	0.3	27.4	1.2	67.0			
36	2.7	0.0	35.6	1.8	59.9	11:15	June 26	5
	1.6	0.3	57.1	2.1	38.9	10:30	June 27	5
	0.2	0.0	45.0	2.3	52.5	11:30	June 27	5
	Av. June 1.5	0.1	45.9	2.1	50.4			
40	3.1	1.3	40.4	1.4	53.8	11:15 a.m.	June 26	5
	1.2	0.2	69.6	1.3	27.7	10:30	June 27	5
	0.0	0.3	41.9	2.2	55.6	11:30	June 27	5
	0.0	0.0	36.5	2.7	60.8	1:30 p.m.	June 28	5
	Av. 1.1	0.5	47.1	1.9	49.4			
44	2.8	0.0	40.7	2.1	54.4	11:15 a.m.	June 26	5
	1.7	0.0	72.9	2.2	23.2	10:30	June 27	5
	Av. 2.3	0.0	56.8	2.2	38.7			
48	1.9	0.3	43.1	1.3	53.4	10:30	June 26	5
	3.3	0.0	44.9	0.0	51.8	11:15	June 26	5
	2.0	0.0	65.2	1.9	30.9	10:30	June 27	5
	0.3	0.3	46.5	2.1	50.8	11:30	June 27	5
	0.0	0.0	35.8	2.7	61.5	1:30 p.m.	June 28	5
	Av. 1.5	0.1	47.1	1.6	49.7			
54	2.8	0.0	42.9	2.0	52.3	10:30 a.m.	June 26	5
	4.4	0.3	44.7	1.3	49.3	11:15	June 26	5
	Av. 3.6	0.2	43.8	1.7	50.7			
60	2.0	0.3	57.5	1.7	38.5	11:15	June 26	5
	0.4	0.0	59.0	2.0	38.6	11:30	June 27	5
	Av. 1.2	0.2	58.3	1.9	38.4			
66	6.3	0.2	71.6	1.2	20.7	10:30 a.m.	June 26	5
	2.3	0.3	58.3	2.5	36.6	11:15	June 26	5
	Av. 4.3	0.3	65.0	1.9	28.5			
72	3.7	0.2	63.3	1.9	30.9	11:15	June 26	5
	1.9	0.0	63.8	1.6	32.7	11:30	June 27	5
	Av. 2.8	0.1	63.6	1.8	31.7			
78	2.1	1.8	54.0	1.7	40.4	11:15	June 26	5

TABLE 8.—*Alabama Furnace No. 3, Tuyere No. 6*

Distance from Nose of Tuyere, Inches	Analysis of Gas, Per Cent.					Time	April, 1922
	CO_2	O_2	CO	H_2	N_2		
12	16.0	4.2	0.0	0.0	79.8	4:20 p.m.	24
18	8.2	12.2	0.0	0.0	79.6	4:45	25
	10.1	11.1	0.0	0.0	78.8	12:01	25
	Av. 9.2	11.7	0.0	0.0	79.1		
20	8.7	0.2	18.0	1.3	71.8	3:30	28
	7.6	11.9	0.0	0.0	80.5	11:15 a.m.	24
	10.3	0.0	17.4	0.5	71.8	3:00 p.m.	24
	10.7	0.0	16.8	1.2	71.3	3:00	24
	16.0	4.0	0.2	0.2	79.6	4:20	24
	Av. 10.7	3.2	10.5	0.6	75.0		
24	14.0	0.5	10.9	0.7	73.9	12:01	25
	7.7	0.8	19.1	2.7	69.7	4:45	25
	14.1	4.1	3.3	1.1	77.4	4:45	25
	Av. 11.9	1.8	11.1	1.5	73.7		
27	11.9	0.0	13.8	0.8	73.5	4:20	24
	12.8	0.0	12.3	0.9	74.0	4:20	24
	2.5	0.0	30.5	1.7	65.3	3:30	28
	Av. 9.1	0.0	18.9	1.1	70.9		
30	13.5	0.0	12.3	1.4	72.8	4:00	26
	12.5	0.0	14.0	1.4	72.1	4:00	26
	9.2	0.0	17.7	0.8	72.3	3:50	24
	9.0	0.5	18.0	0.9	71.6	3:50	24
	Av. 11.1	0.1	15.5	1.1	72.2		
33	13.6	0.2	12.0	0.8	73.4	12:01	25
	13.2	0.0	12.7	1.0	73.1	12:01	25
	2.7	0.3	30.1	2.2	64.7	4:45	25
	3.2	0.0	30.6	2.1	64.1	4:45	25
	Av. 8.2	0.1	21.4	1.5	70.3		
36	0.0	0.0	37.1	2.6	60.3	11:30 a.m.	28
38	0.0	0.0	36.2	2.3	61.5	3:30 p.m.	28
46	0.0	0.0	36.9	1.7	61.4	3:30	28
51	0.0	0.0	45.5	1.8	52.7	3:30	28
57	0.0	0.0	47.7	1.8	50.5	3:30	28
64	0.0	0.0	56.9	1.4	41.7	3:30	28
76	0.0	0.0	56.0	1.9	42.1	3:30	28
87	0.0	0.0	40.6	2.1	57.3	3:30	28
98	0.0	0.0	36.0	2.7	61.3	3:30	28

TABLE 9.—*Alabama Furnace No. 4*

Distance from Nose of Tuyere, Inches	Analysis of Gas, Per Cent.					Time	Date 1922	Tuyere No.
	CO_2	O_2	CO	H_2	N_2			
0	13.4	5.6	0.0	0.0	81.0	11:00 a.m.	May 31	6
6.5	9.4	10.0	0.0	0.0	80.6	11:00	May 31	6
10.5	5.0	15.8	0.0	0.0	79.2	11:00	May 31	6
12.0	18.0	2.0	1.0	0.8	78.2	4:00 p.m.	Apr. 27	9
14.5	15.8	3.8	0.5	0.3	79.6	11:00 a.m.	May 31	6
17	7.3	12.8	0.0	0.0	79.9	10:00 a.m.	Apr. 26	6
	6.3	13.0	0.0	0.0	80.7	10:00	Apr. 26	6
	Av. 6.7	12.9	0.0	0.0	80.4			
18	15.5	4.5	0.0	0.0	80.0	4:00 p.m.	Apr. 27	9
	15.0	3.1	2.8	0.9	78.2	11:00 a.m.	May 31	6
	Av. 15.3	3.8	1.4	0.5	79.0			
19	12.3	6.1	1.2	0.8	79.6	11:00 a.m.	Apr. 27	6
22.5	8.7	0.2	17.6	2.0	71.5	11:00	May 31	6
24	17.7	0.8	2.7	0.5	78.3	4:00 p.m.	Apr. 27	9
26.5	4.7	0.0	25.4	1.6	68.3	11:00 a.m.	May 31	6
28	12.2	8.1	0.0	0.0	79.7	10:00	Apr. 26	6
	12.1	7.9	0.0	0.0	80.0	10:00	Apr. 26	6
	Av. 12.2	8.0	0.0	0.0	79.8			
30	13.5	1.1	10.4	1.3	73.7	4:00 p.m.	Apr. 27	9
31	12.5	0.5	11.7	1.0	74.3	11:00 a.m.	Apr. 27	6
	2.2	0.0	30.7	2.0	65.1	11:00	May 31	6
	Av. 7.4	0.3	21.2	1.5	69.6			
34.5	1.6	0.0	32.1	2.7	63.6	11:00 a.m.	May 31	6
36.0	0.3	0.1	35.2	1.7	62.7	11:00 a.m.	Apr. 27	6
	4.5	0.0	25.4	1.6	68.5	4:00 p.m.	Apr. 27	9
	Av. 2.4	0.1	30.3	1.7	65.5			
38.5	0.0	0.0	34.9	2.0	63.1	11:00 a.m.	May 31	6
40	2.8	0.0	30.4	1.7	65.1	10:00 a.m.	Apr. 26	6
	2.5	0.0	30.6	2.1	64.8	10:00	Apr. 26	6
	Av. 2.7	0.0	30.5	1.9	66.8			
42	0.0	0.0	35.6	1.6	62.8	4:00 p.m.	Apr. 27	9
	1.2	0.0	32.8	1.8	64.2	11:00 a.m.	Apr. 27	6
	0.0	0.0	36.4	2.0	61.6	11:00	May 31	6
	Av. 0.4	0.0	34.9	1.8	62.9			
48	0.0	0.0	36.2	2.2	61.6	11:00 a.m.	Apr. 27	6
	0.0	0.0	36.7	2.0	61.3	11:00	May 31	6
	Av. 0.0	0.0	36.5	2.1	61.4			
53	0.0	0.0	34.4	2.1	63.5	10:00 a.m.	Apr. 26	6
	0.0	0.0	34.2	1.9	63.9	10:00	Apr. 26	6
54	0.0	0.0	34.6	2.4	63.0	11:00	Apr. 27	6
	Av. 0.0	0.0	34.4	2.1	63.5			
60	0.0	0.0	38.9	2.1	59.0	11:00 a.m.	Apr. 27	6
64	0.0	0.0	36.0	1.9	62.1	10:00	Apr. 26	6
	0.0	0.0	37.2	1.7	61.1	10:00	Apr. 26	6
	Av. 0.0	0.0	36.6	1.8	61.6			
66	0.0	0.0	42.2	2.2	65.6	4:00 p.m.	Apr. 27	9
70	0.0	0.0	40.7	1.6	57.7	11:00 a.m.	Apr. 27	6
72	0.0	0.0	40.3	2.1	57.6	11:00	Apr. 27	6
78	0.0	0.0	44.5	1.8	53.7	9:00 a.m.	Apr. 26	6
	0.0	0.0	44.2	1.5	54.3	9:00	Apr. 26	6
	0.0	0.0	41.6	1.7	56.7	4:00 p.m.	Apr. 27	9
	Av. 0.0	0.0	43.4	1.7	54.9			
84	0.0	0.0	44.1	2.1	53.8	11:00 a.m.	Apr. 27	6
90	0.0	0.0	43.2	1.8	55.0	11:00 a.m.	Apr. 27	6
	0.0	0.0	45.3	2.0	52.7	11:00	Apr. 27	6
	Av. 0.0	0.0	44.3	1.9	53.8			
91	0.0	0.0	39.2	1.9	58.9	4:00 p.m.	Apr. 27	9

TABLE 10.—*Alabama Furnace No. 5, Tuyere No. 8*

Distance from Nose of Tuyere, Inches	Analysis of Gas, Per Cent.					Time	Date 1922
	CO_2	O_2	CO	H_2	N_2		
0	6.3	12.9	0.0	0.0	80.8	4:00 p.m.	May 31
4	7.8	10.8	0.0	0.0	81.4		
8	6.2	12.0	0.0	0.0	81.8		
12	8.4	9.7	0.0	0.0	81.9		
16	8.1	10.9	0.0	0.0	81.0		
20	10.5	8.7	0.0	0.0	80.8		
24	9.3	0.1	15.2	1.2	74.2		
28	4.7	0.0	24.2	1.3	69.8		
32	0.0	0.0	35.0	1.9	63.1		
36	0.0	0.0	35.9	2.0	62.1		
40	0.0	0.0	37.4	1.9	60.7		
44	0.0	2.4	33.4	1.5	62.7		
50	0.0	0.8	38.5	1.0	59.7		
56	0.0	1.6	40.0	1.4	57.0		
62	0.0	0.4	42.6	1.6	55.4		
68	0.5	0.0	43.1	1.8	54.6		

TABLE 11.—*Alabama Furnace No. 6*

Distance from Nose of Tuyere, Inches	Analysis of Gas, Per Cent.					Time	May, 1922	Tuyere No.
	CO_2	O_2	CO	H_2	N_2			
33	4.1	0.5	26.8	2.1	66.5	1:30 p.m.	27	2
39	0.0	0.0	32.6	2.3	65.1	1:30	27	2
44	0.0	0.0	32.3	2.2	65.5	1:30	27	2
51	0.0	0.0	38.0	2.9	59.1	1:30 p.m.	24	2
	0.0	0.0	35.4	2.8	61.8	1:30	27	8
	Av. 0.0	0.0	36.7	2.9	60.4			
55	0.0	0.0	35.4	2.5	62.1	1:30 p.m.	27	2
57	0.0	0.0	35.8	3.4	60.8	1:30	24	2
62	0.0	0.0	34.0	3.1	62.9	1:30	24	2
65	0.0	0.0	43.1	2.0	54.9	1:30	27	2
69	0.0	0.0	41.2	3.3	55.5	1:30	24	2
75	0.0	0.0	51.5	4.3	44.2	1:30	24	2
82	0.0	0.0	51.6	2.5	45.9	1:30	27	8

$$\frac{O_2 + CO_2 + \frac{1}{2}CO}{N_2} \times 1000 = 264.$$

Moist air at the temperature in the hearth forms, in addition, H_2 and CO from the water-gas reaction. The formula corrected to dry air is as follows:

$$\frac{O_2 + CO_2 + \frac{1}{2}(CO - H_2)}{N_2} \times 1000 = 264$$

This latter formula has been used in calculating the results in Table 5; it neglects the nitrogen and hydrogen in the fuel. A value less than 264 indicates that other elements than carbon are being burned by the blast, *e.g.*, iron; and a value greater than 264 indicates that carbon is being oxidized from some other source than the blast, *e.g.*, by silica, or iron or manganese oxides, iron silicate in the slag, or in connection with the desulfurization of iron by slag. No nitrogen accompanies the oxygen in these latter reactions, hence the oxides of carbon so produced tend to increase the ratio of total oxygen to nitrogen in the gass.

Discussion of Results

The tables and curves show that combustion in the hearth is rapid, the oxygen diminishing to practically zero at 24 to 30 in. from the tuyeres and carbon dioxide practically disappearing at 32 to 40 in. In other words, the oxygen of the blast has completed its work of gasifying carbon at about 3 ft. from the tuyeres. In the remainder of the hearth, the gases are inert in so far as the coke is concerned and consumption of fuel can take place only by means of oxygen from some other source than the blast. The occurrence of such oxidation will be evidenced by a ratio of total oxygen to nitrogen greater than that in atmospheric air.

Oxygen deficiency is apparent about 32 in. from the tuyeres, after which point an increasing excess of oxygen is evident. The deficiency, as previously pointed out, is probably caused by oxidation of iron at the tuyeres. The excess ratio of oxygen to nitrogen that appears in the central regions of the hearth indicates direct reduction of silica, or manganese or iron oxides. No conclusions can be drawn as to the amount of this direct reduction, because no data are at hand as to the percentage of the blast gases passing through a given point. Single analyses often show 80 per cent. and more carbon monoxide; in such cases, the end of the sampling tube must have been in a region where only a small percentage of the blast gases was passing through.

The first combustion at the tuyere is apparently the burning of hearth gas swept into the blast as it leaves the tuyeres. This is shown by the fact that a large percentage of carbon dioxide is present at the nose of the tuyere. The percentage of carbon dioxide near the tuyere is, in general,

TABLE 12.—*Pennsylvania Furnace No. 1*

(Operating on ferromanganese, July 27 and 31, and on spiegeleisen, Aug. 4)

Distance from Nose of Tuyere, Inches	Analysis of Gases, Per Cent.					Series[a]
	CO_2	O_2	CO	H_2	N_2	
− 6	0.3	20.6	0.0	0.0	79.1	5
− 4	0.2	20.8	0.0	0.0	79.0	5
− 3	0.7	20.2	0.0	0.0	79.1	7
0	10.4	8.1	0.6	0.3	80.6	6
0	9.7	3.5	2.5	0.0	84.3	5
0	11.8	5.9	0.5	0.0	81.0	5
0	11.6	7.3	0.9	0.1	80.1	6
0	8.3	11.9	0.0	0.0	79.8	3
0	7.9	7.9	0.0	0.0	84.2	1
0	8.4	9.4	1.0	0.6	80.6	7
0	8.5	11.6	0.0	0.0	79.9	8
	Av. 9.6	8.2	0.7	0.1	81.4	
4	11.4	7.9	0.3	0.0	80.4	5
4	8.4	11.4	0.0	0.0	80.2	6
4	10.5	7.5	0.8	0.0	81.2	5
4	4.2	12.5	0.0	0.0	83.3	1
4	13.5	5.8	0.3	0.2	80.5	7
4	13.2	6.1	0.3	0.0	80.4	8
	Av. 10.2	8.5	0.3	0.0	81.0	
8	7.8	12.2	0.2	0.0	79.8	5
8	9.7	9.4	0.0	0.0	80.9	5
8	8.2	11.8	0.2	0.0	79.8	6
8	8.4	11.6	0.0	0.0	80.0	3
8*	3.9	0.3	27.1	1.8	66.9	1
8	13.7	4.8	0.3	0.0	81.2	7
8	13.9	5.9	0.0	0.0	80.2	8
	Av. 10.3	9.3	0.1	0.0	80.3	
12	6.8	13.4	0.0	0.0	79.8	6
12	9.0	10.1	0.3	0.2	80.4	5
12	8.3	11.6	0.2	0.0	79.9	3
12*	4.5	0.0	25.2	1.6	68.7	1
12	11.7	7.9	0.0	0.0	80.4	7
12	11.6	8.7	0.0	0.0	79.7	8
	Av. 9.5	10.3	0.1	0.0	80.1	
16	10.2	0.5	11.5	0.6	77.2	6
16	11.0	5.1	0.9	0.3	82.7	6
16	10.4	9.4	0.0	0.6	79.6	5
16	9.1	10.9	0.0	0.0	80.0	3
16	2.6	2.3	21.0	1.4	72.7	1
16	13.6	5.7	0.3	0.0	80.4	
16	15.7	3.0	1.1	0.3	79.9	
	Av. 10.4	5.3	5.0	0.5	78.8	
20	8.9	10.9	0.0	0.3	79.9	5
20	0.0	0.5	32.6	1.7	65.2	6
20	7.5	11.4	0.0	0.0	81.1	3
20	11.3	2.8	5.6	0.6	79.7	1
20	0.2	0.2	33.4	1.7	64.5	6
20	12.0	0.5	3.6	0.8	83.1	4
20	9.8	8.6	0.0	0.1	81.5	
20	14.9	4.3	0.0	0.0	80.8	
	Av. 8.1	4.9	9.4	0.7	76.9	
24	15.5	2.0	6.1	1.4	75.0	5
24	0.0	0.4	33.4	2.2	64.0	6
24	0.8	0.0	31.8	2.0	65.4	6
24	7.0	0.0	20.2	1.4	71.4	4
24	16.6	1.5	2.8	0.2	78.9	3
24	0.3	2.4	27.4	1.8	68.1	2
24	1.2	0.0	32.9	1.6	64.3	1
24	11.1	1.2	11.2	0.7	75.8	7
24	10.4	1.5	13.9	0.9	74.3	8
	Av. 7.0	1.0	20.0	1.4	70.6	
26*	15.3	2.5	3.1	1.8	77.3	5
28	3.2	0.0	26.0	1.6	69.2	4
28	0.8	0.3	31.8	2.0	65.1	4
28	0.0	1.8	44.1	2.2	51.9	2
28	1.8	1.7	30.6	1.6	64.3	1
28	6.7	0.5	16.5	0.7	75.6	7
28	6.3	0.4	21.6	0.8	70.9	8
	Av. 3.1	0.8	28.4	1.5	66.2	

[a] Series 1 taken between 10:30 a.m. and 11:00 a.m. July 27, tuyere 6.
Series 2 taken between 11:05 a.m. and 11:15 a.m. July 27, tuyere 6.
Series 3 taken between 11:35 a.m. and 11:55 a.m. July 27, tuyere 6.
Series 4 taken between 10:00 a.m. and 10:20 a.m. July 31, tuyere 3.
Series 5 taken between 10:25 a.m. and 10:50 a.m. July 31, tuyere 3.
Series 6 taken between 10:55 a.m. and 11:15 a.m. July 31, tuyere 3
Series 7 taken between 10:00 a.m. and 10:30 a.m. Aug. 4, tuyere 6.
Series 8 taken between 10:35 a.m. and 10:50 a.m. Aug. 4, tuyere 6.

* Not in average.

Table 12.—*Pennsylvania Furnace No.* 1 (*Continued*)

(Operating on ferromanganese July 27 and 31, and on spiegeleisen, Aug. 4)

Distance from Nose of Tuyere, Inches	Analysis of Gases, Per Cent.					Series[a]
	CO_2	O_2	CO	H_2	N_2	
32	0.5	0.7	31.2	0.8	66.8	4
32	0.0	0.0	40.2	1.6	58.2	4
32	0.0	0.5	34.1	2.1	63.3	3
32	0.1	2.3	36.4	1.5	59.7	2
32	1.6	1.1	28.7	1.5	67.1	1
32	0.0	0.0	35.2	1.3	63.5	7
32	9.2	0.8	14.8	1.1	74.1	8
	Av. 1.8	0.8	31.5	1.4	64.5	
36	0.0	0.0	33.8	1.8	64.4	4
36	0.2	0.2	45.7	1.6	52.3	4
36	0.0	1.9	31.4	1.6	65.1	2
36	1.4	3.7	26.7	1.1	67.1	1
36	0.0	0.0	36.1	1.7	63.2	7
	Av. 0.3	1.2	34.7	1.6	62.2	
38	0.0	0.0	35.4	2.0	62.6	4
38	0.0	0.3	43.6	1.5	54.6	4
	Av. 0.0	0.2	39.5	1.8	58.5	
40	0.0	2.7	29.4	1.5	66.4	2
40	0.0	1.6	31.9	1.8	64.7	2
40	1.7	0.0	33.5	1.7	63.1	1
40	0.0	0.0	36.3	2.0	61.7	7
	Av. 0.4	1.1	32.8	1.8	63.9	
44	0.2	2.0	29.0	1.7	67.1	2
44	0.0	0.6	35.0	2.3	62.1	1
44	0.0	0.3	35.0	1.0	63.7	7
	Av. 0.1	1.0	33.0	1.7	64.2	
48	0.0	3.9	26.6	1.3	68.2	2
48	0.0	0.5	34.9	1.3	63.3	1
48	0.0	0.0	35.2	0.7	64.1	7
	Av. 0.0	1.5	32.2	1.1	65.2	
54	0.0	2.9	46.6	1.5	49.0	1
54	0.0	0.0	35.6	1.9	62.5	7
	Av. 0.0	1.5	41.1	1.7	55.7	
60	0.0	0.0	47.1	1.6	51.3	1
	0.0	0.0	36.1	1.7	62.2	7
	Av. 0.0	0.0	41.6	1.7	56.7	
66	0.0	2.2	40.8	1.9	55.1	1
	0.0	0.0	39.0	1.6	59.4	7
	Av. 0.0	1.1	39.9	1.8	57.2	
72	0.0	1.1	43.0	1.6	54.3	1
	0.0	0.0	63.6	1.0	35.4	7
	Av. 0.0	0.6	53.3	1.3	44.8	
78	0.0	0.2	44.2	2.0	53.6	1
	0.0	1.7	53.2	1.7	43.4	7
	Av. 0.0	1.0	48.7	1.9	48.4	
84	0.0	1.4	55.5	1.3	41.8	7

TABLE 13.—*Pennsylvania Furnace No. 2, Tuyere No. 6*

Distance from Nose of Tuyere, Inches	Analysis of Gas, Per Cent.					Time, P.M.	August, 1922
	CO_2	O_2	CO	H_2	N_2		
− 3	0.0	21.1	0.0	0.0	78.9	1:40	9
	0.0	21.1	0.0	0.0	78.9	1:38	9
	0.0	21.1	0.0	0.0	78.9	1:55	11
	0.0	20.9	0.0	0.0	79.1	2:52	11
	0.0	21.0	0.0	0.0	79.0	2:44	11
	Av. 0.0	21.1	0.0	0.0	78.9		
0	0.0	20.6	0.0	0.0	79.4	1:37	9
	0.0	21.2	0.0	0.0	78.8	1:13	9
	0.0	21.0	0.0	0.0	79.0	1:54	11
	0.2	21.0	0.0	0.0	78.8	2:50	11
	0.0	20.9	0.0	0.0	79.1	2:44	11
	0.0	20.8	0.0	0.0	79.2	1:32	15
	0.0	20.9	0.0	0.0	79.1	3:00	15
	Av. 0.0	20.9	0.0	0.0	79.1		
4	0.0	20.9	0.0	0.0	79.1	1:35	9
	0.0	21.0	0.0	0.0	79.0	1:12	9
	0.0	20.9	0.0	0.0	79.1	1:53	11
	2.8	16.8	0.0	0.0	80.4	2:48	11
	0.3	20.4	0.0	0.0	79.3	2:43	11
	0.6	20.4	0.0	0.0	79.0	1:31	15
	1.3	17.3	0.0	0.0	81.4	2:05	15
	1.7	19.1	0.0	0.0	79.2	2:59½	15
	Av. 0.8	19.6	0.0	0.0	79.6		
8	0.0	21.2	0.0	0.0	78.8	1:34	9
	0.0	20.9	0.0	0.0	79.1	1:10	9
	1.5	17.2	0.0	0.0	81.3	1:52	11
	3.1	16.5	0.6	0.2	79.6	2:47	11
	5.4	8.3	1.2	0.1	85.0	2:43	11
	1.1	19.0	0.0	0.0	79.9	1:30	15
	0.9	17.6	0.0	0.0	81.5	2:03	15
	2.6	17.4	0.0	0.0	80.0	2:59	15
	Av. 1.8	17.3	0.2	0.0	80.7		
12	0.0	21.0	0.0	0.0	79.0	1:08	9
	0.0	21.0	0.0	0.0	79.0	1:32	9
	4.0	13.7	1.6	0.7	80.0	1:50	11
	10.8	0.7	1.2	0.1	87.2*	2:42	11
	1.1	18.1	0.0	0.0	80.8	1:29	15
	2.8	15.4	0.6	0.0	81.2	2:02	15
	5.0	14.2	0.0	0.0	80.8	2:58	15
	Av. 1.7	17.2	0.4	0.1	80.6		
16	4.0	11.6	12.4	0.8	71.2	1:06	9
	1.2	16.5	0.8	1.0	80.5	1:30	9
	6.0	9.1	2.3	0.0	82.6	1:49	11
	6.9	7.7	0.0	0.2	85.2	2:40	11
	2.9	15.9	0.0	0.0	81.2	1:28	15
	3.0	13.5	0.5	0.1	82.9	2:00	15
	5.2	13.5	0.0	0.0	81.3	2:56	15
	Av. 4.2	12.5	2.3	0.3	80.7		
20	3.8	16.3	0.0	0.0	79.9	1:07	9
	1.1	18.0	0.0	0.0	80.9	1:27	9
	5.6	13.4	1.4	0.2	79.4	1:47	11
	7.9	12.2	4.1	0.2	75.6	2:38	11
	5.0	14.3	0.0	0.0	80.7	1:27	15
	4.2	4.3	1.5	0.8	89.2*	1:59	15
	6.2	13.7	0.0	0.0	80.1	2:55	15
	Av. 4.9	14.7	0.9	0.1	79.4		
24	11.0	0.8	13.5	1.1	66.4	1:40	8
	13.6	6.9	0.0	0.0	79.5	1:05	9
	1.6	12.7	0.0	0.0	85.7	1:25	9
	12.5	2.9	4.6	0.3	79.7	1:45	11
	5.3	13.2	0.5	0.2	80.8	2:46	11
	7.8	8.8	1.2	0.2	82.0	2:36	11
	4.0	15.1	0.0	0.0	80.9	1:24	15
	3.8	13.0	1.7	0.2	81.3	1:58	15
	9.0	8.5	0.0	0.0	82.5	2:52	15
	Av. 6.6	9.1	2.4	0.2	81.7		
28	14.8	4.6	0.0	0.0	80.6	1:03	9
	4.0	1.2	4.3	0.0	90.5*	1:23	9
	14.9	0.0	14.9	0.2	70.0	1:44	11
	15.4	1.3	1.3	0.0	82.0	2:35	11
	4.7	14.9	0.0	0.0	80.4	1:24	15
	12.8	0.5	82.6	0.3	3.8*	1:57	15
	5.4	14.2	0.0	0.0	80.4	2:51	15
	Av. 11.0	7.0	3.2	0.0	78.8		

* Not in average.

TABLE 13.—*Pennsylvania Furnace No. 2, Tuyere No. 6 (Continued)*

Distance from Nose of Tuyere, Inches	Analysis of Gas, Per Cent.					Time, P.M.	August, 1922
	CO_2	O_2	CO	H_2	N_2		
32	19.7	1.9	0.0	0.0	78.4	1:02	9
	5.5	4.7	11.8	0.5	77.5	1:22	9
	0.8	5.0	1.6	0.2	92.4*	2:57	8
	11.5	0.0	22.8	0.5	65.2	1:43	11
	4.2	10.7	3.5	0.5	81.1	2:45	11
	12.6	5.4	0.7	0.2	81.1	2:32	11
	11.3	1.7	16.6	0.7	69.7	1:20	15
	12.9	0.0	70.2	0.9	16.0*	1:54	15
	9.8	0.5	88.0	0.0	1.7*	1:55	15
	5.8	13.6	0.0	0.0	80.6	2:50	15
	Av. 10.1	5.4	7.9	0.3	76.3		
36	8.8	0.0	48.5	0.8	41.9	1:00	9
	6.6	0.0	14.6	1.1	77.7	1:20	9
	1.8	0.0	38.8	1.9	57.5	1:37	8
	10.2	7.4	5.4	0.2	76.8	2:55	8
	7.5	0.0	22.3	0.9	69.3	1:42	11
	6.2	0.0	22.4	0.8	70.6	1:40	11
	13.7	4.1	1.0	0.3	80.9	2:30	11
	7.5	0.0	59.3	1.2	32.0	1:16	15
	9.8	0.0	48.5	1.0	40.7	1:17	15
	5.8	0.0	80.5	0.8	12.9	1:18	15
	11.4	0.6	62.0	0.7	25.3	2:30	15
	14.5	0.0	8.7	0.6	76.2	2:48	15
	Av. 8.7	1.0	34.3	0.9	55.1		
40	1.3	0.0	46.0	1.1	51.6	1:35	8
	6.4*	12.6	1.5	0.1	79.4	2:53	8
	0.0*	9.9	24.0	0.7	65.4	1:38	11(?)
	6.4	0.1	69.2	0.9	23.4	2:24	15
	Av. 3.9	0.1	57.6	1.0	37.4		
44	1.1	1.4	62.0	0.8	34.7	1:33	8
	15.4*	0.3	6.5	0.4	77.4	2:50	8
	0.0	0.0	38.4	1.0	60.6	1:36	11
	2.2	0.1	70.1	1.2	26.4	2:27	15
	Av. 1.1	0.5	56.8	1.0	40.6		
48	1.1	0.0	77.4	0.9	20.6	1:31	8
	13.9*	2.3	3.0	0.6	80.2	2:49	8
	0.0*	7.7	23.2	0.0	69.1	1:33	
	0.0	0.5	56.5	0.9	42.1	1:34	11
	0.8	0.3	47.6	1.7	49.6	2:25	15
	1.0	0.0	51.2	1.4	46.4	2:26	15
	Av. 0.7	0.2	58.2	1.2	39.7		
54	0.0	0.0	76.0	1.0	23.0	1:30	8
	13.3*	1.0	7.3	1.0	77.4	2:47	8
	0.9	0.0	57.0	1.1	41.0	2:24	15
	Av. 0.5	0.0	66.5	1.1	31.9		
56	0.0*	3.8	30.4	0.8	65.0	1:30	11
60	0.0	0.0	69.5	1.5	29.0	1:29	8
	3.3	0.3	88.6	1.4	6.4	2:45	8
	0.0*	7.5	33.8	0.5	58.2	1:32	11(?)
	0.0	0.0	55.0	1.1	43.9	2:22	15
	Av. 1.1	0.1	71.0	1.3	26.5		
64	0.0	3.1	44.8	1.1	51.0	1:31	11
	0.0	0.0	50.5	1.4	48.1	1:30	11
	Av. 0.0	1.6	47.7	1.3	49.4		
66	0.0	0.0	64.2	1.1	34.7	1:15	8
	0.0	0.0	57.0	1.5	41.5	1:17	8
	0.0	0.0	66.0	1.0	33.0	1:28	8
	0.0	0.8	55.0	1.1	43.1	2:19	15
	0.0	0.1	46.0	1.6	52.3	2:21	15
	Av. 0.0	0.2	57.6	1.3	40.9		
72	0.0	0.5	66.4	2.0	31.1	1:28	8
	0.0	0.0	43.2	1.2	55.6	2:17	15
	0.1	0.1	55.5	1.1	43.2	2:18	15
	Av. 0.0	0.2	55.0	1.4	43.4		
78	0.3	0.6	66.0	1.3	31.8	1:25	8

TABLE 14.—*Pennsylvania Furnace No. 3, Tuyere No. 4*

Distance from Nose of Tuyere, Inches	Analysis of Gas, Per Cent.					Time	September, 1922
	CO_2	O_2	CO	H_2	N_2		
– 4	0.7	20.6	0.0	0.0	78.7	2:09 p.m.	18
	0.9	20.0	0.0	0.0	79.1	2:55	18
	Av. 0.8	20.3	0.0	0.0	78.9		
– 3	0.0	20.9	0.0	0.0	79.1	10:52 a.m.	16
	0.2	20.8	0.0	0.0	79.0	11:45	16
	Av. 0.1	20.9	0.0	0.0	79.0		
0	1.7	17.8	0.0	0.0	80.5	2:08 p.m.	18
	0.0	20.9	0.0	0.0	79.1	10:51 a.m.	16
	11.5	7.4	1.2	0.0	79.9		16
	8.2	10.4	0.0	0.0	81.4	1:56 p.m.	18
	Av. 5.4	14.1	0.3	0.0	80.2		
3	14.1	1.9	3.8	0.4	79.8	10:50 a.m.	16
4	5.5	12.6	1.4	0.0	80.5	12:07 p.m.	18
	2.0	15.9	0.6	0.0	81.5	1:57	18
	5.4	14.3	0.5	0.0	79.8		16
	6.8	2.1	4.4	0.2	86.5	1:55	18
	0.3	15.6	4.7	0.3	79.1	1:59	18
	Av. 5.7	10.4	2.6	0.2	81.1		
6	10.5	6.1	1.7	0.4	81.3	10:49 a.m.	16
8	7.1	0.3	9.3	0.8	82.5	1:53 p.m.	18
	3.6	11.0	1.4	0.4	83.6	2:54	18
	5.5	9.3	4.9	0.5	79.8	2:06	18
	Av. 6.7	6.7	4.3	0.5	81.8		
10	13.0	3.6	1.9	0.4	81.1	10:47 a.m.	16
12	6.3	6.0	1.6	0.5	85.6	2:53 p.m.	18
	11.0	8.5	1.5	0.0	79.0	2:05	18
	9.0	10.4	0.8	0.4	79.4	10:46 a.m.	16
	Av. 9.6	7.1	1.5	0.3	81.5		
14	11.0	9.5	0.8	0.0	78.7	2:04 p.m.	18
16	9.9	9.9	0.8	0.0	79.4		
	12.5	6.0	1.5	0.0	80.0	11:42 a.m.	16
	7.8	3.5	2.1	0.2	86.4	2:53 p.m.	18
	Av. 10.1	7.2	1.3	0.1	81.3		
18	10.7	6.8	1.4	0.5	80.6	10:46	16
20	7.4	5.5	3.0	0.2	83.9	2:51 p.m.	18
	10.2	0.0	15.1	0.6	74.1	1:53	18
	9.5	9.3	1.5	0.0	79.7	11:40 a.m.	16
	3.0	1.1	12.4	0.9	82.6		18
	Av. 8.2	4.5	6.7	0.4	80.2		
22	8.5	9.6	1.1	0.3	80.5	10:45 a.m.	16
24	11.0	4.4	4.4	0.1	80.1	2:58 p.m.	18
	14.9	0.3	23.8	0.9	60.1	1:52 a.m.	18
	7.9	0.5	20.0	0.6	71.0	11:39	16
	2.9	2.4	15.7	0.9	78.1	2:03 p.m.	18
	Av. 9.0	3.4	13.0	0.6	74.0		
26	10.0	5.9	1.6	0.3	82.2	10:44 a.m.	16
28	0.8	4.6	12.2	0.5	81.9	2:01 p.m.	18
	13.0	0.1	5.7	0.7	80.5	10:40 a.m.	16
	13.7	0.0	11.5	1.1	73.7	1:52 p.m.	18
	Av. 9.4	2.4	7.8	0.7	79.7		

* Not in average.

TABLE 14.—*Pennsylvania Furnace No. 3, Tuyere No. 4 (Continued)*

Distance from Nose of Tuyere, Inches	Analysis of Gas, Per Cent.					Time	September, 1922
	CO_2	O_2	CO	H_2	N_2		
32	14.8	0.8	9.1	0.1	75.2	11:38 a.m.	16
	15.7	0.6	4.2	0.9	78.6	1:51 p.m.	18
	3.9	5.0	7.7	0.6	82.8	2:00	18
	8.2	0.0	10.0	0.0	81.8		
	Av. 10.7	1.6	7.8	0.4	79.5		
34	13.8	0.7	6.4	0.7	78.4	10:43 a.m.	16
35	2.7	0.4	29.2	0.9	66.8	11:36	16
36	8.9	4.5	9.2	0.1	77.3	2:48 p.m.	18
	0.6	0.8	34.8	0.6	63.2	1:50	18
	1.5	1.1	34.2	0.6	62.6	2:47	18
	0.3	15.0	4.7	0.0	80.0		
	Av. 4.6	3.8	20.7	0.5	70.4		
38	0.8	0.5	32.5	1.4	64.8	10:42 a.m.	16
40	1.8	0.3	34.0	1.0	62.9	1:48 p.m.	18
	0.5	0.5	51.5	1.0	46.5		
	Av. 1.0	0.4	39.3	1.1	58.2		
42	0.3	2.8	31.7	4.2	61.0		18
44	1.3	1.5	55.5	1.0	40.7		16
	0.9	0.0	42.0	1.0	56.1	1:47 p.m.	18
	1.0	0.0	36.4	1.4	61.2		
	Av. 0.9	1.1	41.4	1.9	54.7		
46	0.0	0.3	36.2	0.8	62.7	10:40 a.m.	16
48	0.0	0.3	43.0	1.2	55.5	1:46 p.m.	18
49	0.0	1.0	36.2	1.5	61.3	10:37 a.m.	16
50	0.1	0.6	48.4	1.2	49.7		16
	0.0	3.0	34.0	0.8	62.2	2:45 p.m.	18
	Av. 0.0	1.0	39.6	1.1	58.3		
52	0.0	0.0	42.2	0.8	57.0		18
56	0.0	4.5	32.6	0.2	62.7	3:42 p.m.	18
	0.0	0.8	42.3	1.1	55.8		16
	0.3	0.6	42.5	1.1	55.5	1:45 p.m.	18
	0.3	3.5	33.2	0.2	62.8	3:44	18
	0.2	0.0	44.6	0.5	54.7		
	Av. 0.1	1.6	39.6	0.7	58.0		
58	0.5	0.5	51.5	1.0	46.5	10:34 a.m.	16
62	0.3	0.8	43.3	0.9	54.7	11:24	16
64	0.5	0.0	50.0	1.3	48.2	10:33	16
	Av. 0.4	0.4	48.3	1.1	49.8		
68	0.3	0.3	48.8	0.1	50.5	11:32 a.m.	16
	0.0	0.0	45.2	1.6	53.2		
70	0.0	0.8	40.0	1.3	57.9	10:32 a.m.	16
	Av. 0.1	0.4	44.7	1.0	53.8		
82	0.0	0.0	43.9	0.9	55.2	10:30 a.m.	16
88	0.3	0.3	42.6	1.0	55.8	10:28	16
	0.0	0.2	46.0	0.3	53.5	10:27	16
	Av. 0.2	0.3	44.3	0.7			

TABLE 15.—*Pennsylvania Furnace No. 4*

Distance from Nose of Tuyere, Inches	Analysis of Gas, Per Cent.					Time	Date September, 1922	Tuyere No.
	CO_2	O_2	CO	H_2	N_2			
0	2.1	19.8	0.0	0.0	78.1	11:20 a.m.	15	3
	4.5	15.1	0.0	0.0	80.4	11:38	15	3
	Av. 3.3	17.5	0.0	0.0	79.2			
2	9.6	6.4	1.6	0.0	82.4	5:02 p.m.	19	6
4	0.0	19.1	0.0	0.0	80.9	11:38 a.m.	15	3
	4.3	12.8	0.3	0.0	82.6	11:19	15	3
	Av. 4.6	12.8	0.6	0.0	82.0			
8	0.0	20.8	0.0	0.0	79.2	11:36 a.m.	15	6
	4.1	11.1	0.6	0.0	84.2	11:18	15	6
	4.5	7.2	2.6	0.7	85.0	11:18	15	6
	Av. 2.9	13.0	1.1	0.2	82.8			
10	1.6	18.0	0.5	0.0	79.9	5:03 p.m.	19	3
	6.1	11.4	0.5	0.0	82.0		19	3
12	11.4	6.2	2.1	1.0	79.3	11:17 a.m.	15	6
	0.8	19.5	0.0	0.0	79.7	11:35	15	6
13	6.7	8.8	0.5	0.0	84.0	4:59 p.m.	19	3
14	10.5	1.6	3.4	0.1	84.4	4:09	19	3
	Av. 6.2	10.9	1.2	0.2	81.5			
16	6.6	11.2	0.3	0.0	81.9	11:16 a.m.	15	6
	11.8	2.8	5.7	0.0	79.7	11:34	15	6
	12.6	2.1	3.6	0.3	81.4	11:16	15	6
	Av. 10.3	5.4	3.2	0.1	81.0			
18	13.6	11.3	2.3	0.0	72.8	4:58 p.m.	19	3
	11.2	1.8	1.5	0.2	85.3	4:08	19	3
20	7.5	0.6	10.9	1.0	80.0	11:34 a.m.	15	6
	Av. 10.8	4.6	4.9	0.4	79.3			
22	8.5	8.5	0.7	0.2	82.1	4:56 p.m.	19	3
	11.5	1.3	1.2	0.2	85.8	4:07	19	3
24	2.2	0.0	38.8	1.8	57.2	11:33 a.m.	15	6
	3.4	2.8	23.4	0.9	69.5	11:14	15	6
26	9.2	9.3	4.8	0.0	76.7	4:55 p.m.	19	3
	0.3	1.6	33.7	1.0	63.4	4:05	19	3
	Av. 5.7	4.1	17.1	0.7	72.4			
28	1.1	0.8	61.8	1.2	35.1	11:32 a.m.	15	6
	3.1	0.8	24.2	1.6	70.3	11:14	15	6
	3.3	0.0	26.8	1.6	68.3			
	Av. 2.5	0.5	37.6	1.5	57.9			
30*	14.9	1.3	74.0	0.4	9.4	4:34 p.m.	19	3
32	1.0	2.1	85.6	0.3	11.0	11:31 a.m.	15	6
	3.0	0.5	36.0	1.4	59.1			
	Av. 2.0	1.3	60.8	0.9	35.0			
34	0.1	0.0	44.4	1.2	54.3	4:31 p.m.	19	6
	12.1	1.4	63.6	0.6	22.3	4:52	19	6
36	2.1	2.6	78.2	0.5	16.6	11:30 a.m.	15	3
	3.9	3.4	31.6	0.4	60.7	11:17	15	3
	Av. 4.6	1.9	54.5	0.7	38.4			
38	0.8	0.4	34.8	1.6	62.4	4:03 p.m.	19	6
40	3.4	3.6	70.9	0.5	21.6	11:28 a.m.	15	3
	7.0	1.3	29.4	1.1	61.2	11:11	15	3
	3.1	3.4	26.8	0.6	66.1	11:10	15	3
	Av. 3.6	2.2	40.5	1.0	52.7			
44	0.0	0.3	39.7	1.4	58.6	11:08	15	3
	0.0	0.0	41.6	1.4	57.0	11:09	15	3
	2.6	2.6	74.4	0.5	19.9	11:27	15	3
	Av. 0.9	1.0	51.9	1.1	45.1			
46	0.0	0.3	52.0	0.7	47.0	4:50 p.m.	19	6
49	0.1	1.9	52.0	0.9	45.1	5:49	19	6
	0.3	1.0	34.6	1.4	62.7	4:02	19	6
50	2.4	2.9	36.2	1.0	57.5	11:26 a.m.	15	3
	0.3	3.9	40.0	0.6	55.2	11:07	15	3
	Av. 0.6	2.0	43.0	0.9	53.5			
56	0.5	1.2	45.2	1.1	52.0	11:06 a.m.	15	3
58	0.0	0.2	93.2	0.9	5.7	5:18 p.m.	19	6
61	0.0	0.8	47.5	1.2	50.5	11:06 a.m.	15	3
68	0.0	0.3	42.6	1.6	55.5	11:05	15	3
74	0.0	0.9	39.3	1.4	58.4	11:03	15	3
	0.0	3.1	40.0	1.0	55.9	11:06	15	3
	Av. 0.0	2.0	39.7	1.2	57.1			
78	0.1	2.1	37.2	1.3	59.3	11:02	15	3
82	0.0	0.0	41.4	1.3	57.3	11:00		
	0.0	2.0	37.3	1.1	59.6	11:01	15	3
	Av. 0.0	1.0	39.4	1.2	58.4			

* Not in average.

higher with furnaces using a large amount of coke per ton of metal than with furnaces operating on low consumption.

In Table 17, certain points on the curves of hearth gas composition have been set down for comparison. The column "velocity of blast" gives the calculated wind velocity as the blast leaves the tuyere. In making the calculations, it has been necessary to assume that pressure and temperature are those recorded as blast pressure and temperature in the operating data in Table 3.

The writers have been unable to find a relation between the character of the coke employed at the various furnaces and the extent of the combustion zone. For example, comparison of data for the furnace operating on Illinois coke and the two Pennsylvania furnaces operating on Connellsville beehive coke does not show the difference that might be expected. The Illinois coke is light, porous, and highly combustible, in the generally accepted sense of the term, while the Connellsville coke is dense and apparently less combustible; but under blast-furnace conditions, Connellsville coke is as combustible as the Illinois coke, for in the furnaces operating on Connellsville coke, oxygen has diminished to 2 per cent. at 23 and 30 in. from the nose of the tuyeres, respectively; while in the Illinois furnace the distance is 28 in. Carbon dioxide has diminished to 2 per cent. at 30 and 40 in. for the Connellsville coke, and 40 in. for the Illinois coke.

The combustion zone of Pennsylvania furnace 2 is somewhat different from that of other furnaces, but it is difficult to know whether to attribute this difference to the character of the coke or to a peculiarity of the furnace operation. No carbon dioxide is in evidence at the nose of the tuyeres and oxygen has not diminished to 2 per cent. until 36 in. from the tuyere. The furnace was using a byproduct coke very similar to that employed at Pennsylvania 4. The operation at Pennsylvania 2 differs from that at Pennsylvania 4 in that the coke consumption is considerably lower and the blast velocity higher.

If the four Pennsylvania furnaces and the Illinois furnace are compared without reference to the southern furnaces, the rate of disappearance of oxygen would seem to be proportional to the velocity of the blast at the tuyeres. Since it has been shown by Kreisinger that the rate of disappearances of oxygen in a fuel bed varies little with the velocity of the blast, this phenomenon might be explained as caused by the mechanical action of the blast in keeping coke away from the tuyeres or in making a less concentrated fuel bed near the tuyere—with increasing velocity of the blast.

It was evident, during the experimental work at Pennsylvania 2, that a very open condition existed in the vicinity of the tuyeres, because it was necessary to hold the sampling tube in place to prevent its being blown out by the blast pressure, when the end of the tube was 30 in. or

TABLE 16.—*Illinois Furnace*

Distance from Nose of Tuyere, Inches	Analysis of Gas, Per Cent.					Time	October, 1922	Tuyere
	CO_2	O_2	CO	H_2	N_2			
− 4	0.0	20.9	0.0	0.0	79.1	4:00	4	4
	0.6	20.3	0.0	0.0	79.1	4:04	5	7
	0.1	20.8	0.0	0.0	79.1	3:25	5	7
	0.0	20.0	0.0	0.0	79.1	3:37	4	4
	Av. 0.2	20.7	0.0	0.0	79.1			
0	0.8	20.1	0.0	0.0	79.1	3:43	4	4
	1.2	19.6	0.0	0.0	79.2	4:03	5	7
	7.3	10.5	0.8	0.0	81.4	3:25	5	7
	3.1	16.9	0.0	0.0	80.0	3:35	4	4
	3.1	16.9	0.0	0.0	80.0	11:12	5	4
	1.8	16.7	0.0	0.0	81.5	11:28	5	4
	Av. 2.9	16.8	0.1	0.0	80.2			
4	3.5	16.7	0.0	0.0	79.8	4:06	4	4
	0.4	20.4	0.0	0.0	79.2	4:05	4	4
	1.3	19.4	0.0	0.0	79.3	3:43	4	4
	1.7	19.2	0.0	0.0	79.1	3:35	4	4
	0.6	20.3	0.0	0.0	79.1	11:10	5	4
	4.2	6.7	3.2	0.7	85.2	11:28	5	4
	9.8	0.4	0.6	0.8	88.4	4:02	5	7
	5.6	13.3	0.0	0.0	81.1	11:10	5	4
	4.0	13.9	0.0	0.0	82.1	3:23	5	7
	Av. 3.5	14.5	0.4	0.2	81.4			
8	0.7	20.3	0.0	0.0	79.0	4:05	4	4
	2.7	16.1	2.8	0.3	78.1	3:42	4	4
	11.3	0.7	1.2	0.0	86.8	4:01	5	7
	3.8	14.2	0.8	0.0	81.2	3:34	4	4
	0.7	0.3	9.8	0.7	88.5	11:27	5	4
	7.1	5.4	2.3	0.0	85.2	p.m.	5	7
	Av. 4.4	9.5	2.8	0.2	83.1			
12	1.9	16.2	0.0	0.0	81.9	4:04	4	4
	1.9	13.7	4.7	0.0	79.7	3:41	4	4
	9.3	10.3	0.0	0.0	80.4	4:01	5	7
	2.5	16.4	1.4	0.0	79.7	3:34	4	4
	6.9	8.5	0.0	0.0	84.6	11:08	5	4
	7.6	0.6	17.4	0.9	73.5	11:27	5	4
	Av. 5.0	11.0	3.9	0.2	79.9			
16	3.1	12.3	0.4	0.0	84.2	4:04	4	4
	5.1	9.1	6.3	0.5	79.0	3:41	4	4
	4.4	13.0	9.9	0.0	72.7	3:31	4	4
	6.1	4.0	3.2	0.0	86.7	3:14	4	4
	7.3	0.0	13.7	0.9	78.1	11:20	5	4
	8.1	1.1	4.6	0.5	85.7	11:08	5	4
	10.1	0.3	11.4	0.6	77.6	3:21	5	7
	8.8	12.1	0.0	0.0	79.1	4:00	5	7
	Av. 6.6	6.5	6.2	0.3	80.4			
20	1.6	13.3	0.5	0.0	84.6	4:03	4	4
	7.5	6.0	4.8	0.0	81.7	3:40	4	4
	12.7	0.3	5.3	0.4	81.3	11:25	5	4
	9.4	0.0	8.0	0.7	81.9	11:07	5	4
	5.0	0.8	21.2	1.1	71.9	3:46	5	7
	2.7[*]	1.1	66.5	0.2	29.5	3:31	4	4
	6.9	4.0	2.7	0.0	86.4	3:13	4	4
	6.5	0.3	37.8	0.7	54.7	4:20	5	7
	14.9	0.0	7.4	0.1	77.6	3:55	5	7
	Av. 8.1	3.1	11.0	0.4	77.4			
24	3.3	12.5	1.3	0.0	82.9	4:02	4	4
	6.9	0.4	19.3	0.2	73.2	3:39	4	4
	12.3	2.0	0.9	0.0	84.8	11:25	5	4
	13.4	1.0	9.2	0.4	76.0	3:55	5	7
	8.3	1.7	25.0	0.6	64.4	3:30	4	4
	9.0	0.8	7.2	2.7	80.3	3:12	4	4
	12.7	2.4	2.3	0.4	82.2	11:06	5	4
	9.6	0.7	9.6	0.8	79.3	3:45	5	7
	Av. 9.4	2.7	9.4	0.6	77.9			

[*] Not in average.

TABLE 16.—*Illinois Furnace* (*Continued*)

Distance from Nose of Tuyere, Inches	Analysis of Gas, Per Cent.					Time	October, 1922	Tuyere
	CO_2	O_2	CO	H_2	N_2			
28	5.5	7.5	0.1	0.0	86.9	4:01	4	4
	2.3*	0.1	71.2	0.0	26.4	3:39	4	4
	14.9	2.4	0.0	0.0	82.7	11:05	5	4
	4.2	0.3	27.8	1.1	66.6	3:43	5	7
	13.8*	0.0	72.4	0.2	13.6	3:29	4	4
	10.0	7.4	3.4	0.9	78.3	3:10	4	4
	0.7	0.0	30.3	0.5	68.5	11:02	5	4
	5.7	9.4	5.2	0.9	78.8	11:20	5	4
	13.4	1.6	0.5	0.0	84.5	4:19	5	7
	10.5	0.0	15.6	0.8	73.1	4:13	5	7
	11.6	0.3	1.2	0.5	86.4	3:34	5	7
	8.6	0.8	20.4	0.9	69.3	3:53	5	7
	Av. 8.5	3.0	10.5	0.6	77.4			
32	14.0	0.0	6.2	0.0	79.8	3:58	4	4
	3.2	1.1	33.8	1.0	60.9	11:31	5	4
	7.0	0.5	27.3	0.8	64.4	10:56	5	4
	7.8	0.0	22.8	0.8	68.6	3:32	5	7
	11.7	0.3	53.8	0.3	33.9	3:11	4	4
	9.4	1.2	15.9	1.4	72.1	11:01	5	4
	2.2	0.0	35.3	0.8	61.7	4:12	5	7
	6.1	0.0	27.5	0.9	65.5	3:43	5	7
	Av. 7.7	0.4	27.8	0.8	63.3			
36	3.2	0.3	72.0	0.7	23.8	3:57	4	4
	5.7	0.5	26.0	0.1	67.7	10:55	5	4
	1.4	0.3	34.7	1.3	62.3	3:33	5	7
	Av. 3.4	0.4	44.2	0.7	51.3			
40	3.8	0.4	65.9	0.3	29.6	3:55	4	4
	0.5	0.0	36.6	1.2	61.7	3:32	5	7
	1.0	0.3	52.8	0.4	45.5	3:07	4	4
	3.1	0.8	48.8	0.3	47.0	3:07	4	4
	1.1	0.3	51.0	0.6	47.0	10:55	5	4
	Av. 1.9	0.4	51.0	0.6	46.1			
44	4.3	0.3	45.2	0.3	49.9	3:44	4	4
	0.3	0.9	36.3	1.2	61.3	3:31	5	7
	1.5	0.5	54.2	0.2	43.6	3:06	4	4
	0.1	0.3	54.7	0.3	44.6	10:53	5	4
	0.2	0.5	35.6	1.0	62.7	3:31	5	7
	Av. 1.3	0.5	45.2	0.6	52.4			
48	0.3	0.6	69.2	0.4	29.5	3:43	4	4
	0.8	0.3	62.0	0.4	36.5	3:42	4	4
	1.2	0.3	57.5	1.0	40.0	10:52	5	4
	0.8	0.3	63.7	0.6	34.6	3:05	4	4
	0.0	0.3	50.9	0.8	48.0	3:04	4	4
	2.0	0.3	42.0	0.9	54.8	10:52	5	4
	0.0	0.0	35.8	1.2	63.0	3:30	5	7
	Av. 0.7	0.3	54.4	0.8	43.8			
54	0.8	0.0	48.3	1.2	49.7	10:47	5	4
	0.3	0.3	47.3	0.0	52.1	3:03	4	4
	0.3	0.3	38.0	1.4	60.0	3:30	5	7
	Av. 0.5	0.2	44.5	0.9	53.9			
60	0.0	0.0	46.4	0.9	52.7	10:46	4	4
	0.0	0.0	40.0	1.3	58.7	3:29	5	7
	0.0	0.0	44.4	0.6	55.0	3:01	4	4
	0.3	0.0	42.7	0.7	56.3	3:02	4	4
	0.3	0.3	35.4	1.3	62.7	3:28	5	7
	Av. 0.1	0.1	41.8	1.0	57.0			
66	0.4	0.0	52.4	1.1	46.1	10:45	5	4
	0.3	0.0	40.2	0.9	58.6	3:28	5	7
	0.3	0.0	43.8	0.4	55.5	3:00	4	4
	0.3	0.0	45.2	0.2	54.3	2:59	4	4
	0.0	0.3	36.2	0.9	62.6	p.m.	5	7
	Av. 0.3	0.1	43.6	0.7	55.3			
70	0.3	0.0	54.7	1.0	44.0	10:42	5	4
	0.0	0.3	52.8	0.5	46.4	10:43	5	4
	Av. 0.2	0.2	53.8	0.8	45.0			

TABLE 17.—*Summary of Combustion-zone Data*

Furnace	Coke			Iron Made per Day, Tons	Velocity of Blast at Tuyeres Ft. per Sec.	Combustion-zone Data						
						Oxygen				Carbon Dioxide	Carbon Monoxide,	
	Operation	Lb. per Ton of Iron	Porosity, Per Cent.			Nose of Tuyere, Per Cent.	4 in. from Tuyere, Per Cent.	10 in. from Tuyere, Per Cent.	2 Per Cent. at Distance from Tuyere, Inches	2 Per Cent. at Distance from Tuyere, Inches	2 Per Cent. Distance from Tuyere, Inches	Maximum, Per Cent.
Alabama 1	Semet-Solvay	3160	51	258	340			14.0	28	48	24	36
2	Semet-Solvay	2800	49	260	345	15.0	15.0	10.0	22	32[a]	12	61
3	Koppers	1900	44	418	290			8.0	24	36	17	56
4	Beehive	2645	46	355	365	8.0	8.5	8.5	24	43	22	44
5	Beehive	2765	46	356	340	12.5	12.0	11.0	24	30	20	43
6	Koppers	2750	44	476	505				(26)	35		52
Penna. 1	Beehive	2800	46	220	235	8.0	8.5	9.0	23	30	12	56
2	Koppers	1700	51	575	380	20.9	19.6	17.0	36	42	22	63
3	Beehive	2465	48	449	290	14.0	10.5	8.0	30	40	6	45
4	Koppers	2300	48	514	305	17.5	15.0	11.0	28	36	14	55
Illinois	Roberts	1770	53	515	280	17.0	14.5	10.6	28	40	6	49
Average (from Fig. 12)						13.0	12.0	10.0	26	38	14	55

[a] 3.5 per cent.

less from the nose of the tuyere. This was also the case, to a smaller extent, with the other northern furnaces, but at the southern furnaces and Pennsylvania 1 (operating on spiegeleisen) it was unnecessary to hold the tube in place, perhaps because the fuel was packed more densely around the pipe. Evidently this phenomenon is dependent also on the ratio of ore to coke.

It is evident from the data that, at distances greater than 28 to 40 in. from the tuyeres, the gas samples were taken from regions through which only a part of the blast gases was passing. This is evidenced by the increasing excess ratio of oxygen to nitrogen and the enormously rapid rise of carbon monoxide. An indication of the extent of the zone in which coke is being rapidly consumed is obtained while driving the sampling tube into the furnace. In the average southern furnace and in Pennsylvania 1, the pipe could be pushed in easily for 24 in. from the tuyeres. At Pennsylvania 2, it was possible to push in the pipe to 32 to 36 in., and at Pennsylvania 3 and 4 and Illinois about 32 in. At the point at which the pipe must be driven in by the blows of a dolly, it is evidently in contact with a fairly solid bed of coke, which is not being consumed rapidly.

The data of Van Vloten (Table 2) taken in a plane 2 ft. above the tuyeres show the highest percentage of CO_2 and lowest percentage of CO at a horizontal distance of 12 in. from a point on this plane directly above the nose of the tuyere. This would indicate that the blast began to pass upward rapidly at this point. The percentage of CO on this plane is considerably higher at points nearer the hearth wall. Van Vloten's data do not indicate much greater penetration of CO_2 in a vertical direction from the tuyeres than in a horizontal direction. His data taken at the tuyere plane agree closely with the results obtained by the writers.

It seems desirable to secure further data above the tuyere plane, as our present samples at distances greater than 30 to 36 in. from the tuyeres were probably taken in regions through which the main stream of the blast gases was not passing. Arrangements have been made to carry out this investigation at a blast furnace near Pittsburgh.

SUMMARY

Combustibility of coke, from the standpoint of its use as blast-furnace fuel, is proportional to the mean rate of gasification per unit volume of the combustion zone. The rate of gasification, other factors being constant, is inversely proportional to the volume of the zone around the tuyeres in which oxygen or carbon dioxide are present. At the wind velocities and temperatures prevailing in the combustion zone of the blast furnace, it is doubtful whether ordinary metallurgical cokes vary sufficiently in apparent density, porosity, or character of carbonaceous material to influence to any great degree the extent of the combustion zone.

Analyses of samples of gas taken in the tuyere plane at various points in the hearth at ten blast furnaces show that oxygen has diminished to practically zero at 24 to 30 in. from the tuyeres and carbon dioxide has practically disappeared at 32 to 40 in. The gas analyses indicate that direct reduction of metallic oxides is taking place in the central regions of the hearth. Some difference in the character of the combustion zone at the several furnaces is evident but no relation is apparent between these differences and the physical properties of the coke. It seems probable that the large differences observed in blast-furnace operation with different cokes are caused by other factors than the relative combustibility of the cokes in the tuyere zone.

Acknowledgments

This investigation was made under the supervision of A. C. Fieldner, supervising fuels chemist. The writers acknowledge their indebtedness to P. H. Royster, assistant metallurgist, Minneapolis station, who suggested the problem and gave helpful advice and criticism during the course of the work, and to the superintendents and other officials at the blast furnaces, whose coöperation made the investigation possible.

DISCUSSION

(*Includes also discussion of paper of Sherman and Blizard*, pp 526–542.)

A. L. Feild, New York, N. Y.—The word combustibility has several meanings; sometimes, it is vaguely synonymous with ignition temperature. A material is combustible when it catches on fire on reaching a certain definite temperature; the lower the temperature the more combustible is the material. But in the case of the blast furnace, combustibility refers to the amount of coke burned in unit time per cubic foot; *i.e.*, it is a volume relation.

Some years ago, I had occasion to measure the rate of oxidation of carbon at various temperatures, in grams per square centimeter per minute. It has occurred to me that this figure, while it cannot be directly applicable to blast-furnace coke, might serve as an index from which approximate calculations could be made. The value obtained in these experiments was 0.012 gm. per sq. cm. per min. Carbon rods were heated in air by the passage of an electric current, temperatures being measured optically. By measuring the change in resistance of the rod as it oxidized, the weight loss could be computed.

Strange to say, the rate of oxidation was independent of temperature over the range 800° and 1200° C.; that is, within our experimental error, the carbon rod burned as rapidly at 800° as it did at 1200°. Measurements at 900°, 1000°, and 1100° were also taken. The curves showing the change in resistance coincided as well when the experiments were run at different temperatures as when they were run at the same temperature. Why this reaction should be independent of temperature over such a considerable range I do not know. Evidently combustion is largely determined by some diffusion phenomenon. Another point brought out was that the rod did not oxidize below 650° C.; that is, the weight loss below that temperature was very small, being probably restricted to certain volatile constituents. As soon as this temperature was exceeded, there was a sudden increase in temperature up to about 725°. It was impossible to keep the rod anywhere between these two temperatures no matter how the current was adjusted. Combustion in the blast furnace, of course, takes place at much higher temperatures than 1200° C.

Philip H. Royster, Minneapolis, Minn.—The authors seem to have proved that all cokes made in any kind of oven from any kind of coal heated at almost any temperature will burn at the same rate of speed at the tuyeres. They state that the kind of coke has no effect on the rate at which the coke burns. Sherman and Blizard show, by varying the rate of speed, that the combustibility does not change; Mr. Feild points out that the density of the coke makes no difference. Now, Sherman and Blizard get about 3 or 4 in. for the oxygen going into the furnace and Perrott and Kinney get 30. The reason for this discrepancy is the size of the lumps of coke. Sherman and Blizard mentioned coke which had been coked for 29 hr.; this requires time, gas, and labor to produce something which seems to act in a blast furnace just as material coked in 9 hr. or possibly 6 hr. or even less.

In the Minneapolis experiment station of the Bureau of Mines, we have been running experimental furnaces varying in size from 10 in. high with 1 in. hearth, up to 21 ft. high, and it is obvious that if we had to have a combustion zone of 38 in., as Perrott and Kinney mention, the combustion would not take place inside of our furnace. We, therefore, crushed the coke to the proper size for that size of furnace.

In the experimental furnaces, sample tubes are pushed in the side of the furnace from the bottom to the top to get samples for the gas analysis; we can also get the temperature. We propose to draw the stock from the furnace at various heights for analysis.

The combustion of coke seems to be limited to a small zone at the nose of the tuyeres. In the experimental blast furnace with a 20-in. hearth, by crushing the coke, we reduced the zone to just what Sherman and Blizard got, about 4 in. of oxygen penetration. At times, with a 1-in. hearth, we had a penetration of 0.2 mm. I do not know how coke could move down the middle of the furnace if it is only removed from the side. The layers of coke and ore are put in supposedly level at the stock line by whatever device is used, but if you took it out even at the bottom I doubt if the layers would ascend level; certainly they will not if they are taken out by burning them at the tuyeres. So that a rapidly combustible coke, one which burned nearer the tuyeres, would upset the regularity of the descent of the stock. Fast combustibility does not produce fuel economy.

Forms of Sulfur in Coke, and Their Relations to Blast-furnace Reactions*

By Alfred R. Powell,† Ph. D., Pittsburgh, Pa.

(New York Meeting, February, 1923)

Sulfur has been one of the most troublesome elements encountered since the earliest days of iron smelting, and this problem will become of increasing importance as the higher sulfur coke is used, because of the rapidly diminishing supply of low-sulfur coals suitable for the manufacture of metallurgical coke.

The deleterious effect of sulfur in iron and steel is well known; therefore, except in special cases, it is desirable to make the iron produced by the blast furnace as low in sulfur as possible. Sulfur enters the furnace in all of the raw materials, but principally in the coke. It comes out of the blast furnace distributed between the iron and the slag; very little passes out of the furnace with the gases.

The mechanism of the reactions that cause the sulfur of the coke to pass into the slag and iron has not been studied to any great extent, although the character of the end products, and the conditions controlling the distribution of the sulfur between the iron and slag, are fairly well known. In the iron, the sulfur occurs as ferrous sulfide; or manganese sulfide in case of iron high in manganese. In the slag, the sulfur is present mostly as calcium sulfide. The presence of free sulfur[1] and sulfates in blast-furnace slags is to be accounted for on the basis of oxidation of the calcium sulfide after the slag is tapped.

The generally accepted reaction for the relation between the ferrous sulfide in the iron and in the coke, and the calcium sulfide in the slag, is that given by Howe:[2]

$$FeS + CaO + C \rightleftarrows Fe + CaS + CO$$

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* Published by permission of the Director, Bureau of Mines.

† Associate Chemist, Pittsburgh Experiment Station, Bureau of Mines.

[1] W. G. Imhoff: Sulfur as a Component of Furnace Slag. *Blast Fur. & St. Pl.* (July, 1917); also *Met. & Chem. Eng.* (1917) **17**, 189–90.

[2] H. M. Howe: "Iron, Steel and Other Alloys," 402, 2d ed., 1906.

This is a reversible reaction, and any given set of conditions will determine the relative concentration of FeS in the iron and of CaS in the slag. This reaction does not account for the various reactions the sulfur in the coke will undergo, only a small part of which is normally present as FeS. Also, it is unlikely that this equation expresses the real compounds entering into the reaction, for this system represents four phases —two liquid, one solid, and one gaseous phase.

In such a complex system, the true relationships should be expressed by a series of equations. Thus, the carbon would be in equilibrium with carbon monoxide in the gaseous phase; the carbon monoxide would be in equilibrium with the CaO in the slag; and this, in turn, would be in equilibrium with the complex silicates of the slag. The CO and CaO would have a tendency to form metallic calcium, which would form CaS with the sulfur; and the dissociation pressure of this CaS would balance the dissociation pressure of FeS in the iron, through the medium of a low but definite partial pressure of free sulfur in the gaseous phase.

The desulfurization equation given covers, in a general way, the conditions existing in the slagging zone, but does not explain sulfur reactions in the upper part of the furnace nor the reactions occurring during and after the combustion of the coke. The literature on the reactions of sulfur in the blast furnace always refers to the desulfurization of iron by the slag. Such statements intimate that the iron had taken up sulfur before reaching the slagging zone, and that the action in this zone of the furnace was a part removal of the sulfur from the iron. Feild and Royster[3] have shown that the viscosity of the slag has an important bearing on the desulfurization of the iron in the slagging zone. Many other factors are known to affect the removal of the sulfur from the iron, but apparently little attention has been paid to the mechanism and the factors that affect the entrance of the sulfur into the iron.

Many years ago it was recognized that iron in contact with coke absorbed sulfur very readily. Hilgenstock[4] obtained the following results on melting steel and leaving it in contact with coke for some time:

	Silicon, Per Cent.	Manganese, Per Cent.	Carbon, Per Cent.	Sulfur, Per Cent.
Original metal	0.38	0.45	0.62	0.04
After fusion	0.58	0.18	1.89	0.50

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[3] A. L. Feild and P. H. Royster: Slag Viscosity Tables for Blast-furnace Work. Bureau of Mines *Tech. Paper* 187 (1917); Temperature-viscosity Relations in the Ternary System CaO-Al_2O_3-SO_2. Bureau of Mines *Tech. Paper* 189 (1918)

[4] G. Hilgenstock: Elimination of Sulfur from Iron. *Stahl u. Eisen* (1893) **13**, 451–62; *Jnl.* Iron and Steel Inst. (1893) **44**, 435–7.

This shows that large quantities of sulfur may be absorbed when the iron and coke are in contact.

Experiments were conducted by Wuest and Wolff[5] to determine what action the ascending blast-furnace gases have on the sulfur in the coke before the coke reaches the combustion zone. Although this investigation was carried out on a laboratory scale, it indicates certain tendencies in the upper part of the blast furnace. It was found that carbon monoxide removed a fairly large part of the sulfur at 1000° C., and that the other constituents of the blast-furnace gas removed sulfur from the coke in varying proportions. In order to show how the sulfur removed from the coke acted on the iron ore and limestone of the charge, a mixture corresponding to blast-furnace gas was passed over pulverized coke held at 1000° C., and then over iron ore or limestone held at a high temperature.

From their experiments, Wuest and Wolff concluded that the sulfur of the coke does not reach the level of the tuyeres of the blast furnace without undergoing alteration; but a great portion of it is volatilized by the ascending gases and is then largely absorbed from the gases by the descending charge, in which condition it arrives in front of the tuyeres. They concluded that, up to 800° C., the sulfur is principally absorbed by the oxides of iron from the sulfur-laden gases; while from 800° upwards, the lime becomes the chief absorbent of the sulfur.

Simmersbach[6] disagrees with these investigators. He points out that the experiments were performed with pulverized coke, thereby giving a large surface exposure, whereas the lump coke used in the blast furnace presents an entirely different problem. He quotes several cases where analyses have been made of coke blown out of the tap hole of the furnace. He found in such a coke an average sulfur content of about 75 per cent. of the total sulfur; an investigation by M. Boecker shows that a coke with an average sulfur content of 0.77 per cent. showed 0.50 per cent. in the coke blown out of the tap hole, and in the center of the lump coke, 0.59 per cent. In these cases the coke had been exposed to combustion conditions for a short period, so that all of the loss in the sulfur could not be ascribed to the furnace gases.

As will be shown later, Wuest and Wolff were probably correct in stating that a transference of sulfur from the coke to the iron and lime took place in the upper regions of the furnace; but they were incorrect in assuming that this transference was almost complete by the time the tuyeres were reached. However, if only about 5 per cent. of the sulfur in the coke found its way into the iron, this would represent the average sulfur tolerance in the finished iron; any percentage above this would have to be decreased in the slagging zone of the furnace. For this reason, even a slight desulfurization of the coke in the upper portion of the furnace is important from the standpoint of blast-furnace reactions.

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[5] F. Wuest and P. Wolff: Sulfur in Coke and its Behavior in the Blast-furnace. *Jnl.* Iron and Steel Inst. (1905) **67**, 406–32.

[6] O. Simmersbach: "Grundlagen der Koks-Chemie," 182, Berlin, 1914.

Forms of Sulfur in Coke

In the general investigation that the Bureau of Mines has carried on for the past few years on the sulfur in coal and the reactions that the various sulfur forms in coal undergo during the coking process, it was realized that many metallurgical operations were affected by the sulfur present in coke. The manner in which the sulfur was combined in the coke had never been determined, though it was known that a portion was in the form of ferrous sulfide and that minute quantities of calcium and magnesium sulfides might be present. It was also known that small quantities might be in the form of metallic sulfates, resulting probably from oxidation of the sulfides during the cooling of the coke. But the major portion of the sulfur in the coke, often 90 per cent. of the total sulfur, was generally referred to under the designation of organic sulfur.

It was impossible to use the ordinary methods of chemical examination to determine the nature of this organic sulfur. Distillation and solvents were out of the question, because the coke sulfur did not respond to either treatment; the method adopted depended on the use of the phase rule. Keeping the temperature constant, the relations between the concentrations of sulfur in the coke and the pressures of free sulfur over the coke were determined. The extremely small pressures of free-sulfur vapor were determined by an indirect method, which is beyond the scope of this paper to describe.

These experiments[7] showed that before the coke was removed from the oven, and while still in a reducing atmosphere, the sulfur occurred in the coke in two characteristic forms; namely, the major portion as sulfur closely associated with the carbon, probably as a solid solution of sulfur in carbon, the remainder as ferrous sulfide, with a very small quantity of calcium or magnesium sulfide. But when the coke is quenched, oxidation of these metallic sulfides takes place, so that the final cold coke has lost perhaps half of its ferrous-sulfide content, the oxidation products being mainly free sulfur, which is retained by the coke as adsorbed free sulfur, and a small quantity of sulfate. This oxidation of the ferrous sulfide takes place with great rapidity over a certain temperature range and, despite the rapid cooling of the coke when quenched, air enters the pores of the coke in sufficient quantity to produce appreciable oxidation effects. If the coke is cooled in a strictly reducing atmosphere, only metallic sulfides, and the solution of sulfur in the carbon are found in the cold coke. Furthermore, if coke subjected to oxidation, as is the case with all commercial cokes, is reheated in a reducing atmosphere, reversion of the sulfur to the two original forms takes place. This latter fact is of importance in considering the sulfur reactions in the blast furnace, for in that case we have a reheating of the coke in a strongly reducing atmosphere.

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[7] A. R. Powell: Forms of Sulfur in Coke. A Physico-chemical Study of the Sulfur Held by Carbon at High Temperatures. *Jnl.* Am. Chem. Soc. (1923) **45**, 1–15.

Summing up, then, hot coke in a reducing atmosphere or coke cooled in a reducing atmosphere contains sulfur dissolved in carbon, in the form of a solid solution, and ferrous sulfide, with perhaps small quantities of calcium or magnesium sulfides. Coke cooled in the ordinary manner in the presence of air contains sulfur in these two forms, also adsorbed free sulfur and sulfates.

To illustrate these sulfur forms, we may consider a coke made from Illinois coal in a standard type of oven. The distribution of sulfur in this coke is typical of a great many that have been studied, and gives some idea of the relative quantities of the various forms. The first column of the analysis gives the sulfur distribution in the original coke, just as it was received from the plant, while the second column gives the sulfur distribution in the same coke after a portion has been heated and cooled in an atmosphere of hydrogen:

	ORIGINAL COKE, PER CENT.	COKE AFTER REDUCTION, PER CENT.
Solution of sulfur in carbon	0.84	0.84
Ferrous sulfide	0.26	0.47
Adsorbed free sulfur	0.19	0.00
Sulfates	0.02	0.00
Total sulfur in coke	1.31	1.31

REACTIONS OF SULFUR FORMS OF COKE IN BLAST FURNACE

At present, this topic must be discussed largely from theoretical considerations, because no actual study has been made of the coke, iron, and slag from different zones of the blast furnace. In passing down the furnace, the coke goes through the reduction, the iron sponge, the preliminary slag formation, and the combustion zones.

The Reduction Zone.—In this zone, the gases are reducing, and the temperature varies from below a red heat to 800° C., so the adsorbed free sulfur present in the original coke will be reconverted into ferrous sulfide. This reduction of the free sulfur is important, for the adsorbed sulfur gives a higher partial pressure of free sulfur in the vapor phase than ferrous sulfide. The experiments previously described showed that ferrous sulfide produced a partial pressure of about 10^{-8} atmospheres of free sulfur in the vapor phase at 800° C., while adsorbed free sulfur at the same temperature gave much higher pressures. As the partial pressure of free-sulfur vapor over the coke measures the tendency of the sulfur to pass out of the coke, this reduction causes the coke to retain its sulfur much more tenaceously than it would if no reduction occurred.

The sulfates are a minor factor in the sulfur of the coke, but probably there is, at least, a partial reduction of them in the reduction zone. However, the change is not important, for the sulfates are stable and would be retained in the coke.

The solution of sulfur in carbon is one of the most stable forms of sulfur encountered in the investigation of sulfur in coke. At any given temperature, the partial pressure of sulfur vapor from this form is much lower than that of ferrous sulfide; furthermore, this form of sulfur does not change readily into any other form. Reduction does not change this solid solution of sulfur in carbon, and oxidation affects it only when the coke is oxidized.

In the reduction zone of the furnace, the removal of sulfur from the coke would be almost negligible. The free sulfur would be quickly reduced to ferrous sulfide and would have no chance to be volatilized and then absorbed by the iron oxides. The partial pressure of sulfur vapor from ferrous sulfide would be small at 800°, consequently, little sulfur could be transferred from the coke to the oxides of iron from this source; and any removal of the solid solution form of sulfur from the coke is out of the question.

Summing up, then, it may be said that the only sulfur reaction of any importance in the reduction zone is the reduction of the adsorbed free sulfur of the coke to ferrous sulfide. The coke reaches the bottom of the reduction zone with practically no loss of sulfur.

Iron-sponge Zone.—There is comparative inactivity in this zone, as far as the general blast-furnace reactions are concerned. The limestone loses most of its carbon dioxide, and spongy iron loses the traces of FeO that may remain, and the coke, in general, remains unchanged. However, this zone is probably very important from the standpoint of the sulfur reactions. The sulfur of the coke remains in the solid solutions and ferrous sulfide forms that resulted from the reactions in the reduction zone. The former, being very stable, remains unchanged and has no tendency to pass out of the coke; the ferrous sulfide is subjected to constantly increasing temperature, and the partial pressure of free-sulfur vapor increases at a rapid rate. From the work on the sulfur in coke, it was found that the partial pressure of sulfur vapor from ferrous sulfide (or, more properly, the decomposition pressure of ferrous sulfide) increased about 20 times for every 100° C. rise in temperature. In other words, at 900° C., the decomposition pressure is 20 times that at 800° C.; at 1000°, 400 times as much; at 1100°, 8000 times as much, and so on.

The sulfur from the ferrous-sulfide constituent of the coke will have an increasing tendency to pass out of the coke as the higher temperatures of the furnace are reached, and this desulfurization of coke will be assisted by the upgoing gases. However, this evolved sulfur will not remain long in the gaseous phase, for it will come into contact with two active absorbents in the form of the hot spongy iron and the lime. At first it might appear strange that the iron would absorb the sulfur to form ferrous sulfide, while at the same time the ferrous sulfide of the coke was decomposing; in other words, it might appear that the ferrous sulfide in the coke and that in the iron were in equilibrium and there would be no transfer of sulfur from one to the other. But the ferrous sulfide of the coke exists in a distinctive solid phase, while the ferrous sulfide formed in the iron occurs as a solid solution with the iron and is subject to the laws governing solutions. The dissociation pressure of the ferrous sulfide in the coke is, therefore, the true dissociation pressure of ferrous sulfide; while that in solution in the iron has a dissociation pressure which is only a small fraction of the true value. Equilibrium conditions, therefore, call for a transfer of the sulfur from the ferrous sulfide of the coke to the iron until the iron-iron sulfide solution phase of both the coke and the iron have the same percentage composition. If this equilibrium point were reached, practically all of the ferrous-sulfide sulfur from the coke would be transferred to the iron. This condition is never realized, for complete decomposition of the ferrous sulfide of the coke could not be effected in the short time the charge remains in this zone.

Another factor that complicates the reactions in this zone is the formation of lime. Lime absorbs sulfur readily, under high-temperature reducing conditions, to form calcium sulfide. Tests have shown that calcium sulfide has a decomposition pressure much lower than that of ferrous sulfide at any given temperature, and of about the same order as the low pressure exerted by the solid solution form of sulfur; therefore, in this zone, the iron and the lime will compete for the sulfur evolved. If equilibrium between the various constituents of the furnace charge were attained, we should find that the coke had lost practically all of its ferrous sulfide, and that the sulfur would be distributed between the lime and the iron in such proportions that the decomposition pressure of the ferrous sulfide in the iron would exactly equal the decomposition pressure of calcium sulfide. This relationship has not been determined, and is probably not of much practical importance, for it is improbable that equilibrium is attained. The main determining factor in the distribution of the sulfur between the iron and the lime is the relative intimacy of contact between these materials and the coke. As it is probable that both the spongy iron and the lime are in fairly intimate contact with the lumps of coke, there is not likely to be any preferential absorption of sulfur by either material, and sulfur will enter both the iron and the lime.

In this zone, the absorption of sulfur by the iron may be considerable, as shown by some laboratory experiments, in which the iron used consisted of short lengths of iron baling wire. A definite weight of this iron was buried in a small quantity of granular coke contained in a crucible and the crucible was covered, so that there would be no free access of air, and then heated to a bright red heat in a muffle furnace. Under these conditions, the atmosphere would quickly become reducing because of the formation of carbon monoxide and a smaller quantity of carbon dioxide. After cooling, the iron was removed and analyzed for sulfur.

The first coke used contained over 3 per cent. of total sulfur. One batch of this coke was previously oxidized at about 500° C.; it has been found that this treatment completely oxidizes the ferrous sulfide, so that the coke contains only solid-solution sulfur and adsorbed free sulfur, and a small quantity of sulfate. Another batch was reduced by hydrogen so that it would contain only solid-solution sulfur and ferrous sulfide. In this way the relative effect of the adsorbed free sulfur and ferrous sulfide could be studied. It was assumed (later tests showed this assumption to be correct) that the solid-solution form would be inert and would have no action on the iron. The results of this test were as follows:

	Oxidized Coke, Per Cent.	Reduced Coke, Per Cent.
Sulfur content of original iron	0.064	0.064
Sulfur content of iron after exposure to coke	0.219	0.215

These results show that a rather large amount of sulfur is taken up by the iron, when in contact with coke, and that the relative amounts of absorbed free sulfur and ferrous sulfide in the coke do not affect the sulfur adsorption. This fact is to be explained on the basis of the reducing atmosphere in the crucible and the consequent reduction of any free sulfur to ferrous sulfide. This confirms the statement that any free sulfur in the coke is quickly changed to ferrous sulfide in the reduction zone of the furnace, and that the sulfur which the iron takes from the coke comes from the ferrous sulfide.

In another set of experiments, an effort was made to maintain a slight oxidizing influence in the red-hot crucible containing the coke, by leaving off the lid of the crucible so that the coke would have free access to the air. In this case, the Illinois coke mentioned was used. It contained 1.31 per cent. of total sulfur and was, therefore, more like an ordinary metallurgical coke than the high-sulfur coke just used. To bring out the relative effect of the oxidizing atmosphere, a test in a covered crucible was run parallel with this and the following results were obtained:

	Crucible Covered, Reducing, Per Cent.	Crucible Uncovered, Oxidizing, Per Cent.
Sulfur content of original iron	0.064	0.064
Sulfur content of iron after exposure to coke	0.241	0.339

These results show that an oxidizing atmosphere tends to throw more sulfur into the iron, which was to be expected, for free sulfur with its comparatively high vapor pressure would be formed. Another fact brought out was the comparative amount of sulfur introduced into the iron by this coke and the higher sulfur coke used in the first experiments. About the same quantity of sulfur was given up to the iron by the lower sulfur coke as by the higher sulfur coke, even under reducing conditions. However, an equal absorption might be expected when it is considered that the pressure of free-sulfur vapor over the coke is independent of the amount of sulfur contained in the coke, for ferrous sulfide will exhibit the same dissociation pressure for any given temperature, regardless of the quantity of ferrous sulfide present. The factor that determines how much sulfur will be taken up by the iron in the sponge-iron zone is, therefore, not the amount of sulfur in the coke but the speed with which the ferrous sulfide formed at the surface of the iron diffuses into the interior of the iron. In this connection, it must be remembered that the spongy iron of the blast furnace offers a large surface for the absorption of the sulfur to take place.

The experiments just described offer some explanation of what the absorbed free sulfur and the ferrous sulfide forms of sulfur may be expected to do in the sponge-iron zone of the blast furnace. Under the same conditions, the solid-solution form of sulfur should be practically inert and contribute no sulfur to the iron. In order to determine this, the Illinois coke was reduced by hydrogen and then treated with boiling hydrochloric acid, to remove from the coke all the sulfur forms except the solid solution of sulfur in carbon. The following result was obtained, the coke and iron being kept under the usual reducing conditions:

	PER CENT
Sulfur content of original iron....................................	0.064
Sulfur content of iron after exposure to coke........................	0.083

A comparison of this result with the results obtained from the coke in which the ferrous sulfide was present shows a great difference in the amount of sulfur absorbed. In fact, it is doubtful whether the small amount of sulfur taken up from the coke came from the solid solution form; it is more probable that this came from a small amount of ferrous sulfide left in the coke after the acid treatment.

As lime is present in this zone, in addition to the spongy iron and the coke, tests were run in which lime was mixed with the coke in which the iron was buried. Under these conditions, the lime would be in more intimate contact with the coke than the iron. The results from several tests showed that practically no sulfur was taken up by the iron, all of the sulfur given off by the coke being absorbed by the lime; however, the relative intimacy of contact between the lime and the coke was greater than in the blast furnace, so it must not be assumed that the lime in the spongy-iron zone will protect the iron from contamination by sulfur. The following are the results from tests performed with Illinois coke (the same coke as used before), where the coke and iron were held at almost a white heat for some time:

	ORIGINAL COKE, PER CENT.	COKE + CaO, PER CENT.	COKE + MgO, PER CENT.	COKE + Na_2CO_3, PER CENT.
Sulfur content of original iron.....	0.064	0.064	0.064	0.064
Sulfur content of iron after exposure to coke................	0.462	0.076	0.098	0.098

The great difference between the amount of sulfur absorbed from the untreated coke and the small amount from the limed coke is apparent at once. It is to be noted that magnesia and soda are also very efficient in preventing the sulfur from going into the iron. In every case the corresponding metallic sulfide has been formed, all of which have dissociation pressures far below that of ferrous sulfide.

The sulfur reactions of the sponge-iron zone may be summed up in the general phenomenon of a transfer of the sulfur from the coke to the iron and the lime. The sulfur given up by the coke comes from the ferrous sulfide, probably none comes from the solid-solution form of sulfur. The presence of lime in intimate contact with the coke practically prevents the sulfur of the coke from reaching the iron.

Zone of Preliminary Slag Formation.—The third zone of the blast furnace of interest in the sulfur reactions is that of preliminary slag formation. In this region the lime begins to unite with the siliceous impurities of the iron, and the result is a viscous, limey slag, which will have a tendency to coat the iron and also the coke. When this occurs the transfer of the sulfur from the coke to the iron will be practically stopped, for the path over which this transfer took place in the sponge-iron zone will be blocked by this basic slag, which will have a great affinity for the sulfur. The zone of preliminary slag formation is, therefore, a region where the contamination of the iron by the sulfur of the coke ceases.

Combustion Zone.—This zone is the final region where the sulfur reactions occur. The coke is entirely consumed and the ash of the coke unites with the limey slag to form a fluid slag of the composition desired. The sulfur of the coke probably burns to sulfur dioxide, which is quickly reduced to free sulfur, hydrogen sulfide, or carbon oxysulfide. Carbon bisulfide might also result from this reduction. In any case, the hot lime and the iron, now in a liquid state, will compete for the sulfur. At this temperature, the sulfur will not remain long in the gases of combustion. It is probable that the sulfur of the coke released in the combustion zone will enter the slag in preference to the iron. This reasoning is based on the intimacy of the contact, for the drops of liquid iron are surrounded to some extent by the slag. Of more importance than this, however, is the fact[8] that the coke, just before combustion, becomes impregnated with the slag. This is an important point, as the major portion of the coke sulfur is released at this stage of the process, and the conditions would seem to be ideal for its complete absorption by the slag.

Despite the probability that no sulfur enters the iron in the combustion zone, the sulfur content of the iron has already become higher than is desirable in the finished product, from its contact with the coke in the spongy-iron zone. This means that the iron must be desulfurized by the liquid slag in the vicinity of the combustion zone of the furnace.

Sulfur Equilibrium between Iron and Slag

As the iron probably contains more sulfur when it enters the final combustion and slagging zone of the furnace than when it settles out at the bottom ready for tapping, there must be a transference of the sulfur from the iron to the slag. This probably never continues to the equilibrium point, because of the time factor, but it is desirable to bring the sulfur distribution to as near the equilibrium point as possible, so as to obtain a low-sulfur iron. This has been the reason for the advocacy of very liquid slags, so as to get good contact with the iron. The conditions which determine this distribution equilibrium value, such as temperature and the basicity of the slag, have been stated many times and do not require repetition here. The fundamental conditions for equilibrium between the sulfur of the iron and the slag is that the dissociation pressure of the calcium sulfide contained in the slag be equal to the dissociation pressure of the ferrous sulfide dissolved in the iron. The need for more data on the fundamental conditions that determine the sulfur equilibrium between the iron and slag cannot be too strongly emphasized.

Proposed Further Study of Sulfur Reactions in Blast Furnace as Related to Forms of Sulfur in Coke

This paper has been prepared with a rather thorough understanding of how the sulfur occurs in coke, and how these various sulfur forms found in coke behave under given conditions. However, it has been possible to formulate only probabilities, for the most part, as to the reactions that the sulfur will undergo in the blast furnace. Several small laboratory experiments have been performed to study the absorption of sulfur by iron from the coke, but it will be necessary to check all of this by studying the actual spongy iron, the coke, and the lime or slag from different zones of the furnace.

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[8] J. E. Johnson, Jr.: "Principles, Operation and Products of the Blast Furnace," 146–7, New York, 1918.

It is the intention to place special emphasis on the conditions under which the sulfur *enters* the iron, rather than the slagging conditions that promote its desulfurization. This latter has been studied to some extent, and, at least in its practical aspects, is fairly well understood. However, the factors involved in the entry of the sulfur into the iron are not well known. The main factor that affects the contamination of the iron by the sulfur in the upper part of the furnace, as shown by the small-scale experiments described here, is the form in which the sulfur occurs in the coke. Further experiments on a large scale, using coke from which the ferrous-sulfide form had been removed, will be of special interest. The action of lime in intimate contact with the coke in preventing contamination of the iron by sulfur is significant and worthy of further investigation on a larger scale. It is true that the liming of coke is not a new idea; it was investigated in some detail by the U. S. Steel Corpn. about ten years ago. However, not much seems to be known about the action of limed coke in the blast furnace under different operating conditions.

Summary and Conclusions

In another investigation, the Bureau of Mines has determined the manner in which sulfur is held in coke. Coke contains four characteristic sulfur constituents; namely, ferrous sulfide, adsorbed free sulfur, sulfates, and sulfur held in solid solution in the carbon.

When coke goes through the blast furnace with the remainder of the charge, the sulfur of the coke undergoes the following reactions:

1. In the reduction zone, the free sulfur and the sulfates are reduced to ferrous sulfide.
2. In the zone of spongy iron, considerable sulfur is taken up by the iron from the coke. The ferrous sulfide of the coke gives up sulfur at this point. Removal of this ferrous sulfide from the coke, or its conversion into calcium sulfide by liming the coke, largely eliminate this absorption of sulfur.
3. In the zone of preliminary slag formation, the absorption of sulfur by the iron is largely prevented.
4. In the combustion zone, the sulfur released from the coke is probably entirely absorbed by the slag. In this region, the slag also removes from the iron the excess sulfur, which entered the iron in the spongy-iron zone.

A proposed further study of the sulfur reactions in the blast furnace as related to the forms of sulfur in the coke has been briefly outlined. It is proposed to place the emphasis on the factors causing the entrance of the sulfur into the iron rather than the desulfurization of the iron in the slagging zone.

ACKNOWLEDGMENT

The sincere thanks of the writer are offered to A. C. Fieldner, supervising fuels chemist, Pittsburgh Experiment Station, U. S. Bureau of Mines, at whose suggestion this investigation was started.

DISCUSSION

ALFRED R. POWELL.—The first thing, of course, is to find out whether our theory of the action of sulfur in the furnace is right. The next thing is to see if the form of sulfur in coke cannot be changed so that contamination of iron will not occur in the upper part of the furnace and perhaps at the tuyere level.

Reference is made in the paper to the liming of coke. Some work was carried on by the U. S. Steel Corpn. along that line about 10 years ago, when Mr. Campbell, I believe, presented a paper on the subject. At present we are steaming the coke so that at certain temperatures the ferrous-sulfide constituent is then decomposed. We hope that coke treated in this way will show less contaminating influence when in contact with iron than coke in its original condition.

J. R. CAMPBELL, Everson, Pa. (written discussion).—Limed coke was investigated under my supervision, about a dozen years ago, the tests being made in the Koppers byproduct coke ovens. The coal used was washed Illinois coal of approximately the following analysis:

PROXIMATE ANALYSIS	PER CENT.
Volatile matter	39.05
Fixed carbon	52.34
Ash	8.61
Sulfur	2.09

COMPOSITION OF ASH	PER CENT.
Silica	47.85
Iron oxide	15.84
Alumina	23.98
Lime	4.41
Magnesia	1.66
Sulfur	1.11

At that time, sulfur forms in coal were not as thoroughly understood as they are now, but I give them as we understood them at the time: Sulfur as sulfides, 1.17 per cent.; sulfur as sulfates, 0.10 per cent.; other forms of sulfur, 0.82 per cent.; total iron, 0.93 per cent. Undoubtedly the sulfur reported as sulfides is FeS_2, but we did not have the courage to report organic sulfur, as is being done now by Doctor Powell and his co-investigators.

Instead of mixing lime with the powdered coal, powdered limestone of good quality was used. The coal was crushed until 98.23 per cent. passed through a ¼- in. screen, and 90.71 per cent. through a ⅛-in. screen. The limestone was fairly fine also, nothing on a 10-mesh screen and 63.4 per cent. on a 100-mesh screen.

The object of the test was to determine the effect, if any, of limestone on the sulfur in the coke, and its physical structure, as the claim had been made that the sulfur in the raw coal was transformed to calcium sulfide during the coking process, and the physical quality of the coke improved. Enough limestone was used with the coal, approximately 12½ per cent., to form a theoretical monosilicate. Some of our conclusions were as follows:

1. The temperature of the coking mass is insufficient for the complete transformation of the sulfur in the coal to calcium sulfide in the coke, during the coking process. Such temperature must be over 2100° F., perhaps nearer 2500° F.
2. The temperature in the oven is insufficient for the complete transformation of lime-silicates, as evidenced by the free lime in the coke.
3. It is not practicable to use more limestone than will combine with the ash of the coal to form a monosilicate.
4. The physical quality of the coke is improved by the limestone, if too much free lime does not remain, and if such additions are for the formation of a monosilicate, provided always that high temperatures prevail at, or near, the close of the coking process.
5. The decomposition of the limestone and the action of the resultant CO_2 gas, on the incandescent coke, forms high percentages of CO gas, which lowers the natural B.t.u. value of the gas about 10 per cent.
6. It follows naturally from 5 that the percentages of yield from limed coke is probably lower than from a run-of-mine coke.
7. The coking time will be extended by the introduction of limestone into the charge, if any beneficial results are to be expected.
8. The total ammonia yield will be increased quite materially.
9. To attain sufficiently high temperatures in the coking mass for the desired chemical reactions and permanent physical qualities in the coke will necessitate maintaining the flues of the coke oven at critical temperatures, say 2700° to 3000° F.
10. Limed coke has but little practical value from a blast-furnace standpoint, unless all of the sulfur is converted into calcium sulfide during the coking process, or unless a silicate is formed, for the reason that sulfur other than calcium sulfide will enter the iron readily, and that the free lime will cause the coke to crumble on exposure to the atmosphere.

Our general conclusions were as follows:

1. The tests show that about one-third of the total sulfur in the coke was transformed to calcium sulfide during the coking process by the addition of 12½ per cent. powdered limestone; perhaps too small a percentage to have any metallurgical significance in blast-furnace practice, where the desideratum is to have the total sulfur in the form of calcium sulfide.

2. The tests also show that the total sulfur was actually increased about 10 per cent., caused by the otherwise volatile sulfur combining with the lime of the limestone during the coking process.

3. Even if calcium sulfide was formed completely during the coking process, it would be extremely hazardous to assume that the calcium sulfide, thus formed, would pass unchanged through the blast furnace into the slag, in view of well-established claims that calcium sulfide is reduced by metallic oxides at low temperatures, or in other words, acts as a sulfurizing agent.

4. Much higher temperatures, perhaps critical in the flues and oven walls, are required for the formation of calcium sulfide than ordinarily prevail during the coking process. While there is a one-third transformation of sulfur in the coking mass at 2100 ° F., there is no positive data to show that 100 per cent. efficiency will be obtained at higher temperatures.

5. The free lime is proportional to the temperature in the mass, though it appears to be easier to form a monosilicate than calcium sulfide at the usual working temperatures. Approximately 85 per cent. of the total lime in the limed coke is in combination at 2100° F. A part of the above deficiencies may be offset by finer pulverization of both coal and limestone.

6. It is possible to make a good grade of blast-furnace coke from an inferior coking coal, from a physical standpoint, due, no doubt, to the formation of a monosilicate.

7. The effect on the ammonia yield was to be expected, being increased from 10 to 15 per cent. The limed coke showed from 35 to 40 per cent. more free ammonia, and about 60 per cent. less fixed ammonia.

8. The coking time for limed coke will be increased about one-third over the regular time; *i.e.*, if any real good is to be accomplished by the limestone addition.

9. The cost of the coke would be increased about 50 per cent., by the cost of crushing the limestone, increase in coking time, and decrease in coke yield, so that there would have to be many advantages to offset this one disadvantage; however, this would be partly taken care of in the blast-furnace practice, which would require little, or no limestone when using limed coke.

I do not recall all the details of the laboratory methods for determining calcium sulfide in the limed coke, but they were considered standard methods at the time and I judge would be so considered now. The powdered limed coke was treated with K_2S to determine the CaS, if I recall the method correctly.

In addition to the experiments with limed coke, we used various other chemicals from time to time, mixing them with the coal in beehive ovens. Chief of these chemicals were calcium chloride and sodium carbonate. Calcium chloride could be considered the equivalent of sodium chloride, or common salt, which has been recommended at various times.

The use of soda ash was based on the assumption that the sulfur would be transformed into soluble sulfates, which could be washed out of the coke during the quenching process. So far as we could determine, the sulfur in the coke was not affected by any of the chemicals used. In the case of the calcium chloride, the effect seemed to be deleterious from a byproduct standpoint, as free ammonia was changed to fixed ammonia, which is objectionable to the direct recovery of ammonia as ammonium sulfate in a sulfuric-acid bath. It means increased still capacity and the use of more lime.

The author hints at the trying out of limed coke in the blast furnace; I do not think that we did this because we did not have sufficient limed coke to make such a test. We did, however, test out the limed coke in cupolas, but with very indifferent and inconclusive results. At the time, we thought that the limed coke would not do any good in eliminating the sulfur in the iron; in fact, I think the iron picked up more than the usual amount.

R. H. Sweetser, Columbus, Ohio.—In 30 years of blast-furnace experience, I have had two opportunitites to get iron sponge. On one occasion, when the blast-furnace lining telescoped we stopped a furnace that was in normal working condition and put water in at the top, completely drowning out the furnace. When we dug it out we found iron sponge near the stock line, but I was more interested in getting the furnace working again than in finding out the composition of that iron sponge.

The next opportunity was in 1912, when a furnace that had been in blast only five weeks blew off part of its top; inside the bosh at the level of the mantle there was some iron sponge, which contained 0.38 per cent. silicon; the phosphorus in basic iron at the time was 0.004 per cent. It contained a trace of manganese but the iron made by the furnace the day before had 0.107 per cent. The graphitic carbon was a trace, the combined carbon was 0.68 per cent., and the sulfur was 0.136 per cent. I think that proves that the iron sponge at the level of the bosh had absorbed sulfur from somewhere. The day before this explosion, when the furnace was working normally, the iron contained 0.94 per cent. silicon, 0.154 per cent. phosphorus, 0.107 per cent. manganese, and 0.022 per cent. sulfur. I am inclined to think that the iron sponge was formed while the furnace was working all right, so that the author has proved in his laboratory, and we have proved in the blast furnace, that iron sponge will absorb sulfur that later must be taken out.

F. W. Davis, New York, N. Y.—On page 595, the author says: "The factor that determines how much sulfur will be taken up by the iron in the sponge-iron zone is, therefore, not the amount of sulfur in the coke but the speed with which the ferrous sulfide formed at the surface of the iron diffuses into the interior of the iron." Could that be stated in this way: The amount of sulfur taken up by the sponge iron will be in inverse proportion to the time which the iron is exposed in the zone of sulfur vapor?

Alfred R. Powell.—Yes; it could be stated that way.

F. W. Davis.—From the outline of procedure of the investigation of oxygen-enriched blast prepared for study by the Bureau of Mines, we feel assured of a minimum coke saving of 15 per cent. with no changes in present furnace design, except elimination of the hot-blast stoves, and with a blast of about 28 per cent. oxygen. This means that if we use coke with a sulfur content 18 per cent. higher than is permissible today, the amount of sulfur charged per ton of iron will be the same as at present. The smelting capacity will be increased as the coke consumption decreases, and the length of time that the iron remains in the sponge zone will be shortened in proportion. If it is true that the sulfur absorbed by the iron is in direct proportion to the time of exposure in the spongy zone, we may expect to be able to use coke with a sulfur content from 35 to 40 per cent. higher than today's limit and make low-sulfur iron in this particular practice. Much higher percentages of sulfur may be allowable as we have higher oxygen concentration. Do you think that would be the case?

Alfred R. Powell.—That would be hard to say. Of course, one cannot say that the amount of sulfur in the coke will not affect the amount of sulfur eliminated because there is a slag equilibrium which helps determine that fact. The slag will take up just so much calcium sulfide and no more.

F. W. Davis.—If the iron does not absorb the sulfur in the sponge zone by passing it through more rapidly, would not that have the effect?

Alfred R. Powell.—I should think so, provided the sulfur is not so high when the zone of slag formation is reached as to overburden the slag.

A. L. Feild, New York, N. Y.—If you increase the oxygen content of air you will also increase the pressure of oxygen in the blast furnace, especially in the region around the tuyere. In an oxidizing atmosphere, the tendency of sulfur is to go into the iron and not into the slag, as is well known by any one who has tried to desulfurize steel.

In what form does the iron get down from the ore: Does it run down the side of the furnace in a good size stream? or does it come down in the form of molten drops? You can see what effect that has on desulfurization in the hearth; if the iron comes down in the form of drops it has to fall through a slag area and the rate of fall depends on the viscosity of the slag. On the other hand, if the iron trickles down in a stream and goes right to the bottom, all desulfurization of the hearth is between the slag and the pig iron.

If we could analyze a few pellets of iron or iron shot found in the slag and some iron from the same tap, and found much difference in the sulfur, we could say there had been considerable desulfurization in the hearth of the furnace.

George A. Orrok, Willimantic, Conn.—I have read many things about how the iron comes down in a blast furnace and there are as many theories covering the reduction of iron. My idea is that anything as fusible as iron sulfide comes down as drops, at times quite large ones; but a good deal of the iron sulfide goes into the slag. If you can keep the slag at the right composition, there will be considerable reduction of sulfur. At any rate, the hearth of the blast furnace, or just above the hearth, is the place where the sulfur should be eliminated.

R. H. Sweetser, Columbus, Ohio. (written discussion).—Some years ago, the top of a furnace that had been in blast only five weeks was blown off and all the stock down to about 35 ft. from the top of the furnace was also blown out.

When we dug out that furnace, we found some peculiar conditions. Immediately after the stock was blown out and the top blown off we "drowned out" the furnace by closing all the tuyeres and pouring water into the top; then we opened up all the tuyeres and dug out all the stock that would come out. The next day the hearth was empty, but in the bosh a lot of stock was hanging around the bosh wall, forming a ring scaffold.

A funnel of sponge iron about 1 in. thick extended from the top of the bosh to just a few feet above the tuyeres; this funnel was about 2 ft. 11 in. from the walls of the bosh and had the same angle of slope as the bosh walls. Between this funnel of sponge iron and the walls was a hard mixture of small coke, fine burned lime, and fine coke dirt. We tried to knock down this scaffold by dumping heavy scrap into the top of the furnace, but we could not dislodge much of it, so we opened holes through the walls around the top of the bosh and tried to break the iron funnel, but the bars rammed through it just as if it were sheet lead. We finally cut the iron apart by going inside the furnace with cutters. The analysis of this iron was as follows:

	Per Cent.
Silicon	0.38
Phosphorus	0.004
Manganese	Trace
Graphitic carbon	Trace
Combined carbon	0.68
Sulfur	0.136

Previous to the explosion, or slip, that had blown off the top of this furnace, there had been many bad slips for a period of 36 hr. and the stock would not move in the furnace unless we took off the blast.

I believe that the normal path of coke in the lower part of the furnace is along the bosh walls. The experiments of the investigators of the Bureau of Mines show that coke is completely burned within a short distance of the nose of the tuyeres; therefore, great volumes of hot gases must be ascending through the center of the furnace from the tuyere level upwards. It is doubtful if much of the stock goes down through the center of the furnace below the top of the boshes.

About 5 years ago, Mr. Feild presented a paper on blast-furnace slags, in which he calculated that the size of the little particles of iron coming down through the bath of slag is more like a mist than anything else. I do not agree with his idea of the way in which the iron comes down into the hearth. My experience has been that iron comes down in rather large bunches instead of in small drops. The iron sponge will take somewhat the form of the lumps of ore from which the iron has been reduced. My experience with the furnace that blew off its top points to the fact that the iron sponge comes down from the upper part of the furnace into the hearth in the path between the slowly descending coke along the walls of the bosh and the rapidly ascending volume of hot gases. If the men of the Bureau of Mines can intercept iron sponge at different levels as it comes down through the experimental blast furnace at Minneapolis, it will give a great deal of light on this subject.

William H. Blauvelt, New York, N. Y.—Does not Mr. Campbell sum up the situation when he says: Aside from the control of the sulfur that enters the iron by means of a viscid or limy slag, which method of course has its limits, the only practical control of the amount of sulfur that finally gets into the iron is the limiting of the sulfur in the coal from which the coke is made.

Alfred R. Powell.—The suggestion that limed coke be tried out in the blast furnace is only a minor part of the paper. It has never been used in the blast furnace so far as we know; therefore, the objections, given here to its use are not based on practical experience. The principal object of the paper was to attempt an explanation of how the sulfur in coke behaved in the furnace and the mechanism of the transfer of sulfur from the coke to the iron and slag. Mr. Sweetser's observations, gleaned from experience, seem to bear out some of the theories advanced. It will be interesting to gather more data of this nature, for we must know how the sulfur behaves in the blast furnace before we can formulate practical methods of control.

Desulfurizing Power of Iron Blast-furnace Slags

By Richard S. McCaffery,* E.M., and Joseph F. Oesterle,† Madison, Wis.

(Canadian Meeting, August, 1923)

This investigation was undertaken to determine the quality of different iron blast-furnace slags as desulfurizing agents, and the possibility of using, in the blast furnace, materials of higher sulfur content than is now common. The solution of the problem would render possible the use of higher sulfur iron ores for pig iron and cokes containing more sulfur than now employed, and the economic manufacture of irons and steels of lower sulfur content.

It is generally assumed that the sulfur that reaches the lower part of the iron blast furnace is removed from the pig iron by combining with lime to form calcium sulfide. Most of the sulfur introduced with the charge is in combination with iron or manganese or, perhaps in the coke, as a carbon compound.[1] The iron and manganese sulfides are more soluble in the iron than is calcium sulfide, and as the latter is generally said to to be more soluble in the slag than the iron and manganese sulfides, this greater solubility of calcium sulfide in the slag tends to free the metal from sulfur. This greater solubility of the calcium sulfide is used not only to reduce the sulfur contents of the pig iron in the blast furnace, but by this means also low-sulfur steels are produced in the open-hearth and electric furnaces. The reaction[2] $FeS + CaO + CO = Fe + CaS + CO_2$ requires a high temperature and strongly reducing conditions in the presence of lime.

The constitution of blast-furnace slags, first studied scientifically by Vogt,[3] indicated certain ranges of composition of the slag in which a primary crystallization of one mineral was followed by a crystallization of other minerals. Akerman[4] later determined the heats of solidification of silicates. The Geophysics Laboratory,[5] at Washington, has published the results of its work on the equilibrium of the binary systems lime-silica, alumina-silica, and lime-alumina, and later Rankin and Wright[6] published the equilibrium of the ternary system, lime-alumina-silica. This work of Rankin and Wright is the basic investigation on which all work on blast-furnace slags must be built, for all furnace slags are embraced in the lime-alumina-silica system, with the modification, that part of the lime is replaced with magnesia.

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* Professor of Metallurgy, University of Wisconsin.

† Assistant Professor of Metallurgy, University of Wisconsin.

[1] Robert Forsythe: "The Blast Furnace and Manufacture of Pig Iron." 220. N. Y., 1908. David Williams Co.

[2] H. M. Howe: "Iron, Steel and Other Alloys." 395. Boston, 1903. Sauveur & Whiting.

[3] Vogt: Int. Kongress Angew. Chemie (1903) II, 70.

For the Bureau of Mines,[7] Feild and Royster made determinations of viscosity over a range of composition that included the usual blast-furnace slags. The results of their work enable a blast-furnace operator to determine the effect of composition on melting point and fluidity of the slag. The fluidity is of greatest importance because it regulates the flowing qualities of the slag through the slag notch and away from the furnace, also because it controls the rate at which the slag descends through the bosh. The longer that the slag globules remain in contact with the iron globules, while descending through the bosh, the better for the formation of calcium sulfide and its solubility in the slag, which facilitate the desulfurization of the iron.

Regulation by changes of composition, which, in turn, affect the viscosity and thus the contact time in the bosh, would give a method of control of furnace operation that offers the furnace man a scientific basis of procedure. The viscosity of the slag would also be of great importance in the crucible, or hearth region, where the metal and slag remain in contact between tapping times and where further desulfurization of the iron may take place by partition solution of the calcium sulfide between the iron and the slag. If the slag is of low viscosity, molecules may move more freely through it, diffusion of the sulfur through the mass of the slag is more rapid and desulfurization is thus increased. Although authorities practically agree that the sulfur, as calcium sulfide, is dissolved in the slag, no information as to the amount of sulfur any slag can take into solution is available; it is to supply this data and also data with regard to manganese sulfide that the investigation here described was undertaken.

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[4] R. Akerman: *Stahl und Eisen* (1886) 281.

[5] Arthur L. Day, E. S. Shepherd, and Fred E. Wright: The Lime-silica Series of Minerals. *Am. Jnl. Sci.* (1906) **22**, 265.

E. S. Shepherd, G. A. Rankin, and Fred E. Wright: The Binary Systems of Alumina with Silica, Lime, and Magnesia. *Am. Jnl. Sci.* (1909) **28**, 293.

[6] G. A. Rankin and Fred E. Wright: The Ternary System $CaO-Al_2O_3-SiO_2$. *Am. Jnl. Sci.* (1915) **39**, 1.

[7] Bureau of Mines, *Tech. Papers* Nos. 157, 187, 189.

From the work of Rankin and Wright, it is known that a lime-alumina-silica mixture of definite composition will begin to freeze, or solidify, by crystallizing out a definite mineral entity. As the temperature of the melt decreases, this primary mineral crystallizes from it and the composition of the residual liquid changes until a critical composition of the liquid is attained, when a crystallization of two minerals begins. As the temperature drops still further and as a result of the two minerals crystallizing out, the composition of the liquid residue changes until a second critical composition is reached, when three minerals separate out; and so on until the whole mass has solidified. The slag thus formed is composed of certain mineral entities in definite proportion. These minerals, or compounds, exist as entities in the molten mass as well as in the solid,[8] and the minerals or compounds possess definite chemical and physical properties. From Feild and Royster's work, the viscosity of any melt in the range of composition of ordinary furnace slags can be obtained. If, in addition to the viscosity, the solubility of the various sulfur compounds that might be present in the slag were known, additional means for controlling the blast-furnace operations would be given the operator. With this idea in mind, the solubility of certain compounds of sulfur in the various mineral entities or compounds present in blast-furnace slags was studied.

SUMMARY OF WORK OF RANKIN AND WRIGHT

In their work on the ternary system, lime-alumina-silica, Rankin and Wright[9] determined the concentration temperature diagram for the

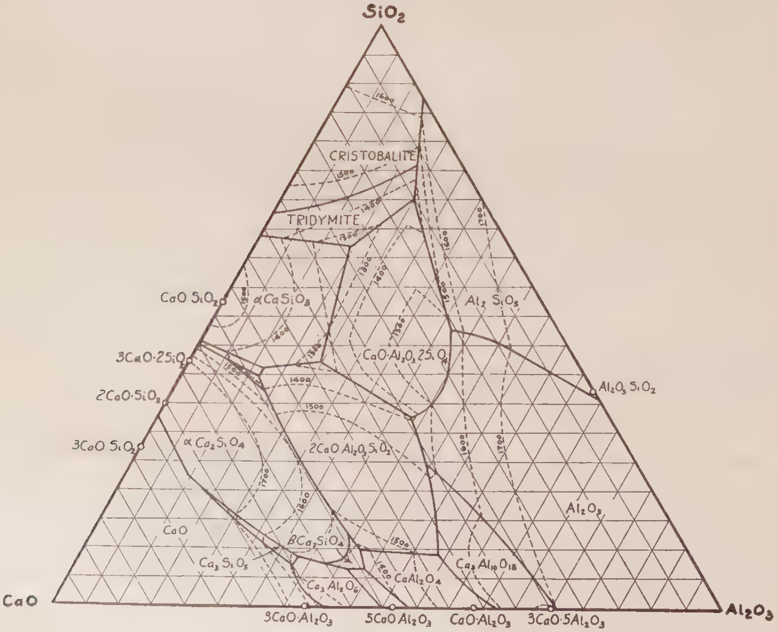

FIG. 1.—EQUILIBRIUM DIAGRAM.

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[8] Bureau of Mines, *Tech. Paper*, No. 187, 5.

[9] *Am Jnl. Sci.*, 189, 1.

system, with isotherms and melting temperatures of compounds and invariant points. Their results are reproduced in Fig. 1. All blast-furnace slags are included in the limits $CaO.SiO_2$ (calcium bisilicate), $CaO.Al_2O_3.2SiO_2$ (anorthite), $2CaO.Al_2O_3.SiO_2$ (gehlenite), and to a lesser extent of $3CaO.2SiO_2$ (akermanite, or calcium sesqui-silicate) and $2CaO.SiO_2$ (calcium monosilicate). Practically all slags will be included in the region of the first three, so that this will be the part of the work of Rankin and Wright to which detailed reference will be made hereafter.

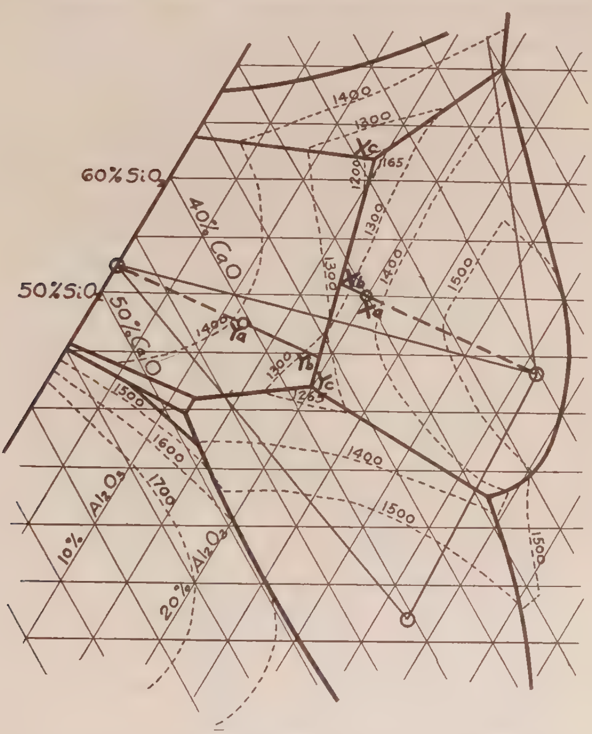

FIG. 2.—PART OF EQUILIBRIUM DIAGRAM ENLARGED.

To illustrate the use of the equilibrium diagram, Fig. 1, a part is shown enlarged in Fig. 2. If a molten slag of the composition SiO_2 50 per cent., CaO 30 per cent., and Al_2O_3 20 per cent., indicated by the point X_a, is allowed to cool and reach equilibrium conditions, as the slag composition is located in that region in which anorthite ($CaO.Al_2O_3.2SiO_2$) first crystallizes out, this mineral will separate out at 1350° C. As anorthite is higher in alumina than the slag X_a, its separation from the mass will decrease the alumina in the residual liquid slag. The locus of the composition of this residual liquid will be on the straight line prolonged, connecting the point X_a with the point that represents the composition of the component that separates. With the crystallization of anorthite and a further reduction of the temperature, the composition of the residual liquid passes from X_a to X_b on the line separating the anorthite field from the bisilicate field. At the point X_b and a temperature of 1275° C., both anorthite and calcium bisilicate are in equilibrium with the liquid and crystallize simultaneously from the molten material. The composition of the residual liquid changes from X_b to X_c as the temperature continues to drop, until 1165° C. is attained at X_c. At this point, silica is in equilibrium with calcium bisilicate, anorthite, and the liquid, and the three minerals crystallize out, the composition and temperature remaining stationary until the slag has entirely solidified.

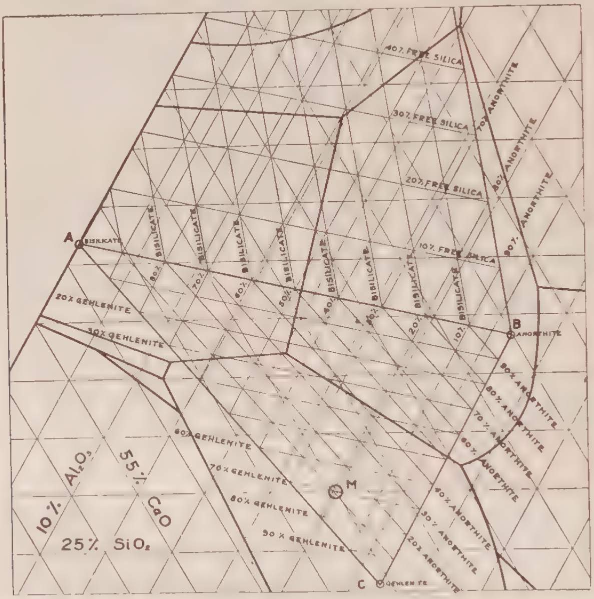

FIG. 3.—SHOWING TERNARY SYSTEM INSIDE OF AND PART OF LARGER SYSTEM.

If Y_a represents the composition of a slag, the first mineral to crystallize out will be calcium bisilicate, and as this continues to crystallize, the composition of the residual liquid passes along the line Y_a to Y_b where calcium bisilicate and anorthite are in equilibrium. These minerals crystallize and the composition of the residual liquid passes along the line Y_b–Y_c until the point Y_c is reached, where calcium bisilicate, anorthite, and gehlenite are in equilibrium with liquid when these three minerals separate until the whole melt solidifies.

The percentage of the different minerals present in any mixture may be determined as follows: In the area included in the triangle formed by joining the points representing the composition of calcium bisilicate, anorthite, and gehlenite, indicated by the points *A*, *B*, and *C* respectively, in Fig. 3, there is a ternary system inside of and part of the large ternary system $CaO-Al_2O_3-SiO_2$; the sum of the three components of the smaller system is equal to 100 per cent. Lines drawn parallel to each side divide the triangle into smaller triangles, representing differences of 10 per cent. each in the amount of each compound present; from these lines the percentage composition in bisilicate, anorthite, and gehlenite can be determined. If the percentage lines are drawn for the larger ternary system, the percentage of bisilicate, anorthite, and gehlenite corresponding to a given $CaO-Al_2O_3-SiO_2$ composition is obtained. Connect the points in the diagram representing the composition of the minerals that may separate; that is connect $CaO.SiO_2$ with $2CaO.Al_2O_3.SiO_2$ and $CaO.Al_2O_3.2SiO_2$, and the point $CaO.Al_2O_3.2SiO_2$ with $2CaO.Al_2O_3.SiO_2$. Parallel to each side draw lines one-tenth of each altitude apart and then read percentages of calcium bisilicate, anorthite, and gehlenite that correspond to a given percentage of lime, alumina, and silica on the base percentage system of division. Thus the point *M*, 40 per cent. lime, 30 per cent. alumina, 30 per cent. silica, in Fig. 3, will crystallize out 20 per cent. calcium bisilicate, 10 per cent. anorthite, and 70 per cent. gehlenite. Fig. 3 shows other triangles for deter-

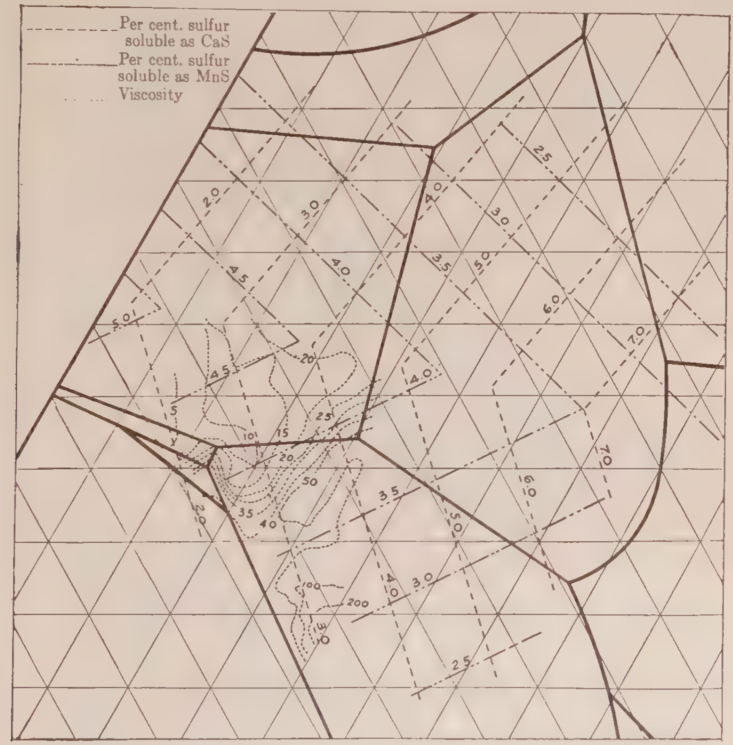

FIG. 4.—SULFUR SOLUBILITY AND VISCOSITY CURVES AT 1350° C.

mining the mineral composition of other slags in the ordinary range of blast-furnace composition.

SUMMARY OF WORK OF FEILD AND ROYSTER

The viscosity determinations of Feild and Royster[10] were made by rotating a crucible containing the melt, in which a graphite cylinder was immersed. This cylinder was so supported that its angular displacement could be accurately measured. The rotating crucible was heated by an electric resistor furnace and, while the melt was maintained at a constant temperature, the drag of the melt in the rotating crucible on the suspended graphite cylinder was measured and the viscosity determined.

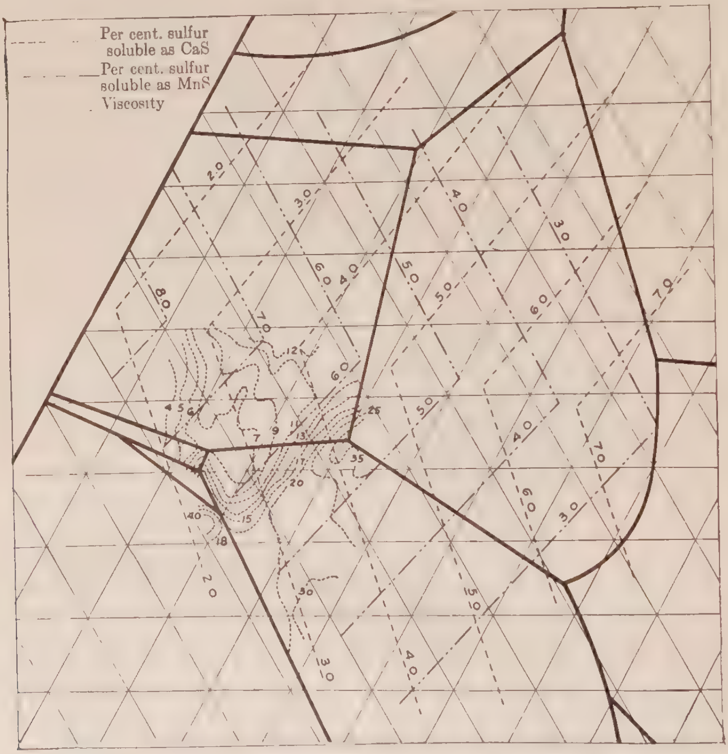

FIG. 5.—SULFUR SOLUBILITY AND VISCOSITY CURVES AT 1400° C.

The viscosity of melts varying in composition from 12 to 24 per cent. alumina, 28 to 50 per cent. silica, and 36 to 50 per cent. lime, was determined at various temperatures between 1300° and 1600° C. and the results tabulated, giving the temperatures at which a certain viscosity is attained, or the viscosity when a certain temperature is reached.[11] The viscosities found by Feild and Royster are plotted with the sulfur solubilities determined in this investigation so that both appear on the same diagrams; see Figs. 4 to 9.

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[10] *Loc. cit.*

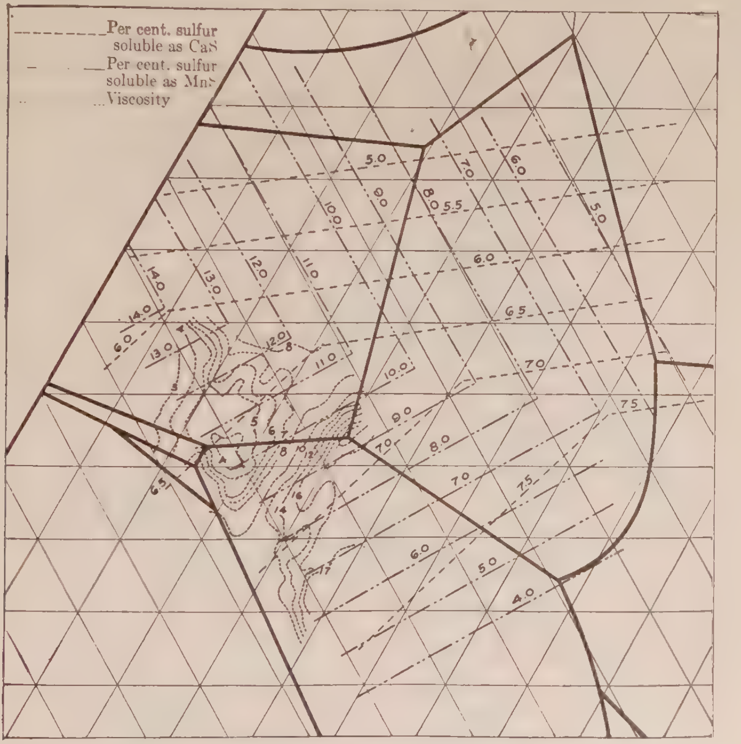

FIG. 6.—SULFUR SOLUBILITY AND VISCOSITY CURVES AT 1450° C.

SOLUBILITY OF CALCIUM SULFIDE IN ANORTHITE, GEHLENITE, AND CALCIUM BISILICATE

The investigation of sulfur solubility was begun by determining the solubility of calcium sulfide in each of the three minerals, anorthite, calcium bisilicate,[12] and gehlenite, which constitute the largest part of the ordinary blast-furnace slags. Mixtures of chemically pure lime, alumina, and silica were made in the proportions present in anorthite, and calcium sulfide was added to these in definitely varying amounts. The slags resulting from these melts were carefully analyzed and the results reported are on the basis of these analyses. A series of cooling curves was determined for anorthite with these different percentages of calcium sulfide. The melts were made in an electric resistor furnace of the Hoskins type, Fig. 10; the crucibles used were made of graphite, and the temperature determinations made by a Leeds & Northrup optical pyrometer. This work amounts to the determination of the equilibrium of the binary systems: $CaS–CaO.Al_2O_3.2SiO_2$, $CaS2C–aO.Al_2O_3.SiO_2$, and $CaS–CaO.SiO_2$.

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[11] Bureau of Mines, *Tech. Paper* No. 187.

[12] Unpublished thesis of K. L. Hussissian, Univ. of Wis., 1921.

The mixtures placed in the graphite were heated until they were freely liquid and, by means of the optical pyrometer, readings were made at equal time intervals after the current had been cut off and the furnace was cooling. These cooling curves do not give sharp retardations as do the cooling curves of metals, such as the cooling curve of metallic silver[18] *a*, Fig. 11, but they were similar to the general silicate cooling curve, as *b*, *c*, *d* and *e*, in which the critical points are indicated by changes in the slope of the curves. A number of these cooling curves are shown in Fig. 12, those selected being types for the various silicate minerals and sulfides investigated. Tables 1 to 6 give the notebook logs for the curves illus-

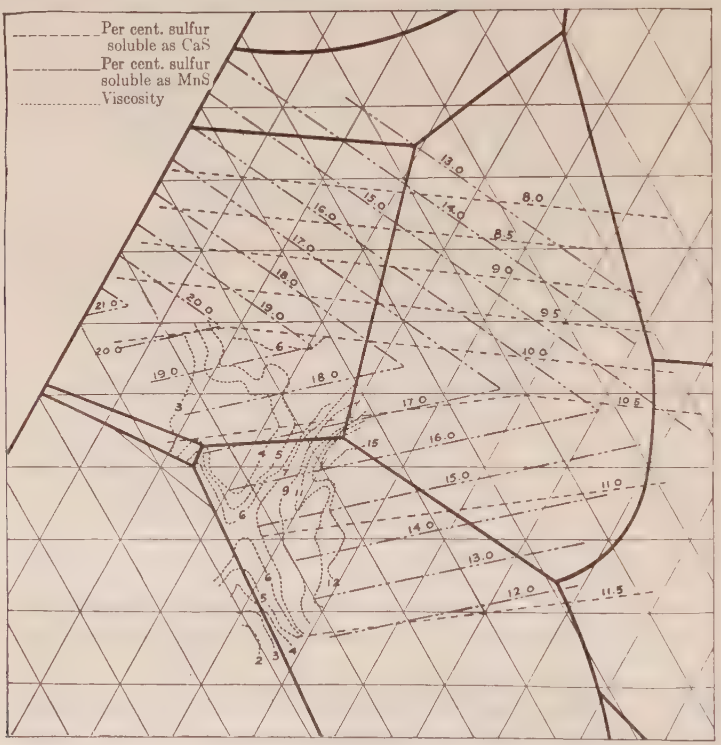

FIG. 7.—SULFUR SOLUBILITY AND VISCOSITY CURVES AT 1500° C.

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[18] Charles H. Fulton: "Principles of Metallurgy; an Introduction to the Metallurgy of the Metals." 237. N. Y., 1910. McGraw-Hill Book Co.

trated. On an individual curve, it was sometimes difficult to select the critical point, but when interpreted with the other curves of the series, the thermal arrests or changes in the cooling rate were readily distinguishable in every case.

The system calcium sulfide-anorthite forms a eutectic with 6.00 per cent. calcium sulfide melting at 1444° C., with a probable compound containing 11.0 per cent. calcium sulfide melting at 1500° C.; this is illustrated in *a*, Fig. 13. In a similar way, the equilibrium of the system CaS–CaO.SiO_2 (calcium bisilicate) was determined, showing a eutectic of 2.00 per cent. calcium sulfide melting at 1200° C. (see *b*); and for the system CaS–$2CaO.Al_2O_3.SiO_2$ (gehlenite) the eutectic contains 4.5 per cent. calcium sulfide, which melts at 1390° C., and a probable compound containing 6 per cent. calcium sulfide, which melts at 1460° C. (see *c*).

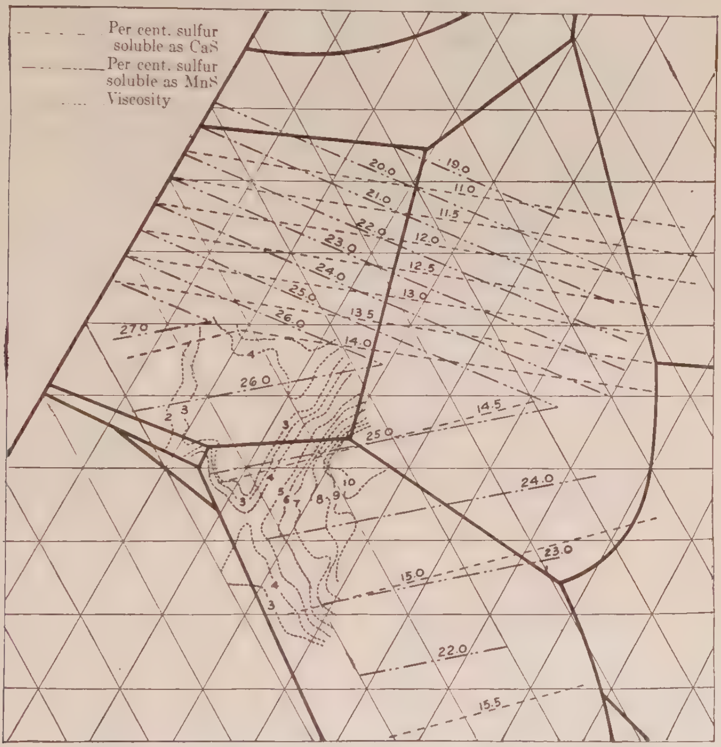

FIG. 8.—SULFUR SOLUBILITY AND VISCOSITY CURVES AT 1550° C.

In all our thermal equilibrium work, the pressure was atmospheric; but the atmosphere was neutral or reducing, because of the presence of the graphite crucibles. For high sulfide concentrations, the sulfide loss was considerable. The liquidus curves from the point where the present work ended to that point representing the melting point of the pure sulfide,

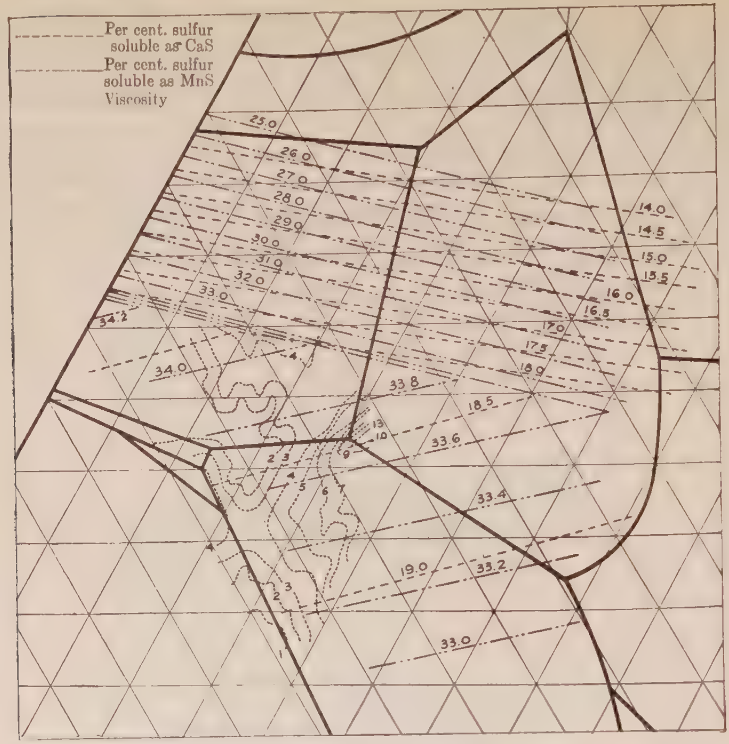

FIG. 9.—SULFUR SOLUBILITY AND VISCOSITY CURVES AT 1600° C

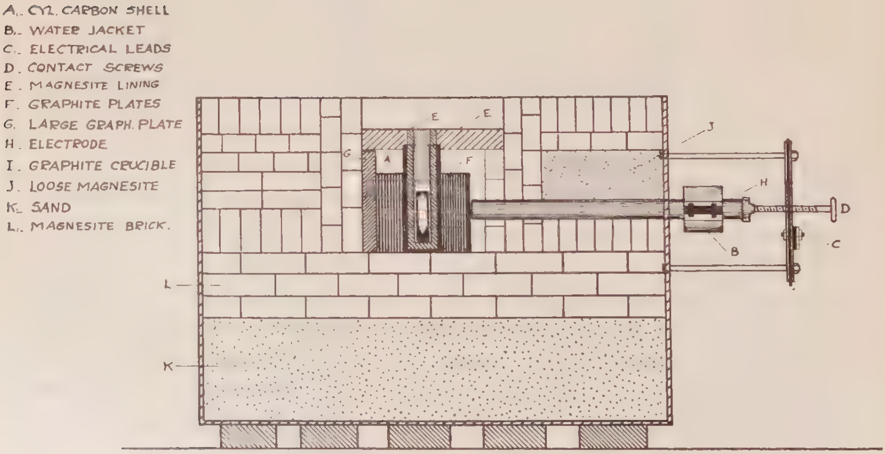

FIG. 10.—HOSKINS TYPE PLATE-RESISTANCE FURNACE.

represents only average conditions of equilibrium, but suffices for the present problem in showing possible solubility limits.

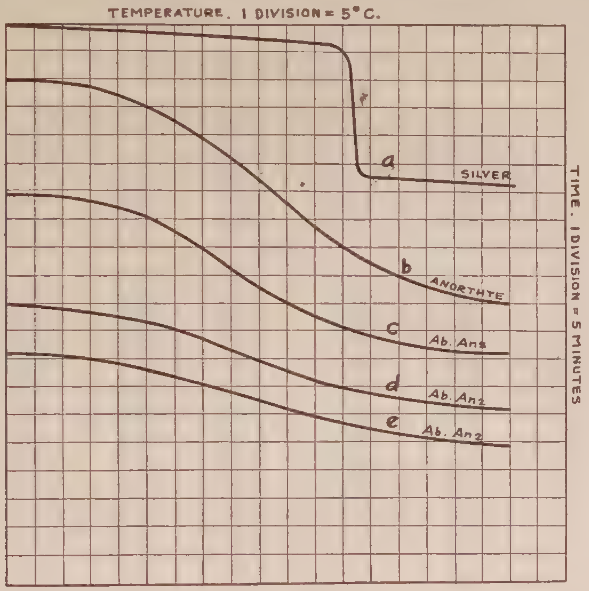

FIG. 11.—COMPARATIVE COOLING CURVES OF SILVER AND SILICATES.

TABLE 1.—*Calcium Sulfide 11 Per Cent. Plus Anorthite*

(Lab. No. BC-20; vol. 1, p. 149; readings every 20 sec.)

Milliamperes	Degrees C.	Milliamperes	Degrees C.	Milliamperes	Degrees C.	Milliamperes	Degrees C.
466	1601	446	1538	428	1475	416	1435
465	1597	446	1538	428	1475	416	1435
465	1597	444	1531	425	1465	414	1428
463	1591	444	1531	424	1462	414	1428
462	1588	442	1524	424	1462	414	1428
461	1585	441	1520	424	1462	413	1424
460	1582	438	1511	424	1462	413	1424
458	1576	435	1500	424	1462	413	1424
457	1573	436	1503	422	1455	412	1420
456	1570	434	1497	422	1455	411	1416
455	1567	433	1493	421	1452	411	1416
455	1567	432	1489	420	1449	410	1413
454	1564	431	1485	421	1452	409	1409
453	1561	431	1485	420	1449	409	1409
452	1558	431	1485	419	1445	408	1405
450	1551	430	1481	417	1438	408	1405
449	1548	429	1478	417	1438		

With the equilibrium of the systems, calcium sulfide with calcium bisilicate, anorthite, or gehlenite, being known, the values of calcium sulfide solubility taken from the curves in Fig. 13 are given in Table 7. The solubility used is the percentage indicated by the intersection of the horizontal temperature line with the liquidus curve.

TABLE 2.—*Calcium Sulfide 3.2 Per Cent. Plus Calcium Bisilicate*

(Lab. No. AC-4; vol. 1, p. 61; readings every 30 sec.)

Milliamperes	Degrees C.	Milliamperes	Degrees C.	Milliamperes	Degrees C.	Milliamperes	Degrees C.
465	1597	429	1478	403	1386	387	1322
464	1594	427	1472	401	1378	385	1314
461	1585	425	1465	400	1374	383	1306
458	1576	422	1455	399	1370	381	1297
456	1570	421	1452	397	1362	379	1289
452	1558	418	1441	396	1358	378	1285
450	1551	416	1435	395	1354	377	1281
448	1545	414	1428	393	1346	375	1272
444	1531	413	1424	392	1343	374	1268
442	1524	410	1413	391	1339	373	1264
438	1511	409	1409	389	1331	372	1260
436	1503	408	1404	388	1327	371	1256
435	1500	406	1397	389	1331	370	1252
432	1489	404	1389	388	1327	368	1243

TABLE 3.—*Calcium Sulfide 4.4 Per Cent. Plus Gehlenite*

(Lab. No. CC-8; vol. 2, p. 60; readings every 20 sec.)

Milliamperes	Degrees C.	Milliamperes	Degrees C.	Milliamperes	Degrees C.	Milliamperes	Degrees C.
463	1591	442	1524	420	1449	401	1378
462	1588	441	1520	419	1445	401	1378
461	1585	439	1514	416	1435	400	1374
460	1582	438	1511	415	1431	400	1374
459	1579	436	1503	414	1428	399	1370
458	1576	435	1500	414	1428	397	1362
456	1570	434	1497	412	1420	397	1362
455	1567	432	1489	411	1416	398	1366
455	1567	431	1485	410	1413	398	1366
452	1558	429	1478	408	1405	397	1362
451	1555	429	1478	408	1405	396	1358
450	1551	427	1472	406	1397	395	1354
448	1545	426	1468	405	1393	395	1354
447	1541	425	1465	405	1393	393	1346
446	1538	421	1452	403	1386	392	1343
445	1534	421	1452	402	1382	391	1339
443	1528	421	1452				

TABLE 4.—*Manganese Sulfide 2.5 Per Cent. Plus Anorthite*

(Lab. No. BM-6; vol. 2, p. 37; readings every 20 sec.)

Milliamperes	Degrees C.	Milliamperes	Degrees C.	Milliamperes	Degrees C.	Milliamperes	Degrees C.
463	1591	441	1520	425	1465	413	1424
463	1591	438	1511	425	1465	412	1420
460	1582	439	1514	424	1462	412	1420
459	1579	439	1514	424	1462	411	1416
457	1573	436	1503	423	1459	411	1416
454	1564	435	1500	422	1455	409	1409
454	1564	435	1500	420	1449	409	1409
453	1561	435	1500	421	1452	408	1405
452	1558	433	1493	418	1441	407	1401
451	1555	432	1489	418	1441	407	1401
448	1545	430	1481	418	1441	406	1397
448	1545	430	1481	417	1438	405	1393
447	1541	429	1478	416	1435	405	1393
447	1541	429	1478	416	1435	404	1389
445	1534	427	1472	414	1428	403	1386
444	1531	427	1472	414	1428	402	1382

TABLE 5.—*Manganese Sulfide 1.5 Per Cent. Plus Calcium Bisilicate*

(Lab. No. AM-1; vol. 1, p. 114; readings every 30 sec.)

Milliamperes	Degrees C.	Milliamperes	Degrees C.	Milliamperes	Degrees C.	Milliamperes	Degrees C.
466	1601	445	1534	426	1468	413	1424
465	1597	443	1528	425	1465	412	1420
464	1594	442	1524	425	1465	411	1416
462	1588	442	1524	423	1459	411	1416
462	1588	440	1517	422	1455	410	1413
460	1582	439	1514	421	1452	409	1409
458	1576	438	1511	420	1449	409	1409
457	1573	435	1500	420	1449	408	1405
456	1570	435	1500	420	1449	408	1405
455	1567	434	1497	419	1445	407	1401
454	1564	432	1489	417	1438	405	1393
452	1558	431	1485	416	1435	404	1389
451	1555	429	1478	415	1431	404	1389
450	1551	428	1475	415	1431	404	1389
449	1548	428	1475	414	1428	403	1386
448	1545	427	1472	414	1428	402	1382

TABLE 6.—*Manganese Sulfide 2.1 Per Cent. Plus Gehlenite*

(Lab. No. CM-1; vol. 2, p. 90; readings every 20 sec.)

Milli-amperes	Degrees C.	Milli-amperes	Degrees C.	Milli-amperes	Degrees C.	Milli-amperes	Degrees C.
465	1597	448	1545	434	1497	420	1449
464	1594	447	1541	432	1489	419	1445
463	1591	447	1541	431	1485	419	1445
461	1585	445	1534	428	1475	419	1445
459	1579	444	1531	425	1465	418	1441
458	1576	442	1524	425	1465	417	1438
457	1573	442	1524	425	1465	413	1424
456	1570	441	1520	425	1465	416	1435
456	1570	440	1517	424	1462	416	1435
454	1564	437	1507	422	1455	416	1435
453	1561	437	1507	422	1455	415	1431
452	1558	437	1507	421	1452	416	1435
449	1548	436	1503	421	1452	413	1424
448	1545	434	1497	420	1449		

TABLE 7.—*Solubility of Calcium Sulfide at Various Temperatures, in Percentage of Total Melt*

Degrees C.	Bisilicate	Anorthite	Gehlenite
1600	40.00	42.00	44.00
1550	31.00	33.00	35.00
1500	22.00	24.00	27.00
1450	13.00	17.00	18.00
1400	4.00	17.00	10.00
1350	4.00	17.00	10.00

In the higher silica compositions, some of the slags shown in the areas where sulfide solubility is indicated in our results in the solubility diagrams contain relatively small amounts of mineral silica; and, in the lower silica range, small amounts of calcium monosilicate and akermanite. The laboratory work on the calcium sulfide solubility in silica and these latter minerals is not completed because of certain difficulties. As these minerals are present in only a small part of the slag-composition range and as the calcium-sulfide solubility in silica is probably low, in the calculated results no calcium-sulfide solubility has been assumed for any of these minerals.

In the higher silica ranges of the diagram, this assumption will not change the solubility curves shown very much; if silica has any solubility for calcium sulfide, the curves will be spaced somewhat farther apart, but their direction will not change. In the lower silica ranges, the amount of akermanite and monosilicate is so small that the results in the region of commercial blast-furnace slags will be only slightly affected.

With the calcium sulfide solubilities from Table 7, and with the mineralogical composition of slag of any analysis determined as illustrated

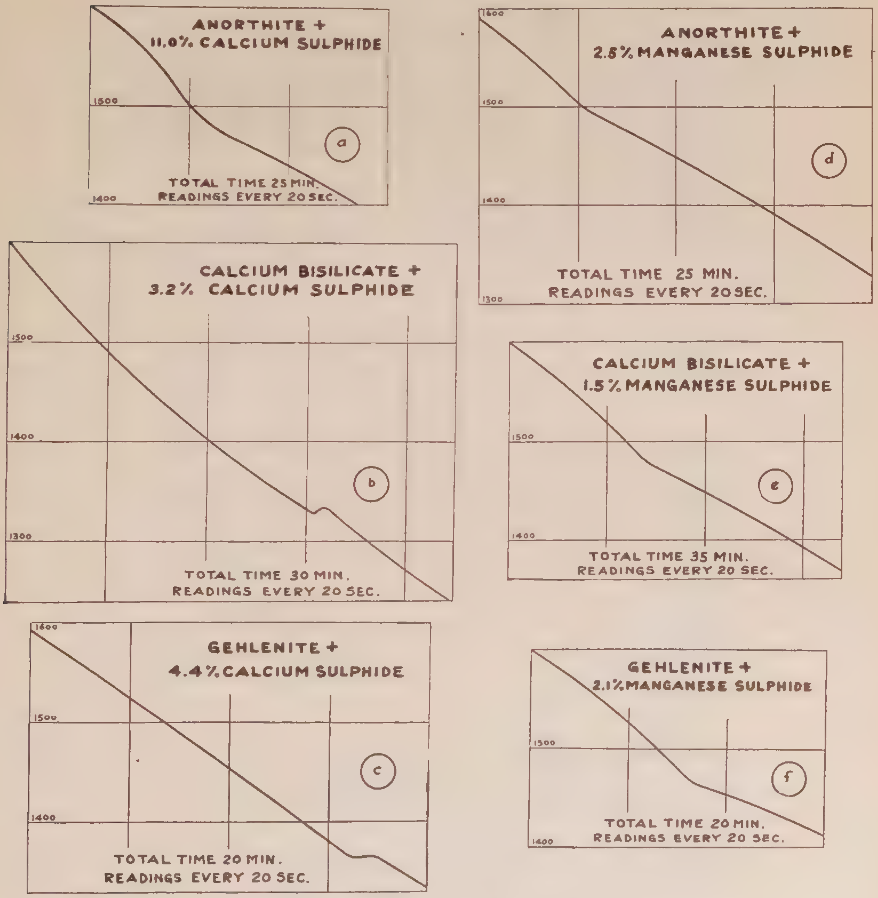

FIG. 12.—TYPICAL COOLING CURVES FROM LABORATORY NOTES.

in Fig. 2, the sulfur solubility in the form of calcium sulfide for slags of all compositions, including the range of composition generally employed for iron-furnace slags and extending well beyond this, was calculated at temperatures of 1350°, 1400°, 1450°, 1500°, 1550° and 1600° C., the results of these calculations being plotted on the triaxial diagrams, Figs. 4 to 9.

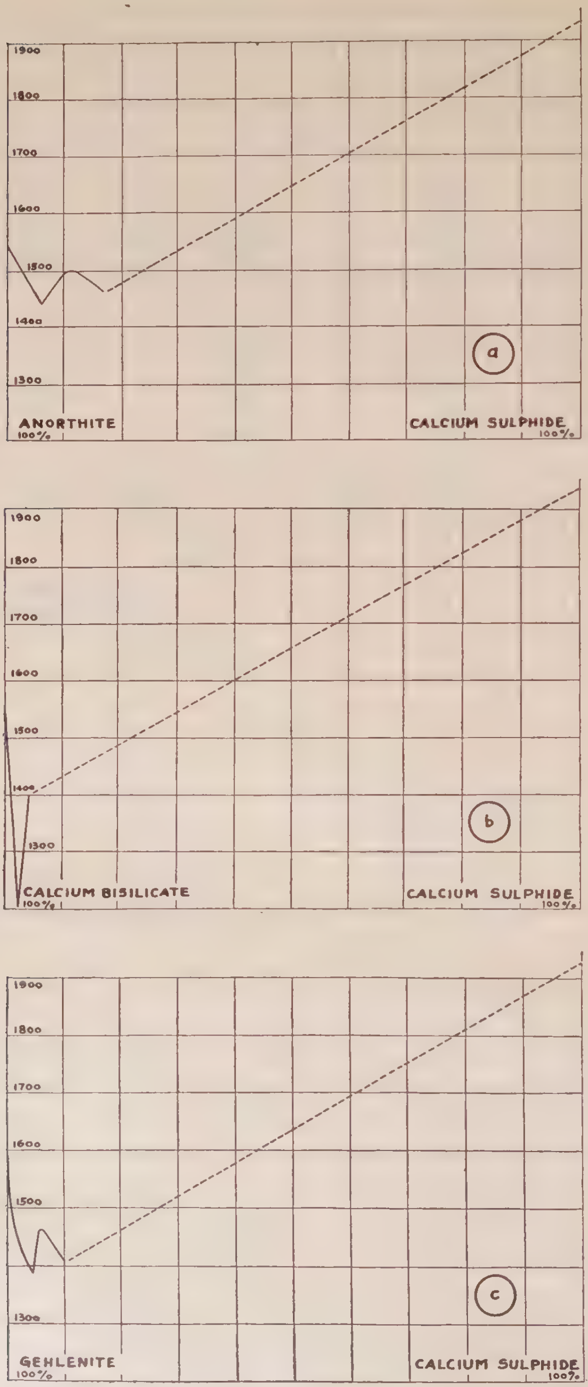

FIG. 13.—EQUILIBRIUM DIAGRAMS, CALCIUM SULFIDE-SILICATE MINERALS.

Solubility of Manganese Sulfide in Anorthite, Calcium Bisilicate and Gehlenite

As sulfur may reach the bosh of the furnace in the form of manganese sulfide, the solubility of manganese sulfide in calcium bisilicate, anorthite, and gehlenite was determined in the same way as the solubility of calcium sulfide. The equilibrium of the binary systems MnS–CaO.SiO$_2$, MnS–CaO.Al$_2$O$_3$.2SiO$_2$, and MnS–2CaO.Al$_2$O$_3$.SiO$_2$, is represented in Fig. 14, from which curves the manganese-sulfide solubilities given in Table 8 are taken.

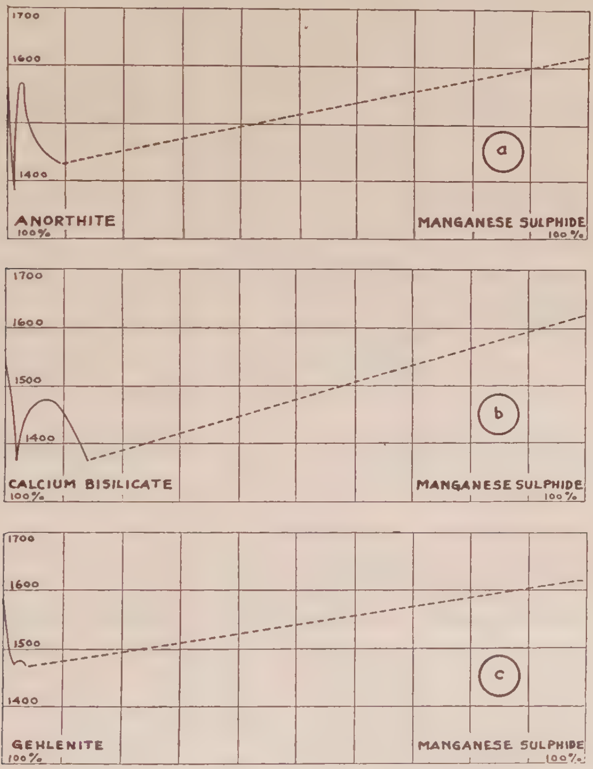

Fig. 14.—Equilibrium diagrams, manganese sulfide-silicate minerals.

With the mineralogical composition of the slag from Fig. 3 and the manganese-sulfide solubility from Table 8, the solubility of sulfur in the form of manganese sulfide was figured over the same range of composition as was used for calcium sulfide, and the results are plotted at 1350°, 1400°, 1450°, 1500°, 1550°, and 1600° C. on the same diagrams as the calcium sulfide, Figs. 4 to 9, inclusive. In this case, also and for the same reasons, no sulfur solubility has been assumed for silica or calcium monosilicate in any slag containing these minerals.

TABLE 8.—*Solubility of Manganese Sulfide at Various Temperatures, in Percentages of Total Melt*

Degrees C.	Bisilicate	Anorthite	Gehlenite
1600	93.00	91.20	89.00
1550	75.00	67.00	57.00
1500	58.00	43.00	25.00
1450	41.00	18.00	6.00
1400	23.60	9.00	6.00
1350	14.00	9.00	6.00

DISCUSSION OF RESULTS

Generally, an inspection of the solubility curves here given indicates that those slags highest in alumina have, at any definite temperature, a higher calcium-sulfide solubility than slags of lower alumina content. This confirms the observations of Turner[14] that the proportion of alumina is highest (in a block of slag measuring 5 by 5½ ft. at the top and 4½ ft. deep) in the parts that contain the most sulfur (as calcium sulfide). The results of this investigation also indicate that the maximum solubility of sulfur, as manganese sulfide, is in those slags having the larger content of calcium bisilicate. The effect of alumina, then, is to increase the solubility of the slag for sulfur in the form of calcium sulfide, and to decrease the solubility of sulfur in the form of manganese sulfide.

The solubility curves show, also, the marked increase of sulfur solubility, either as calcium sulfide or as manganese sulfide, with rises in the temperature of the slag. To emphasize this point, plots of sulfur solubility with varying temperatures have been made for slags of definite composition, and on these diagrams are plotted the viscosities from our plot of the determinations of the Bureau of Mines. The compositions selected are at the intersection of the 5 per cent. coördinates and are shown in Figs. 15 and 16. These figures show that if the temperature of the slag is increased from 1400° to 1600° C., the solubility of calcium sulfide is increased about three times and the solubility of the manganese sulfide is increased about six times. The temperature is a much more important factor than the chemical composition in effecting the sulfur solubility of the slag. In any particular region of slag composition, a change in the chemical composition one way or another, except as it affects indirectly the hearth temperature, has a comparatively slight effect on sulfur solubility compared with the solubility changes effected by a rise or fall of the slag temperature.

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[14] *Jnl.* Soc. of Chem. Ind., **24**, 1146.

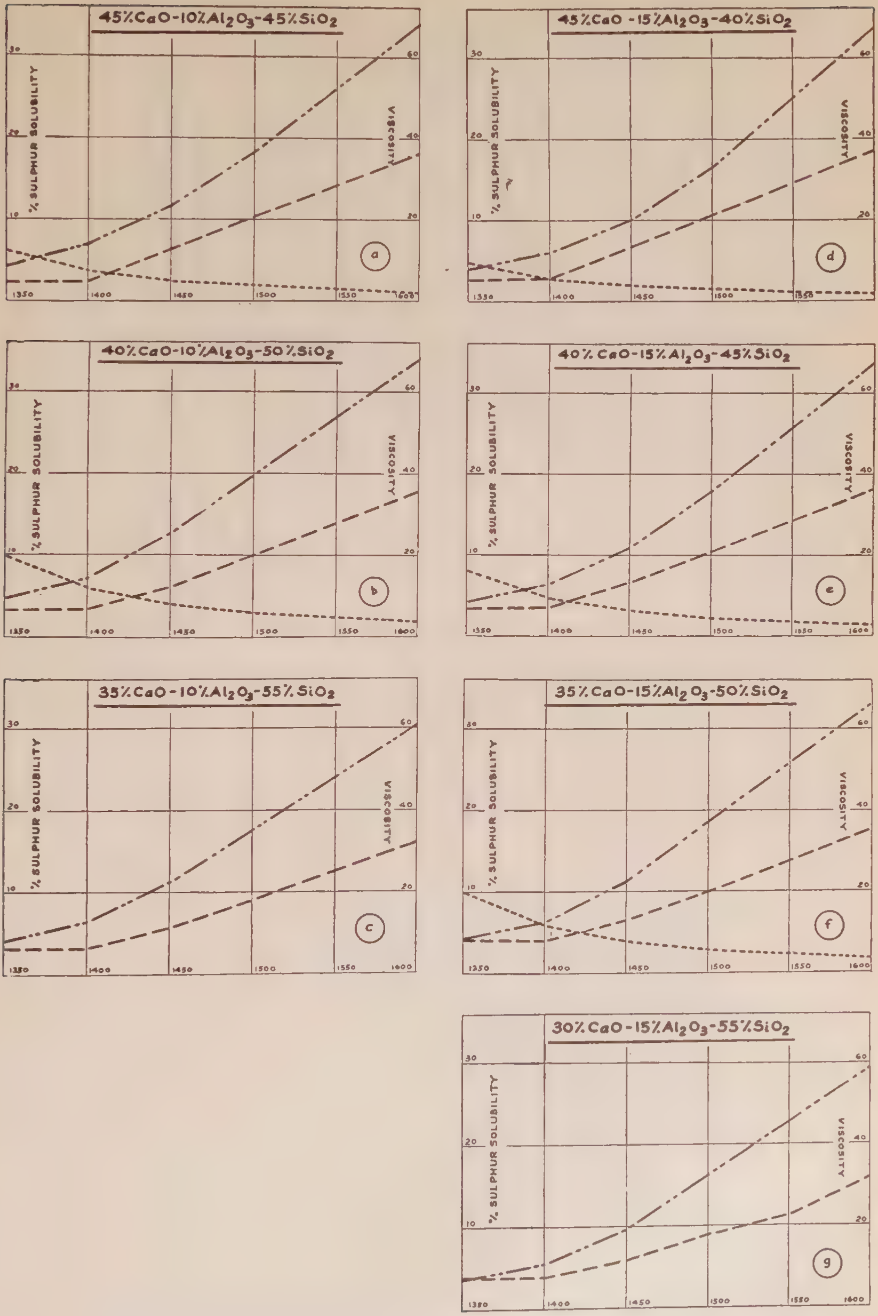

Fig. 15.—Calcium-sulfide and manganese-sulfide solubility and viscosity of certain slags.

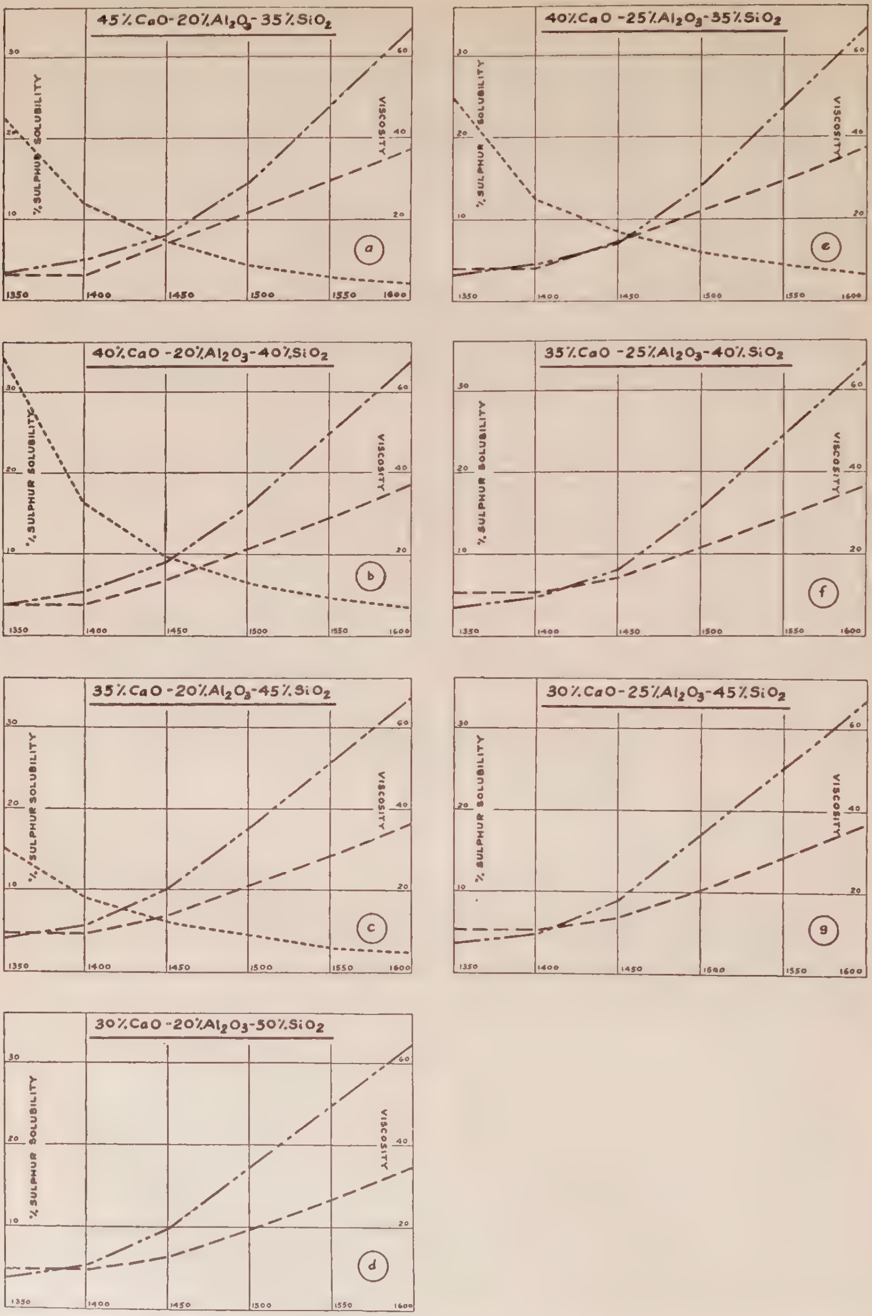

FIG. 16.—CALCIUM-SULFIDE AND MANGANESE-SULFIDE SOLUBILITY AND VISCOSITY OF CERTAIN SLAGS.

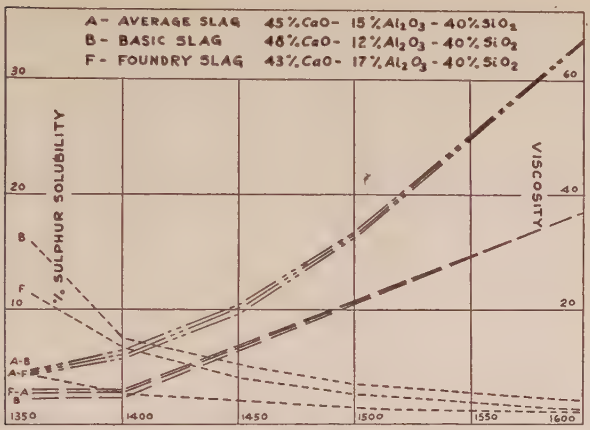

FIG. 17.—SULFUR SOLUBILITY AND VISCOSITY OF SLAGS OF FURNACES PRODUCING TYPE IRONS.

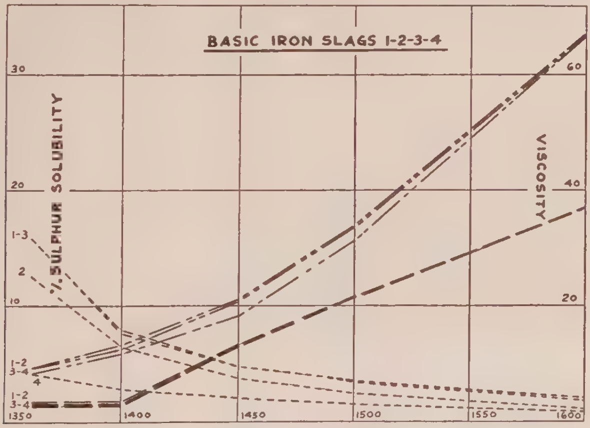

FIG. 18.—SULFUR SOLUBILITY AND VISCOSITY OF SLAGS OF FURNACES PRODUCING BASIC IRONS.

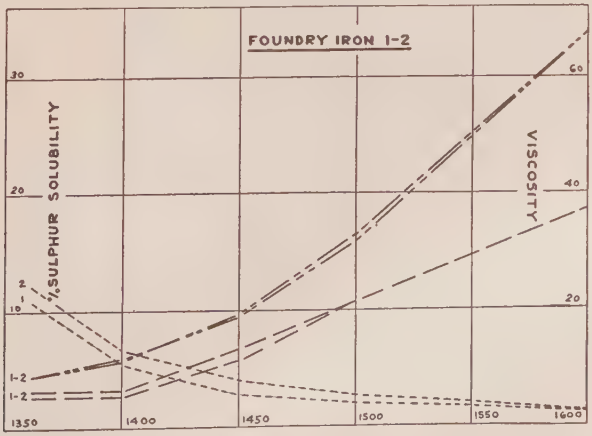

FIG. 19.—SULFUR SOLUBILITY AND VISCOSITY OF SLAGS (FROM DIAGRAMS) OF FURNACES PRODUCING FOUNDRY IRONS.

Feild and Royster[15] give an average slag composition used in the United States and analyses of slags from furnaces producing basic and foundry iron. The sulfur solubilities and viscosities of these slags, calculated on the basis of SiO_2–CaO–Al_2O_3 = 100 per cent., and taken from our diagrams, Figs. 4–9, are given in Fig. 17. Similar data from furnaces running on basic, foundry, bessemer, and special grades of iron are given in Figs. 18 to 21, the sulfur solubilities and viscosities being taken from the diagrams given here. The analyses of these slags and the corresponding iron analysis are summarized in Table 9.

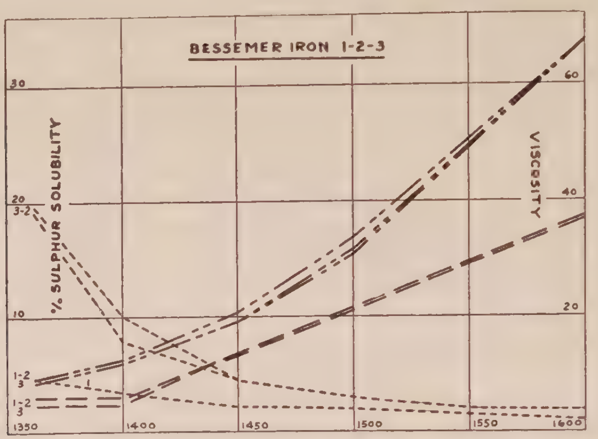

FIG. 20.—SULFUR SOLUBILITY AND VISCOSITY OF SLAGS OF FURNACES PRODUCING BESSEMER IRONS.

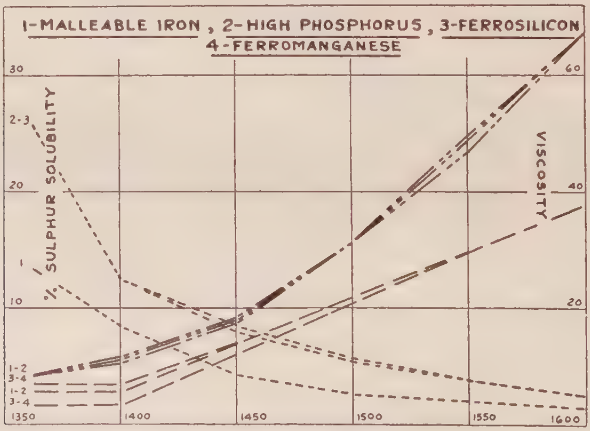

FIG. 21.—SULFUR SOLUBILITY AND VISCOSITY OF SLAGS OF FURNACES PRODUCING SPECIAL IRONS.

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[15] Bureau of Mines, *Tech. Paper* No. 187.

TABLE 9.—*Slag and Iron Analyses*

Kind	Fig.	Slag									Iron				
		CaO, Per Cent.	Al_2O_3, Per Cent.	SiO_2, Per Cent.	MgO, Per Cent.	MnO, Per Cent.	FeO, Per Cent.	Fe, Per Cent.	TiO_2, Per Cent.	CaS, Per Cent.	C, Per Cent.	Mn, Per Cent.	P, Per Cent.	S, Per Cent.	Si, Per Cent.
1-Basic	18	44.21	14.38	34.11	2.94			1.04		4.10	4.06	1.25	0.190	0.046	1.45
2-Basic	18	44.18	14.63	34.12	2.86			0.94		3.85	4.10	1.08	0.183	0.043	1.49
3-Basic	17, 18	44.34	11.34	35.97	2.90	0.66	0.43		0.25	3.93					
4-Basic	18	42.42	13.53	38.54	1.21	1.39				3.26		1.29	0.153	0.033	1.13
1-Foundry	19	43.85	14.51	36.48	1.53	0.49				3.33		0.65	0.364	0.030	1.99
2-Foundry	17, 19	37.90	15.25	35.89	4.50	0.58	1.09		0.44	3.20					
1-Bessemer	20	42.57	13.94	39.12	1.25	0.53				3.03		0.53	0.083	0.030	1.61
2-Bessemer	20	44.33	13.89	34.48	3.09			0.87		4.01	4.17	0.046	0.091	0.043	1.80
3-Bessemer	20	44.74	13.43	34.54	2.84			0.99		4.16	4.14	0.50	0.089	0.035	1.87
1-Malleable	21	44.14	15.64	34.89	1.72	0.69				3.96		0.68	0.163	0.038	1.88
2-High phosphorus	21	37.96	17.30	33.74	6.32	0.34				3.66		0.64	0.664	0.030	1.87
3-Ferrosilicon	21	34.72	16.22	32.34	4.41	3.31				8.26		12.47	0.065	0.016	9.37
4-Ferromanganese	21	46.29	13.24	26.78	3.70	5.03				6.61		78.26	0.122		1.69

The general solubility curves, Figs. 4 to 9, inclusive, also give the variation in composition of slags that have equal sulfur solubilities and viscosities. For example, the solubility curve for 1400° C., Fig. 5 shows that slag Al_2O_3 15.5 per cent., SiO_2 40 per cent., CaO 44.5 per cent., has the same viscosity and calcium-sulfide solubility at 1400° C. as slag Al_2O_3 11.0 per cent., SiO_2 46.0 per cent., CaO 43.0 per cent.; and that slag Al_2O_3 14 per cent., SiO_2 39 per cent., CaO 47 per cent. has the same viscosity and manganese-sulfide solubility as slag Al_2O_3 16 per cent., SiO_2 46.0 per cent., CaO 38 per cent. Again, on the 1550° C. curve, Fig. 8, the slag Al_2O_3 9 per cent., SiO_2 45 per cent., CaO 46.0 per cent. has the same viscosity and manganese-sulfide solubility as has Al_2O_3 17 per cent., SiO_2 47 per cent., CaO 36 per cent.; and while their viscosities and manganese-sulfide solubilities are the same, the curve Fig. 8 shows their calcium-sulfide solubilities are nearly equal.

When using the solubility curves, the fact must be kept in mind that they represent conditions when equilibrium is attained at a definite temperature; they indicate the maximum amount of sulfur that will dissolve. However, when two slags have different sulfur solubilities, the one giving the greater solubility, other things remaining equal, will be the better desulfurizer. Because of the much greater solubility of sulfur when it is in the form of manganese sulfide than when calcium sulfide, it would be desirable to cause reactions forming manganese sulfide to take place in the blast furnace. At the present time, however, the factors controlling this reaction are not known, but an attractive field for research to determine them is opened up.

The desulfurizing reactions take place in the bosh and hearth of the blast furnace. The carburized iron melts and the calcined limestone, the gangue of the ore, and the coke ash fuse together to form slag. When the iron and slag become sufficiently fluid, they trickle down over the incandescent coke of the tuyere region. The slag is less fluid than the iron, so that the slag will hinder the drops of iron in their descent through the bosh, and in this way increase the contact time between the drops of iron and the slag globules. This increased contact time will promote greater desulfurization than a shorter contact time; so, as Feild and Royster point out, a slag that has a great viscosity change through a given temperature range is the best desulfurizer for the part of the sulfur that is removed before the hearth is reached.

When the iron and the slag reach the hearth of the furnace, they separate and the contact surface between the molten iron and the molten slag is equal to the area of the hearth. The iron and slag are at high temperatures, their viscosities are low, and the sulfur solubility of the slag is high. On account of the lower viscosity of the slag in the hearth, diffusion through its mass is more rapid than if it were more viscous; and if the surface layer of slag in contact with the iron should tend to approach saturation, the diffusion facilitated by the low viscosity will tend to equalize the slag composition and permit more rapid desulfurization. In the hearth, too, the globules of iron dropping down from the bosh pass through the slag layer in their descent into the iron layer, so that the actual surface on which partition of the sulfur will adjust itself is greater than the hearth area alone.

In the bosh, while the iron and slag globules are descending, the temperature of the iron and slag will be lower and the sulfur solubility of the slag will be lower; while the area of contact of the particles will be large, the time of that contact will be short. In the hearth, the temperature of the metal and slag is higher and the sulfur solubility of the slag will be greater; while the area of contact of slag and metal will be less, the time of contact will be much longer.

The desulfurizing that takes place in the hearth is very important. Some laboratory work on the partition coefficient of sulfur between slag and metal indicates that the time factor is of the greatest consequence.[16] Some slag analyses, Tables 10 and 11[17] confirm this view.

The slags reported were sampled at the flushes and at the cast. These analyses show a progressive increase in the sulfur content, the first flush

TABLE 10.—*Analyses of Slag from Furnace Producing Basic Iron*

Slag at	SiO_2, Per Cent.	Al_2O_3, Per Cent.	CaO, Per Cent.	MgO, Per Cent.	FeO, Per Cent.	TiO_2, Per Cent.	MnO, Per Cent.	CaS, Per Cent.
2:00 p.m. flush								
start	37.56	12.40	43.91	2.01	0.45	0.26	0.93	2.73
middle	33.87	11.40	46.10	2.94	0.52	0.27	0.83	3.53
end	34.93	11.47	45.21	2.96	0.55	0.26	0.57	3.89
3:00 p.m. flush								
start	36.30	12.94	43.33	3.08	0.31	0.22	0.55	4.00
middle	36.48	11.51	43.76	2.68	0.39	0.30	0.60	4.09
end	36.40	9.98	45.43	3.07	0.26	0.30	0.74	3.30
3:50 p.m. flush								
start	34.05	10.63	44.88	3.24	0.81	0.22	0.68	4.50
middle	33.76	11.44	45.06	2.95	0.77	0.26	0.73	4.55
end	33.46	11.04	45.52	3.35	0.75	0.23	0.68	4.57
4:00 p.m. cast								
start	37.54	11.26	42.76	3.55	0.13	0.18	0.66	3.85
middle	38.48	11.32	43.13	1.62	0.06	0.32	0.36	4.06
end	38.76	10.66	42.99	3.35	0.13	0.21	0.60	4.07
Average	35.97	11.34	44.34	2.90	0.43	0.25	0.66	3.93

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[16] Unpublished Thesis, Flom and Lueck, Univ. of Wis. (1922).
Unpublished Thesis, Hahn and Lundberg, Univ. of Wis. (1922).

[17] Feild and Royster: Bureau of Mines, *Tech. Paper* No. 187, 25–26.

TABLE 11.—*Analyses of Slag from Furnaces Producing Foundry Iron*

Slag at	SiO_2, Per Cent.	Al_2O_3, Per Cent.	CaO, Per Cent.	MgO, Per Cent.	FeO, Per Cent.	TiO_2, Per Cent.	MnO, Per Cent.	CaS, Per Cent.
9:25 a.m. flush								
start	37.40	16.85	33.73	4.73	2.89	0.45	1.05	2.59
middle	38.10	15.35	34.60	4.56	2.54	0.48	0.96	3.06
end	37.50	14.70	38.08	4.13	1.96	0.54	0.87	2.11
10:10 a.m. flush								
start	37.80	15.25	36.70	4.99	0.58	0.45	0.34	2.86
middle	37.97	14.25	37.81	4.47	0.78	0.50	0.56	2.99
end	37.22	14.90	37.33	5.05	0.60	0.42	0.49	3.18
11:00 a.m. flush								
start	36.74	15.38	39.07	4.00	0.54	0.44	0.35	3.26
middle	35.82	14.64	37.92	5.10	0.56	0.52	0.42	3.22
end	34.37	14.18	41.36	4.45	0.74	0.60	0.43	3.18
11:55 a.m. cast								
start	34.13	16.29	38.90	4.34	0.54	0.45	0.46	3.80
middle	34.38	15.40	40.94	3.65	0.65	0.32	0.48	3.28
end	34.24	15.85	38.37	4.86	0.68	0.30	0.41	3.87
Average	35.89	15.25	37.90	4.50	1.09	0.44	0.58	3.20

being lowest in sulfur and the last flush and the cast highest in sulfur. Figuring the increase of sulfur in each cast over the first flush and assuming the same rate for all the time between casts, this increased sulfur content will account for the greater part of the sulfur charged into the furnace, assuming the furnace to be operating on the charge that would be usual when making basic and foundry irons, respectively. Any desulfurization in the bosh may be assumed to proceed at a regular rate so that an increased sulfur content for the slag tapped from the hearth would indicate the amount and intensity of the desulfurization taking place in the hearth. As already pointed out, because of the increased solubility for sulfur when the slag is hottest and because its lowered viscosity permits more rapid diffusion, the greatest part of the desulfurizing action takes place in the hearth. This would indicate that the fewer slag flushes made between casts the better for desulfurization, because a larger amount of slag of lower sulfur content would then be in contact with the molten metal and sulfur partition between the metal and the slag will be more rapid and complete.

To continue the study of the desulfurizing power of blast-furnace slags, work should be carried out on the solubility of iron sulfide, on the effect of replacing lime with magnesia, on the velocity of the reactions affecting the formation of sulfide compounds, and on the partition coefficients between irons and slags at various temperatures. It is along these lines that we hope to continue this work.

SUMMARY OF RESULTS

1. The effect of alumina in blast-furnace slags is to increase the solubilities of the slags for calcium sulfide and to diminish the solubilities of the slags for manganese sulfide.
2. The effect of high temperatures in increasing the solubilities of the slags for both calcium and manganese sulfides is very marked.
3. The actual chemical composition of the slag, as affecting the solubilities of calcium and manganese sulfides, is of less importance than the temperature of the slag.
4. The change in calcium- and manganese-sulfide solubilities is probably less influenced by the basicity of the slag chemically than it is by the indirect action of the more basic slag in increasing the hearth temperature.
5. The region in which solution of calcium and manganese sulfides in the slag is most rapid is the hearth of the blast furnace.

DISCUSSION

A. L. FEILD, Long Island City, N. Y. (written discussion).—The authors have done much to clarify our ideas on the chemical processes of sulfur absorption and its partition between slag and iron in the blast furnace. They have correlated their own results with the previous work of the Geophysical Laboratory and the Bureau of Mines in an admirable manner.

The authors have shown that the effect of temperature largely overshadows the effect of changes in slag composition on sulfide solubility. Hence any objection to their assumption that the solvent capacity of a slag is a strictly additive property of its mineralogical constituents loses much of its force from the practical standpoint. Several solubility determinations, however, made with mixtures of two of the constituents calcium orthosilicate, anorthite, and gehlenite in selected proportions would clear up this point definitely and remove any possible objection to the authors' method of calculation in arriving at the curves of Figs. 15 to 21. There are instances, of course, where a mixure of two solvents, chemically inactive toward each other, possess a much greater solvent power than either solvent alone; such, for instance, is the action of a mixture of alcohol and ether upon nitrocellulose.

The experimental determination of how much sulfur is taken up by the slag in the bosh smelting zone and how much in the hearth remains a very important question, as already pointed out by Royster and the writer. A continuance of their work by the authors along the general lines they indicate would give the desired answer and greatly enlarge the sphere of practical utility of both slag-viscosity and sulfide-solubility data. The authors have deduced, from the variation in sulfide content of slag between casts and the observed rapid increase in solvent power of slag with temperature, that the major portion of desulfurization occurs in the hearth. If this is really the case, and it appears to be so, the quantitative forecast of sulfur elimination on the blast furnace from a knowledge of slag-viscosity and sulfide-solubility data is to be expected in the near future. The work remaining to be done is concerned chiefly with rates of diffusion and the areas of contact between metal and slag in the hearth.

Use of Magnetic Ore in the Blast Furnace

By G. P. Pilling,* Philadelphia, Pa.

(New York Meeting, February, 1923)

The use of magnetic ore in the blast furnace is a subject of increasing importance. The end of the deposits of lake ore is in sight, although not imminent, and unless some new field is discovered, the use of magnetites will be an increasing factor in the iron production of the country. Enormous deposits of magnetic ore in the country that have been practically untouched contain ore of sufficient richness and purity to prolong materially our ore supply without extensive importation of foreign ores.

Magnetic ores can be divided into three general classes: Fine unsintered, or raw ore; raw lumpy ore; fine ore that has been sintered.

Fine Unsintered Ore

Typical analysis and screen test of this class of ore from the Adirondack region is as follows:

	Per Cent.
Fe	58.00
SiO_2	7.50
Al_2O_3	1.60
Bases	2.77
P	1.00
Mn	0.13
S	Trace

Screen Test	Per Cent.
Remaining on 4 mesh	7.50
Through 4 on 8 mesh	13.75
8 on 10 mesh	3.75
10 on 20 mesh	22.50
20 on 40 mesh	27.50
40 on 60 mesh	2.50
60 on 80 mesh	10.00
80 on 100 mesh	2.50
100 mesh	10.00
	100.00

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* Manager Eastern Office Freyn, Brassert & Co. Engineers.

OTHER ANALYSES AT 212° F.

		PER CENT.		PER CENT.
Fe		63.00		63.80
SiO_2		6.00		8.20
Al_2O_3		1.33		1.63
Bases		3.50		0.90
CaO	3.08		0.50	
MgO	0.43		0.40	
P		0.70		0.021
Mn		0.03		0.03
S		Trace		Trace
Titanium		Trace		1.05

As none of these ores contain any appreciable amount of manganese, it is necessary to add to the ore mixture, material containing this element. The material is usually a high manganese ore from the lake region; very high imported manganese ore; or, where it is available, the cinder from a furnace producing ferromanganese.

The successful use of fine magnetic ore in the blast furnace depends on a number of factors, all of which must be carefully determined for each individual case. However, there are a number of basic principles, which are briefly discussed here.

Correct Furnace Design

The first thing to consider when designing a furnace is that the reduction of this ore requires from 3 to 4 hr. longer than an equal amount of hematite. The working volume above the tuyeres must therefore be larger. It has been found that approximately 4500 cu. ft. working volume above the tuyeres is required for each 100 tons of iron produced in 24 hours.

The next thing to be considered is the design of the bosh. Unless every precaution is taken, the fine magnetic ores tend to work in a high zone of fusion in the furnace, thereby raising the ratio of direct to indirect reduction, which of course tends to high fuel and poor furnace results. The height of the bosh above the center lines of the tuyeres must, therefore, be kept as low as possible, not over 12 ft. and preferably lower than this. In a new furnace, the bosh angle should be not less than 78°, but if an old plant is being remodeled, the bosh should be kept low, even if necessary to flatten it somewhat.

The next point for consideration is the batter of the inwall. This is governed by three considerations: (1) These fine magnetic ores, in their passage through the furnace, are practically free from swelling due to carbon deposition. (2) These ores, when distributed from the bell on to the stock, will travel down through the furnace in a practically straight line and will not move laterally to any extent. (3) The tendency is, at all times, for the ore to run ahead of the rest of the charge, thereby arriving at the zone of fusion in an unreduced state. The batter of the inwall should therefore be between 0.55 and 0.7 in. per ft. Anything greater than this allows the charge to loosen up too rapidly, thereby permitting the ore to run ahead; the ore has a tendency to work its way down through the center of the charge in a column approximately the diameter of the stock line, leaving an annular ring of open material against the walls of the furnace. Because of the difficulty of distributing an even layer of ore in the furnace, the stock line must not be too large: neither must it be too small or the flue-dust loss will be excessive. The stock-line diameter, however, is more or less fixed, being determined by the other dimensions of the furnace.

Stock Distribution

One of the greatest difficulties met with in working fine magnetic ores is the correct stock distribution. It is necessary to procure almost theoretically perfect distribution of the ore from the bell on to the stock, for after the ore has been placed on the top of the charge, it has practically no lateral movement, *i.e.*, the ore will not work from the outside of the furnace toward the center or vice versa. This means that the size of the coke unit, which necessarily governs the size of the ore unit, must be carefully studied and regulated; it is considerably larger than the coke unit used with hematite ores. For instance, a furnace having a stock-line diameter of 12 ft. should have a coke unit of not less than 7000 lb. and a furnace with a stock line of 15 ft. should have a coke unit of about 10,000 lb. It has been found that the best results are obtained by filling the ore and coke separately from the large bell into the furnace. The typical round for a furnace with a 12-ft. stock line on this class of ores would be as follows: Place 7200 lb. of coke and all the limestone on the bell and lower into furnace; place all the ore on the bell and lower into furnace. The mixed or stratified charge does not seem to give good results. The reason for this round is that to distribute the ore evenly over the surface of the proceeding charge, advantage must be taken of the rebound of the ore from the walls to place the correct proportion of ore at the center of the furnace.

It is common practice, when using fine magnetic ores, to use a 45° angle bell and not over 2 ft. clearance between the bell and the stock line.

Time Required for Reduction, or Rate of Driving

On account of the hard, dense nature of fine magnetic ore, it is necessary to allow not less than 11 hr. and preferably 12 to 14 hr. for reduction. If the furnace is driven faster than this, in from 48 to 60 hr., it will start to work cold and unreduced ore will appear in front of the tuyeres, the slag will become black with small particles of ore floating on the top, the silicon in the iron will be lowered, and the sulfur in the iron will rise very rapidly.

Air Distribution

When working these fine magnetic ores, care must be taken not to have the tuyere area too large. At a number of plants, the best work is being done with about 130 to 150 cu. ft. of air per minute engine displacement per square inch of tuyere area. In one case, 30,000 cu. ft. of air per minute was blown through twelve 5½ in. tuyeres, but the results. both as to tonnage and fuel consumption, were far from satisfactory. After the tuyeres were changed to 4½ in. the fuel consumption was reduced by about 100 to 150 lb. per ton of iron, and the product increased 20 tons per day.

It is not practicable to blow as much air when smelting fine magnetic ores as when smelting an equivalent hematite ore. If the furnace is overblown, the zone of fusion tends to rise, thereby increasing the percentage of direct reduction, the ore comes through in an unreduced state, and the top heats immediately rise above the level for most economical operation.

Flux

Most furnaces using these magnetic ores use a calcite stone when it is available, although very good work is being done with a low magnesia dolomite. All the stone used for flux should be crushed to not larger than 4 in. and all fines under ¾ in. screened out.

Selection of Coke

It is very important, when working magnetic ores, as well as all other ores, that the coke used burn freely and quickly at the tuyeres, thereby keeping the zone of fusion as low as possible; at the same time it should be dense enough so that there will not be an excess of loss by solution and direct reduction by carbon. Ordinarily, a coke of fairly high sulfur content can be used, inasmuch as practically none of the magnetic concentrates used in the raw state carry any appreciable percentage of sulfur.

Furnace Operation and Results

The fuel consumption, when using 100 per cent. of this grade of ore, is uniformly higher than that of hematite or sintered magnetic ore. A fair average of the work of a furnace using 100 per cent. of this fine magnetic ore will probably be about as follows:

FOUNDRY IRON

Silicon, 2.25 to 2.75 per cent	2400 to 2500 lb. of coke per ton iron
1.75 to 2.25 per cent	2250 to 2300 lb. of coke per ton iron

BASIC

Silicon, under 1.00 per cent	2000 to 2100 lb. of coke per ton iron

The iron produced is usually quite low in sulfur. If it is necessary to produce a large tonnage of high silicon iron, the best results are obtained by using a small percentage of siliceous hematite ore in the mixture, as it is extremely difficult to produce high silicon iron using 100 per cent. of fine magnetic ore. Gravel or other siliceous material may be used but do not give up their silica as easily for reduction into the iron.

Slags

On account of the absence of sulfur in these ores, it is possible to run uniformly high acids in the slag. Some plants are being successfully operated on an average of 50 to 51 per cent. total acids in the slag with the sulfur in the iron well under 0.030 per cent.

Furnace Irregularities

Furnace operation on this class of ore is practically entirely free from the troubles resulting from hanging and slipping, such as are encountered when working hematities. However, the furnace appears to be extremely sensitive to every minor change in stove heat, atmospheric condition, and particularly any irregularity of stock distribution. Very careful supervision is therefore necessary, as a very slight irregularity in the furnace operation is accompanied by serious trouble, which appears with unusual swiftness and practically without warning.

Conclusion

Typical results of a small furnace using 100 per cent. fine magnetic concentrates are as follows:

FURNACE LINES

Hearth diameter, feet	12
Bosh diameter, feet	16
Stock-line diameter, feet	12
Height, center line of notch to top platform, feet	80
Inwall batter, inch per foot	0.6
Bell diameter, feet	8
Bell angle	45°

IRON ANALYSIS

Silicon, per cent	1.75 to 2.25
Daily product, tons	210 to 220
Coke consumption, pounds per ton of iron	2350
Air blown, cubic feet per minute	19,000
Flue dust produced, tons per day	15
Slag volume (calculated) pounds per ton of iron	1100
Average stove heats, degrees	1100
Average top heats, degrees	450

RAW LUMPY MAGNETIC ORE

The writer is not familiar with any furnaces operating on 100 per cent. raw lumpy magnetic and trusts that this practice will be brought out in the discussion of this paper. The plant here described is operating in the following ore mixture:

20 per cent. ore A, lumpy magnetite (magnetic ore).
20 per cent. ore B, magnetic concentrates (magnetic ore).
40 per cent. Old Range (Lake ore).
20 per cent. Central Western (Lake ore).

	ORE A PER CENT.	ORE B PER CENT.		CENTRAL WESTERN (NATURAL) PER CENT.
Fe	56.55	51.85	62.65	49.96
SiO_2	11.82	4.29	10.62	6.03
Al_2O_3	1.59	1.19	1.26	2.33
CaO	2.28	0.15	1.18	0.32
MgO	1.25	0.54	0.51	0.22
P	0.028	0.066	0.094	0.073
Mn	0.09	0.40	0.03	0.89
S	0.694	0.009		0.011
Ignition loss	0.659	3.83		5.87
Moisture	1.35	15.00		12.50

SCREEN TESTS

	ORE A PER CENT.	ORE B PER CENT.
Remaining on 4 mesh	7.40	7.50
Through 4 on 10 mesh	9.03	13.75
10 on 20 mesh	7.03	3.75
20 on 30 mesh	1.55	22.50
30 on 40 mesh	4.60	27.50
40 on 60 mesh	2.61	2.50
60 on 80 mesh	2.86	10.00
80 on 100 mesh	0.11	2.50
100 mesh	2.29	10.00
		100.00

The average iron in the above mixture is 53 per cent.

Furnace Design

Inasmuch as these lumpy magnetic ores are used with a large percentage of hematite ores, the furnace design is naturally a compromise. The inwall batter is made slightly larger than when using 100 per cent. magnetic ores, to allow for the swelling action of the hematite ore. Typical lines for a large furnace using the above ore mixture are as follows:

Hearth diameter	18 ft. 5 in.	Bell diameter	12 ft.
Bosh diameter	22 ft. 0 in.	Bell angle	45°
Bosh angle	82° 2′	Total height of furnace	93 ft. 6¾ in.
Stock-line diameter	16 ft. 6 in.	Height of furnace from center line of iron notch to top platform	91 ft. 2¾ in.
Height of bosh	10 ft. 11 in.		
Inwall batter	0.75 in. per ft.		

Stock Distribution

The problem of stock distribution, when using lumpy magnetic ores mixed with hematite ores, is not so difficult as with 100 per cent. fine magnetic concentrates, for these lumpy magnetic ores do not show the same tendency to run ahead in the furnace, particularly when used with a large percentage of finer hematites. As with the finer magnetic concentrates, however, the ore and the coke should be lowered into the furnace separately from the large bell. A typical round of the furnace under discussion is as follows: Place 10,000 lb. of coke on large bell and lower same into the furnace. Place 16,000 lb. of ore and 4400 lb. of limestone on large bell and lower same into the furnace. The mixed or stratified charge does not seem to give good results. It is also the best practice, when using these lumpy magnetic ores, to use a 45° angle bell and approximately 2 ft. clearance between the bell and the stock line.

Time Required for Reduction, or Rate of Driving

On account of their hard dense nature, these lumpy magnetic ores require as long or longer for reduction in the furnace as the fine magnetic concentrates; from 12 to 14 hr. are required for thorough reduction. This means that the advantage of the rapid driving of hematite ores is lost and the furnace must be driven at a rate that will give thorough reduction of the hard ore. If the furnace is driven faster than this, the ore comes down to the zone of fusion unprepared, and raw ore appears in front of the tuyeres, resulting in a cold hearth and off-grade iron.

Air Distribution

The same practice holds good for air distribution and tuyere area on the lumpy magnetic ores as on the fine magnetic ores, although good work is being done with a somewhat lower wind velocity through the tuyeres than on fine magnetic concentrates. The average seems to be about 130 cu. ft. of air per minute engine displacement per square inch of tuyere area. On this furnace with a hearth diameter of 18 ft. 6 in. from 38,000 to 40,000 cu. ft. of air per minute engine displacement, corrected to 62° F., are blown through ten 6 by 12 in. tuyeres. About 5 to 10 per cent. less air is required for these lumpy ores when an equivalent hematite ore is used. If the furnace is overblown, the zone of fusion tends to rise, thereby increasing the percentage of direct reduction; the ore comes down in an unreduced state and the top heats immediately rise above the level for most economical operation.

Furnace Operation and Results

The fuel consumption, when using the ore mixture given here, is uniformly higher than when using hematite ore or sintered magnetic ore and somewhat higher than when using fine magnetic concentrates. Using only 20 per cent. lumpy and 20 per cent. fine magnetites and making iron with 1.75 to 2.75 per cent. silicon, requires 2300 lb. of coke per ton of pig iron; this is with 12 per cent. scrap in the mixture. If the percentage of hard lumpy ore in the mixture is increased from 20 to 50 per cent., the fuel consumption is increased about 10 to 20 per cent. and the tonnage is lowered about 10 to 20 per cent.

Slag

Typical slag analysis of a furnace running on these ores is: Silica 35 per cent., alumina 13 per cent. The slag volume carried is 1300 lb. per ton of pig iron.

Furnace Irregularities

Furnace operation when using a large percentage of lumpy magnetic ores is practically entirely free from the troubles resulting from hanging and slipping, such as are encountered when working 100 per cent. hematites. The furnace does not seem to be nearly so sensitive to minor changes as when using 100 per cent. fine magnetic ores, although it is extremely sensitive to any change or irregularity in stock distribution.

Fine Magnetic Ore That Has Been Sintered

There is no furnace, as far as is known at the present time, operating on 100 per cent. sintered magnetic ore. The closest approach to it is a small furnace working an ore burden of 80 to 85 per cent. sintered fine magnetic ore, 10 to 15 per cent. mixed puddle and heating cinder, and 5 per cent. hematite ore. The typical analysis of the sintered magnetic ore is: Iron 55 per cent., silica 10.4 per cent., alumina 1.69 per cent., bases 6.50 per cent., phosphorus 0.075 per cent., manganese 0.18 per cent., sulfur 0.18 per cent. No screen test is available, but the average would probably be from lumps the size of a man's two fists down to ⅛ inch.

Preparation of Ore

The magnetic ore under discussion (eastern Pennsylvania), as mined, contains about 53 per cent. of iron and 3 per cent. of sulfur. The ore is then crushed by jaw crushers and rolls to pass through a ⅜ in. round hole. It is then mixed in a pug mill, with flue-dust or fine coal, and sintered on a Dwight-Lloyd sintering machine. The resulting sinter contains about 0.18 per cent. sulfur and physically is in ideal shape for blast-furnace use. The sinter is extremely open and porous and therefore exposes a maximum surface to the blast furnace gases for reduction. Also a considerable percentage of the magnetite has been reduced to hematite by the sintering process.

Furnace Design

The furnace to be discussed has the following lines:

Hearth diameter	12 ft. 0 in.
Bosh diameter	17 ft. 1 in.
Bosh angle	76° 30′
Bosh height	12 ft. 1¼ in.
Stock-line diameter	12 ft.
Bell diameter	8 ft.
Bell angle	45°
Inwall batter	0.78 in. per ft.
Total height	75 ft. 0 in.
Number of tuyeres	12

The bosh angle is rather flatter than for most modern furnaces, as this is a remodeled furnace; it was the steepest bosh that could be procured unless the entire lower part of the furnace was rebuilt.

The inwall batter is greater than for either raw fine magnetic concentrates or lumpy magnetic ore. It has been found that sintered magnetic ore apparently works in the furnace in a manner about half way between raw magnetic concentrates and hematite ore, and therefore allowance must be made for this in the furnace design. That these sintered magnetic concentrates will not work successfully on a smaller inwall batter than this has not been demonstrated; although three furnaces have been built to use this class ore with an inwall batter of less than 0.6 in. to the foot, none of these had been operated when this was written.

Stock Distribution

The problem of stock distribution with sintered magnetic concentrates does not seem to be as difficult as when lumpy or fine unsintered magnetic ore is used, although the basic principles are the same. Owing to the large, lumpy coarse nature of the sintered ore, there is no tendency for it to run ahead in the furnace. But practice has shown that best results are obtained, as with the other two classes of ore, by lowering the ore and the coke into the furnace separately. The typical round for the furnace under discussion is: Place 8000 lb. coke and all the limestone on large bell and lower into furnace; place all the ore on large bell and lower into furnace. The mixed or stratified charge does not seem to give quite as good results, although it can be used, if necessary or desirable.

Time Required for Reduction, or Rate of Driving

One of the greatest advantages of using sintered magnetic ore is that the rate of driving can be materially increased. Because of the open nature of the sintered ore, a large surface is exposed to the action of the blast-furnace gases and reduction proceeds rapidly. It is perfectly feasible to procure thorough reduction of sintered magnetic ore in the furnace in about 9 to 10 hr., which is about 3.5 hr. less than the time required for raw magnetic ore. This, of course, is a big advantage and increases the production to a considerable extent.

Air Distribution

As for the raw magnetic ores, about 150 cu. ft. per min. engine displacement of air per square inch of tuyere area give the best results. The flue-dust losses, on sintered ore, are very low so that it is possible to blow more air than with fine magnetic concentrates. This increased volume of air does not make the furnace drive too fast for proper reduction, inasmuch as the time required for reduction is much less.

Furnace Operation

Sintered magnetic ore can be successfully mixed in any proportion with hematite ores and good results obtained. Mixtures have been used ranging from 20 per cent. sintered ore and 80 per cent. hematite to 80 per cent. sintered ore and 20 per cent. hematite. The fuel required per ton of pig iron, when using a large percentage of sintered magnetic ore, is much lower than when using raw magnetic ore. Results seem to indicate that iron can be made with sintered ore with approximately 150 to 250 lb. less coke per ton of pig iron than when using raw magnetic ore. The furnace operation is exceedingly smooth and regular; the blast pressure uniformly low and even; and the furnace settles regularly with practically no trouble from hanging and sticking. Very high blast heats can be carried without difficulty, thereby effecting further saving in fuel.

Slag

Because of the fairly high sulfur content of the sintered ore, which comprises 85 per cent. of the mixture, it is necessary to carry a large slag volume. About 1500 to 1600 lb. of slag per ton of pig iron is required. If the total acids in the slag rise much above 48 per cent., some difficulty is encountered in eliminating the sulfur. This large slag volume, of course, is not required when using an ore with no sulfur content.

Furnace Irregularities

A furnace operating on a large percentage of sintered magnetic concentrates seems to be freer from furnace troubles and irregularities than any other kind of ore. There is practically no trouble from hanging or sticking and the furnace does not seem to be sensitive to changes in atmospheric conditions or other minor irregularities; in fact, it is unusual to have any trouble, and there have been occasions when the furnace has run for months with no irregularity or trouble of any kind whatever developing.

Conclusions

From the results obtained on this furnace using this large percentage of sintered magnetic ore, it has been demonstrated that the fuel consumption is from 150 to 300 lb. lower and the product about 30 tons per day higher than could be expected from an equivalent size furnace using the same proportion of either raw fine magnetic concentrates or raw lumpy magnetic ore. The operation of the furnace is greatly simplified and the results obtained are most satisfactory from every standpoint.

The results obtained at this furnace on basic iron during the month of February, 1922, are as follows. The furnace was blown in Dec. 26, 1921, and operated on foundry iron until Feb. 1, when it was turned on to basic. The burden was:

BURDEN

Coke	8000 lb.
Ore	14,500 lb.
Stone	5300 lb.
Borings, turnings	15 per cent. of ore weight

OTHER DATA

Air per minute, average	20,750 cu. ft.
Stove heats, average	1300° F.
Blast pressure, average	13 lb.
Top heats, average	400° F.

IRON ANALYSIS

	PER CENT.
Silicon, average	0.80
Sulfur, average	0.032
Phosphorus, average	0.50
Manganese average	1.60

SLAG ANALYSIS

	PER CENT.
Silica	34
Alumina	12.50

The furnace averaged 266 tons of iron per day on 1832 lb. of coke per ton of pig iron. The 1832 lb. of coke per ton was based on railroad weights, which includes all the fine coke screened out at the bins, so that the actual fuel consumption was somewhat lower. The results for the last two weeks of February after the furnace had settled down on to basic iron were: Average production per day 287.7 tons of iron on 1719 lb. of coke per ton of pig iron.

Grade	Start Date	Stop Date	Average Tons	Average Coke	Mixture	Per Cent.
1X and 2X	1/ 1/22	2/ 3/22	201	2220	Sinter	75
					Lake	10
					Puddle	15
					Turnings	10
Basic	2/ 4/22	3/ 8/22	274	1800	Sinter	80
					Lake	5
					Puddle	5
					Heating	10
					Scrap	15
Malleable	3/ 9/22	3/14/22	254	2040	Sinter	80
					Lake	20
					Turnings	10
1X and 2X	3/15/22	4/12/22	227	2120	Sinter	82
					Lake	12
					Puddle	6
					Turnings	10
Malleable	4/13/22	4/16/22	234	2000	Sinter	80
					Lake	5
					Heating	15
					Turnings	10
2 plain	4/17/22	4/27/22	233	2080	Sinter	82
					Lake	12
					Puddle	6
					Turnings	10
Basic	4/28/22	5/ 5/22	257	1876	Sinter	82
					Heating	15
					Lake	3
					Turnings	15

SUMMARY

Of the three ores here mentioned, the least desirable is the lumpy or coarse magnetic ore. The raw fine magnetic ore is slightly better, provided the furnace design is correct and careful study is given to its use. The operating difficulties with the fine unsintered magnetic ore are very great, as any small change or irregularity in furnace operation quickly results in serious trouble, so that constant attention is required for its successful use.

The sintered fine magnetic ores are by far the best of the three for blast-furnace use. Sintering does away with furnace trouble and irregularity almost entirely. It permits faster driving of the furnace and results in considerably lower fuel consumption. This is somewhat offset by the cost of sintering—about 80 cents to $1.00 per ton of sinter. Of course, where the raw ore contains an appreciable amount of sulfur, sintering, roasting or nodulizing is absolutely necessary before the ore can be used in the furnace.

In some cases, very poor results have been obtained by attempting to use a mixture of raw magnetic ore and hematite ore; these were probably due to the fact that this mixture was used on furnaces designed to operate on all-hematite ore. In addition, in operating the furnace, the point was probably overlooked that it was necessary to handle the operation as though the entire mixture was magnetic ore, allowing the hematite ore to take care of itself. This means that the furnace must be driven at a rate that will allow plenty of time for the reduction of the magnetic ore, even though the hematite does not require so long.

The entire problem of working magnetic ore in any form requires a great deal of study and care to bring all the elements into thorough accord.

Review of Present Status of Drill Steel Breakage and Heat Treatment

By Francis B. Foley,* Charles Y. Clayton,† Rolla, Mo., and Henry S. Burnholz,‡ New York, N. Y.

(New York Meeting, February, 1923)

FOREWORD

This work was first undertaken for the U. S. Bureau of Mines, in 1919–20, by C. E. Julihn, superintendent of the station at Minneapolis. Learning of the interest, in this subject, of B. F. Tillson, of the New Jersey Zinc Co., Mr. Julihn and Mr. Foley went to Franklin, N. J., and discussed the matter with him. A conference in Chicago, in November, 1920, was attended by Dr. F. G. Cottrell, who was then Director of the Bureau of Mines, and representatives of the Bureau of Mines and of the Missouri School of Mines and Metallurgy; at this time a coöperative plan of work was agreed upon by the Bureau of Mines and the Missouri School of Mines.

In February, 1921, Mr. Tillson arranged a conference on this subject at the time of the annual meeting of the American Institute of Mining and Metallurgical Engineers, at which several technical papers were presented. In a paper, the Breakage and Heat Treatment of Rock Drill Steels, he suggested a program that might be carried out in connection with the investigation of the problem. At this conference Mr. Manning, former Director of the Bureau of Mines, suggested the formation of an advisory committee to the Bureaus of Mines and of Standards for prosecution of this investigation.

On Feb. 20, 1922, a second conference in New York, at the time of the annual meeting of the American Institute of Mining and Metallurgical Engineers, was attended by representatives of various technical societies. At this meeting, the report of Nov. 20, 1921, of the Chiefs of Divisions of Metallurgy of the Bureaus of Mines and of Standards, was discussed and it was agreed:

1. That the survey recommended by the two bureaus should be undertaken, which survey would have for its object the ascertaining of the present status of the practice of the heat treatment of drill steels, the extent to which the breakage occurs, etc.; and
2. That a report should be presented, of the findings of those who made the survey, to the Advisory Board to the Bureaus of Mines and of Standards.

The Bureau of Standards assigned $4000 to the prosecution of a three-months' survey, which was carried out in April, May, and June, 1922, of about sixty mines west of the Mississippi River by Prof. Charles Y. Clayton, F. B. Foley, and Henry S. Burnholz, metallurgists, under the direction of the U. S. Bureau of Mines and the U. S. Bureau of Standards, which in turn acted under advisement of the executive committee of an advisory board which is composed of representatives of various technical societies interested in "the investigation of the breakage and heat treatment of rock drill steels and other steels and alloys subjected to similar impact stresses." The Executive Committee is composed of the following:

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* Metallurgist, U. S. Bureau of Mines.

† Professor of Metallurgy, Missouri School of Mines; Metallurgist, U. S. Bureau of Standards; Consulting Metallurgist, U. S. Bureau of Mines. Messrs. Foley and Clayton were transferred from the staff of the Bureau of Mines to that of the Bureau of Standards to carry out this survey.

‡ Metallurgist, U. S. Bureau of Standards.

B. F. Tillson, assistant superintendent, The New Jersey Zinc Co., Franklin, N. J.
W. H. Leonard, president, Denver Rock Drill Mfg. Co., Denver, Colo.
J. A. Mathews, president, Crucible Steel Co., of America, New York City.
Van H. Manning, director of research, American Petroleum Institute, New York City.
G. K. Burgess, chief, Division of Metallurgy, U. S. Bureau of Standards, Washington, D. C.
D. A. Lyon, chief metallurgist, U. S. Bureau of Mines, Washington, D. C.
F. W. Denton, vice-president, Copper Range Co., Boston, Mass.

In making this survey, the field stations of the Bureau of Mines were used as bases of operation and the superintendents of these stations, likewise the mining engineers of the Bureau in the field, were called upon by the Bureau of Mines to assist in the work.

This report is based on a study of the progress reports made by Messrs. Clayton, Foley, and Burnholz during the time of the survey which was carried on during April, May and June, 1922, and is a summary of the impressions, or a critical review of the information secured during the survey.

Respectfully submitted,
D. A. Lyon,
Chief Metallurgist,
U. S. Bureau of Mines.
G. K. Burgess,
Chief of Division of Metallurgy,
U. S. Bureau of Standards.

Introduction

Generally speaking, the mining industry has not recognized that there is a drill-steel problem of vital importance. This is shown in the lack of true records of drill-steel breakage and performance and in the methods of heat treatment. True, some mines, usually large ones, give considerable daily attention to the details of performance and breakage and have expended considerable money in perfecting their means for heat treating; but mines equipped with a smith shop run as an independent unit, with but casual concern for performance, are more commonly found. The importance of heat treatment and handling of drill steel depend on the character of the ground to be worked. In a mine where the conditions are extremely favorable to drilling, the question of drill-steel performance receives but cursory attention, whereas it is a much mooted question at properties where drilling is quite hard.

How vitally important the drill problem is, we cannot say. We cannot arbitrarily say that drill-steel performance can be doubled—we do not know. That it can be improved is undeniable. We cannot expect each mining company to support its own drill-steel research department. No doubt, other problems in the mining industry involve sums far beyond any saving that may result from improved drill steels. But what may be a small saving to the individual operator may mean considerable to the industry as a whole.

Little space is necessary for convincing those familiar with drilling operations that actual performance figures obtained in one mine may be meaningless if applied to another. The drill-steel problem of a given mine may be peculiar to that mine and to no other. We, therefore, in this review, refrain from the use of figures, except for purposes of illustration, and confine ourselves to an exposition of the general impression received from a visit to sixty mines west of the Mississippi River. These companies mine copper, iron, gold, silver, lead, and zinc ores.

Heat Treatment

In studying the heat-treatment problem, as it concerns drill steel, we are impressed with the lack of means for controlling temperature. Without this, the product in any heat-treating operation is bound to suffer. Of six companies that had tried pyrometry, five had discarded the equipment; invariably the complaint was that the steels did not attain the temperature indicated by the pyrometer. It appears to be entirely a question of time. If a furnace is operated at a constant temperature and steels of a constant cross-section are introduced into that furnace, a definite period of time must elapse before such steels reach that temperature. If production is to be hastened, the way to get it is to increase the capacity of the furnace. If it takes 15 min. to heat a drill steel to 1475° F. and the furnace has a capacity of ten steels, only forty steels can be treated per hour; to double that number the size of the furnace must be increased to accommodate twenty steels; the heating cannot be hastened. To run the furnace at a higher temperature and remove the steels in a shorter period of time increases the production, but it practically eliminates the possibility of temperature control of the steel. The need, therefore, is for a furnace of sufficient capacity in which a pyrometer can be used to indicate the temperature of the steels heated therein.

In addition, such a furnace should be capable of attaining the necessary heat with a reducing atmosphere in the heating chamber. Rarely, we might almost say never, have we seen a mine smith-shop furnace heating drills for sharpening being operated with a reducing flame. One smith-shop foreman only stated that he aimed to have his men keep a soft flame in the heating furnace. Most of the hardening heats, also, are taken under oxidizing conditions.

Where thermocouple pyrometers have been abandoned, and in a few other instances, optical pyrometers were found in use for checking the workman's eye. The optical pyrometer was always of the LeChatelier type and was often used for sighting on the tip of the steel as it rested on the edge of the quenching tank. Under such circumstances, the reading may be within 100° of the true temperature. We question if this type of pyrometer, if any optical pyrometer is used, is adaptable to measuring temperatures in drill-steel work. The Holborn-Kurlbaum, or disappearing-filament type, offers more chance for selective temperature measurement and gives a better end point. It is our opinion that optical pyrometers are not suitable for drill-steel work, particularly in inexperienced hands.

The magnet is used quite extensively to control the low point in hardening temperature. Both the magnet and the optical pyrometer must be considered makeshifts and an effort made to supply a more accurate means of temperature measurement.

Oil, coal, and coke furnaces were used and even with coke more air was being used than necessary to produce complete combustion and the furnaces were usually pushed to as high a temperature as they would stand in order to hasten the heating of the steels.

In most of the mines visited, bits were quenched cold in water and not drawn before using. It was our experience that drawing was necessary in the treatment of churn drills of 6-in. gage used in drilling very hard rock. If such drills are used in the hardened state without drawing, spalling is inevitably the result. We all know that such large sections get a less severe quench than the small steels used in ordinary underground mining. It has also been the observation of one of the authors that drills about 1½-in. gage have as long a life when started with a dulled edge as when started with a keen one. Apparently when a keen-edged drill has struck the first blows its edge is gone. We wonder if many tools are not started with a sharp decarburized edge, which is flattened out immediately, and if, in other instances, the quenched edge is so brittle that it spalls off immediately. It is difficult to get evidence of the mechanism of failure at the cutting edge because, if there were spalling, the succeeding blows of the cutting edge of the drill would abrade the fractured surface and give it the appearance of wearing. The question therefore arises, in view of the experience with large churn drills, whether drawing at a low temperature will not improve a drill steel heated in a proper atmosphere and quenched from the correct temperature by enabling it to hold its keen edge for a longer period of time. This question is further pressed by the belief that few steels are quenched from the proper temperature or heated in a proper atmosphere. In comparing the experience with small drills with that of the 6-in. churn drill, it is to be remembered that the compositions are the same and that the cutting edge of the smaller steels is more acute than that of the larger ones, and therefore is quenched more severely. If the larger drill spalls when used without drawing why does not the smaller one? One answer is that the smaller drill is decarburized on its thinner cutting edge; another is that it may spall and we do not observe it; and third, that the cutting edge of the larger drill has more energy expended on it in service. With the present mine smith-shop equipment and methods, this problem could not be solved; but with improved conditions, drawing at low temperature may some day be resorted to. There are so few instances in metallographic practice when high-carbon steel is used in the undrawn condition under such severe conditions of shock that we wonder at the measure of success that is attained every day with drill steel.

In ten of the sixty mines visited, the practice in hardening was to quench from the sharpener without reheating. Because the temperature to which the steels are heated for sharpening is not controlled, the finishing temperature from which this quenching is done varies. In addition, the steel is heated for a greater distance from the end than is the practice in reheating for hardening. Only from ½ to 1 in. of the tip of the bit is quenched and the heat stored up in the portion back of this immersed part retards the quench, with the result, probably, of a residual heat drawing, at least so it appeared to us. We do not know how injurious is the variation in the hardening temperature in this practice. We understand that no growth of grain persists during hot working so that, if severe working of the metal is carried on up to the moment of quenching, the quenching temperature ought not to be of such vital importance. From observation, however, we are inclined to believe that the finishing strokes of the sharpener are quite light and only in the beginning of the operation of sharpening is much work expended on the steel; consequently the quenching temperature may be of considerable importance. There is some compensation in the method in that those steels that finish hottest are quenched less severely than the cold ones, because the greater amount of residual heat produces a greater drawing effect on them. The users of this method express satisfaction, but, as in the case of drill-steel heat-treating practice at large, having no index of true performance there is no knowing to what extent improvement may be made.

It was noted in a great many shops that where the heat for hardening had crept back too far along the steel the portion in which heat was not desired was placed under a tap to cool before quenching the tip. In such instances the tip was invariably overheated. The average drill steel heat-treater seems to know that steel should not be quenched from too high a temperature, and if he has inadvertently overheated his steel he will adopt the method just described or, if the heat has not crept back on his steel, he may hold the bit on the edge of the bath to wait until its temperature has fallen to what he considers the proper degree before quenching. He does not realize that once the steel has been overheated the damage is done and allowing to cool before quenching does not repair it. Reheating to the proper temperature after cooling the overheated piece to black is the only method which will restore the grain of the steel to its proper fineness.

Whatever may be said regarding the variation in the severity of the work which the bit end of drill steel may have to meet in drilling rock of different hardness, we are forced to acknowledge that the service of the shank end is fairly constant throughout the mining field. Yet there is far more variation in the heat-treating methods applied to the shank than to the bit of drill steels. The length of shank heated for hardening, the length immersed, the quenching medium and the time of holding in the quenching bath, not to mention the hardening temperature, all vary from shop to shop. Quenching in water and either not drawing or drawing to anything from a light straw to a dark blue are found. Oils of different kinds have different quenching power, and many kinds of oil are used. The result is Brinell hardness on shanks of from 250 to 700. Who knows how hard a shank should be? It should be hard enough not to mushroom and not hard enough to chip itself or the piston of the hammer, but how hard is that? Tentatively we should say that the hardness should be about 400 Brinell and such hardness can be procured by a plain oil quench.

When a lugged steel is made it is heated some distance back—say 6 or 7 in. from the end—for shaping. Only the metal in the immediate neighborhood of the lug is worked in the machine and consequently all the remainder of the portion heated is permitted to cool from a quite high temperature. In order to reclaim this overheated and unworked metal a second low heat, at about 1400° to 1425° F. should be taken over the entire portion heated for the shaping operation and the steel permitted to cool in air. If this entire portion is quenched in oil, a sharp and undesirable structural change is made between the oil-quenched portion and the shaft of the steel which is in the "as rolled" state. The air cooling tends to produce a more uniform condition. While this portion is air cooling an inch of the end of the shank may at the same time be cooling in oil.

BREAKAGE

Amount of Breakage

It is practically impossible to make anything like a fair estimate, from data collected, of the percentage of new drill steel that enters into the mine and is broken in service. It may be said that records of breakage are rarely kept and it is a question whether the figures derived from such records as have been kept give a true breakage figure. Not that such records are not accurately kept, but the basis of estimating is questionable. Invariably the figures appear as "percentage of steels sharpened." Such a figure does not tell how long a piece of drill steel lasts before breaking; how many pieces of steel wear down to unusable length without breaking. May it not be that every length of steel entering a mine is broken at some time before it is ultimately discarded as too short for further use? But all the steels broken in service are not rendered useless because in most cases the break occurs near one end and entails nothing more serious than the loss of the short piece, say a foot or so in length, and the time necessary to forge either a new bit or shank end as the case may be. It is the frequency of the occurrence that is probably of most interest. In one mine, where good records were kept, about 6500 steels were broken and 2600 new steels were issued during the same year, from which one might draw the conclusion that it was not necessary to replace all of the 6500 broken steels. The 2600 new steels replaced steels worn too short or broken into unusable length; therefore, more than 60 per cent. of the 6500 broken steels continued in service. In this property, each machine broke nineteen drills per month, or roughly two in every three days. The breakage was found to be 6.2 per cent. of the drills sharpened for the year; a total of about seventeen steels were broken per day. In general we find this to be average experience of the mines where breakage records are kept. Where only an estimate was had the breakage was given as 1 or 2 per cent. of the sharpenings. No doubt such estimations are far from correct, but they indicate that such mines consider breakage of negligible importance.

Location of Breaks

Cruciform steel was found to have an exceedingly high breakage, no doubt in a measure because this section is lighter than others. Stoper steels usually gave a greater breakage than those used in mounted drills; the service required of steels used in stopers is necessarily more severe than in other drilling operations. In drilling with a machine the weight of which is carried to some extent by the drill steel, there is friction between the steel and the lower side of the hole, which may be a great disadvantage where shaped steels are used. Partly for this reason, one company that proposes to standardize has selected a round steel.

Only one mine kept account of the point in the length of the steel at which the breaks occurred; there 80 per cent. of all breaks were within 6 in. of the bit end. This was the only mine reporting such experience. Mines that kept no record, but depended on general observation, usually reported the greater breakage at the shank end. The bit breakage just referred to may result from small but rather numerous checks, resembling pulls on the surface of the steel, where the change of section brought about by the upsetting incident to forming the bit occurs, or to a "metallurgical" break, *i.e.*, to the change in structure from that of the "as rolled" bar stock to that of the forged and heat-treated tip. This particular bit breakage is under investigation. Little breakage was reported in the main portion, or shaft, of steels. The curves shown in Fig. 1 have been derived from the records of the company that experienced this bit breakage; the actual figures are given in Table 1. It is at once apparent that the bit breakage is greatest in the short lengths; there are a number of peculiarities that may be responsible for this.

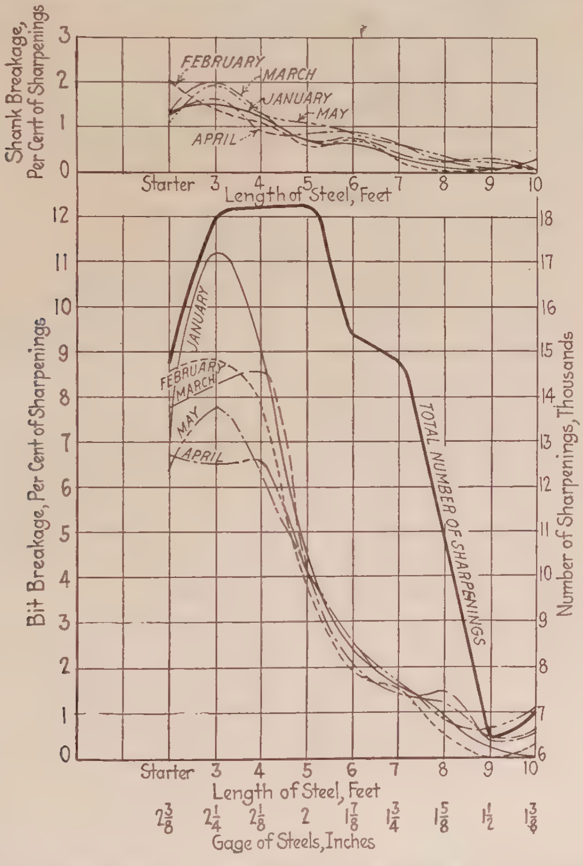

Fig. 1.—Shank breakage, by months, in per cent. of sharpenings vs. length of steel; bit breakage, by months, in per cent. of sharpenings; and total sharpenings vs. length and gage of steels.

The curve of number of sharpenings has the same general shape as that of the percentage of breaks, although it by no means follows. If the number of steels of the different lengths were available, it would be interesting to plot the sharpenings per steel of given lengths against breakage. This figure of sharpenings per given length of steel would represent performances, for a sharpening follows a performance period. From the point of view that perhaps most of the shorter steels have been

TABLE 1

Month	Starters 2⅜ in. Gage			3-ft. Steel 2¼ in. Gage			4-ft. Steel 2⅛ in. Gage		
	Total Sharpened	Broken Steel, Per Cent.		Total Sharpened	Broken Steel, Per Cent.		Total Sharpened	Broken Steel, Per Cent.	
		Bits	Shanks		Bits	Shanks		Bits	Shanks
January	2,751	7.1	1.3	2,801	11.2	1.5	3,317	9.0	1.2
February	2,385	8.55	2.01	2,919	8.84	1.4	2,916	7.85	1.08
March	2,836	7.72	1.34	3,948	8.25	2.0	3,531	8.58	1.33
April	3,335	6.71	1.22	4,290	6.5	1.61	4,190	6.61	0.9
May	3,364	6.39	1.07	4,006	7.76	1.94	4,258	6.31	1.29

Month	5-ft. Steel 2 in. Gage			6-ft. Steel 1⅞ in. Gage			7-ft. Steel 1¾ in. Gage		
	Total Sharpened	Broken Steel, Per Cent.		Total Sharpened	Broken Steel, Per Cent.		Total Sharpened	Broken Steel, Per Cent.	
		Bits	Shanks		Bits	Shanks		Bits	Shanks
January	3,089	4.5	0.7	2,608	2.4	0.6	2,397	1.6	0.3
February	3,052	3.86	0.55	2,343	1.92	0.76	2,406	1.53	0.24
March	3,709	4.26	0.64	2,982	2.11	0.77	2,984	1.44	0.3
April	4,277	4.77	0.84	3,631	2.58	0.90	3,549	1.41	0.62
May	4,101	4.21	1.09	3,844	2.6	0.83	3,433	1.7	0.60

Month	8-ft. Steel 1⅝ in. Gage			9-ft. Steel 1½ in. Gage			10-ft. Steel 1⅜ in. Gage		
	Total Sharpened	Broken Steel, Per Cent.		Total Sharpened	Broken Steel, Per Cent.		Total Sharpened	Broken Steel, Per Cent.	
		Bits	Shanks		Bits	Shanks		Bits	Shanks
January	1,954	0.9	0.1	942	0.02	0.0	1,120	0.0	0.0
February	1,910	0.52	0.05	1,088	0.0	0.09	1,122	0.35	0.08
March	2,106	1.47	0.023	1,392	0.43	0.07	1,432	0.62	0.27
April	2,552	1.21	0.27	1,553	0.38	0.32	1,755	0.51	0.05
May	2,325	0.81	0.30	1,476	0.67	0.2	1,561	1.08	0.06

Month	All Lengths		
	Total Sharpened	Broken Steel, Per Cent.	
		Bits	Shanks
January	20,979	5.07	0.78
February	20,141	4.4	0.81
March	24,920	4.64	0.9
April	29,132	3.75	0.86
May	28,388	4.13	0.98

made from longer ones which either have been worn down or have been reduced in length by breaking, it seems likely that the shorter steels are older and therefore have seen longer service.

None of the foregoing reasons explains the predominance of bit breakage over breaks elsewhere in the length of the steel. A reason that may be advanced is that the short steels have the larger gage bits. Perhaps a higher heat is taken and more steel heated to facilitate the formation of a large-gage bit because it requires more upsetting of the end of the bar. In addition, the fact that the gage is larger means more work must be transmitted through the steel in order to advance a given distance and therefore there is more chance for breakage.

Cause of Breakage

We find no records showing how much breakage is caused by using drill steel for work for which it is not intended; for example, using it for a pinch bar or hammering a drill stuck in a hole.

There are several conditions that may be responsible for shank breakage, all of which are fairly well known. In lugged steel of a given shape, breakage is usually higher than in lugless steel. Often breaks occur through the lug itself, also where the lug joins the original section of the bar. No doubt such breaks are caused, in many instances, by the irregularities produced in the hole of the hollow drill steel when the steel is upset to form the lug. One mine at least has taken special pains with the forming of the shank of its lug steel to avoid shank breakage. Three and sometimes four heats are taken on the steel for this operation. Following the forging, a heat is taken over all the shank end that has been heated in forging, but only 2 in. of the end is quenched and drawn to a blue, thus allowing the forged portion to cool in air. The breakage is estimated at but 2 per cent. of sharpenings by this company.

In any steel, lugged or lugless, we should expect a higher breakage in the neighborhood of the point where the steel leaves the chuck of the drill, because the chuck holds the steel within it fairly rigidly while the greater length of the steel is subject to vibratory and bending stresses. In the lugged steels, the condition is aggravated by the change in structure brought about in forging. The mine referred to in the preceding paragraph takes a wise precaution, if the lug so produced is of sufficient hardness, in running the heat back over the portion heated for forging and permitting an air cooling from there. We do not believe that a slow cooling, as in lime, of this portion is at all desirable. The air cooling probably gives a stronger and tougher steel.

Such breakage at the bit end of the steels as may be attributed to change of structure brought about through heating back some distance for forging might be mitigated by carrying the heat farther back in the hardening, and quenching only about ½ in. or so of the end, permitting the remainder to cool in air. Such an expedient might be resorted to where breakage back of the bit occurs rather than to allow the steels to go into service with a coarse structure at the point of change of section.

Some breaks at the bit end no doubt result from notches produced on the surface of the bar and perhaps also in the bore during the forging and upsetting in sharpening. We have seen many such defects but have not experimentally determined the cause. Overheating in an oxidizing atmosphere, which we found to be common, might start them. Under such conditions, the metal exposed to the oxidizing flame is decarburized. The point at which dilation occurs during the cooling of a low-carbon steel does not coincide with that at which dilation occurs in a higher-carbon steel. Therefore the decarburized surface may be contracting at the same moment that the higher carbon interior is expanding. The greater mass of the steel, being of high-carbon, may pull the external low-carbon metal apart, particularly if the steel has been rendered brittle by overheating. Notches may also result from rough handling and from laps, which open up in cooling.

Where raised-center bits were in use, the mines reported trouble from general breakage. Insofar as could be observed, their general drill-steel practice showed no great departure from common mine practice, except in the shape of the bit. The raised center might well be the cause. The ease of rotation of this bit is due to the pivot-like support of the steel at the raised point. The brunt of the work is thus thrown on the point, which has least rock to break, and the edges of the bit, which have the greatest amount of rock to remove, probably hang up, thus causing a torque in the bar. Furthermore when an ordinary bit without a raised center fails from dullness, the ends of the wings have been worn away more than the center, giving it something of a conical appearance and we all know that breakage results from persisting in drilling with such a bit. The raised-center bit has some of the characteristics, in shape, of a dull bit of the flat-end type.

Breaks occurring in the shaft portion of drill steels are few, but may result from a number of causes, chief among which are foreign inclusions in the steel, nicks in the surface or bore, and poor metallographic structure in the rolled material. Since such breaks are relatively uncommon, it may be assumed that steel of good quality is being furnished at present, for it is only such breakage that can, with any degree of certainty, be charged against the steel itself. It seems to us that in every case where a new steel was tried out, it gave better results than the old steel; probably because the new steel was given more attention by the salesman and by the smith. One mine was trying a steel containing a small quantity of alloy and another was trying a foreign make and so far both had found an improvement. Perhaps there are some steels which are inferior to others, but we found one company strongly endorsing a brand that is strongly condemned by another. In a record of a competitive test, the company brand, which it stated was normalized after rolling, showed a few breaks at the bit end and none at the shank end; whereas, the competitor's brand, apparently not normalized, showed little bit breakage but a number of shank breaks.

Another probable cause of breakage is poor alignment of the drill with the hole.

There is considerable talk of "fatigue." It is said that the older steels break more often than the newer steels and one receives the impression that steel tires or crystallizes in time. We find no support for such a theory. If the elastic limit of the steel has been exceeded we of course look for failure; but Prof. H. F. Moore records one hundred million reversals of stress on a steel of eutectoid composition under loads up to the elastic limit without failure. This, of course, can be true of a steel only in a state of almost perfect uniformity. When a bar of hypoeutectoid steel is pulled in the tensile machine, we get the tensile strength, or elastic limit, of an aggregate composed of ferrite and pearlite. One of these constituents, ferrite, is weaker than the other. Conceivably such steel may be subjected to repetitions of stress with loads in excess of the elastic limit of the weaker constituent and thus cause failure of the entire section in time, though the load is below that of the elastic limit of the cross-section as a whole. It may take a long time to cause failure with the loads involved in drilling and the resulting failure may well be taken to be a fatigue failure whereas the mechanism is exactly that of failures produced through overloading. An imperfection may act in much the same manner. A small discontinuity in the metal may be spread gradually under repeated blows until it has reduced the effective cross-section of the steel to the point where it has not the mechanical strength to withstand another blow, and it fails. Such a failure takes time, also, and may not occur until the steel has been long in use. Many so-called fatigue failures show evidence of starting from such a nucleus. The surfaces of the discontinuity are pounded together with each blow of the drill and become smooth faces, and as the discontinuity spreads the smooth faces become larger and so the final fracture shows this smooth area, in which the crack has been spreading for a long time, and a bright crystalline area (the last to break). A failure may show a coarse crystalline fracture and we are told that the metal has crystallized in service and are asked to observe the coarse fracture. The answer is that the metal was coarsely crystalline from the beginning and failed for that reason.

The breakage situation may be summed up as follows: We are told that the mines experience a great amount of breakage. The mines report that their breakage is not excessive; true, they have a small amount and, while we find but little evidence of a breakage problem, we observe that there are precautions to be taken which are not well known to the mine smith. The mines, generally speaking, feel that their breakage is so inconsequential that it does not warrant expenditure of time and money to determine its details; however, if the drill steel in use can be improved in quality, steps to determine how to bring about the improvement might well be taken. The fact that where records are carefully made the breakage figures are higher than in mines where none are kept indicates that in the latter case there is probably more steel breakage than is supposed.

Welding

In but four mines of the sixty visited had welding been seriously considered. In one it had been abandoned; in another, practically so; and in the other two, it was being carried on in a more or less routine way. Generally speaking, any steel which, in breaking, produces a length greater than 2 ft. can be made into a shorter steel without welding; this is probably the reason why more welding together of short lengths has not been done. Drill steel weighs roughly 3 lb. per ft. of length and if two 2-ft. pieces are reclaimed in making a 4-ft. steel a saving of 12 lb. of steel is effected. With drill steel selling at say 18 cents per pound the saving is $2.16, less the cost of welding. Figures from one mine indicate a breakage of seventeen steels into unusable lengths. Roughly, some part of $36.72 would be saved per day if successful welding were done. In plants where welding is being done the work is done by a man not engaged in routine work for the time being; in such circumstances welding probably pays. No plant furnishes figures showing the number of successful welds made; it is impossible therefore to say how successful the operations are.

Welding requires the greatest skill on the part of the smith for the production of a piece in which the steel has been completely united. The heating required for successful welding leaves the unworked portion of the steel in poor condition to withstand shock and annealing is necessary for the restoration of the overheated metal. Where annealing is not done, the smith claims that the steel broke outside the weld, that his welded steel is better than the original metal; he ignores the fact that while he has by working restored the good condition of the metal immediately around the weld he has left the unworked and overheated steel outside the weld in an inferior condition. At one mine, the steels were annealed completely after welding but periods were experienced when batches of welded steel failed by breaking. The annealing was not under pyrometer control, the pieces being heated to a "dull red" and cooled down in the furnace.

DETACHABLE BITS

Considerable interest is manifest in the use of detachable bits. Two types are now being experimented with in the field. One is attached to the drill-steel shaft by a dovetail joint tapering in two directions so that it tends to tighten in service rather than loosen. The other type is attached by means of a thread and collar. Jamming of the thread in this latter type was an undesirable feature, which we understand has been overcome. There is no questioning the advantages that may accrue from the use of detachable bits. It seems rather ideal for the miner to equip himself, before going underground, with sufficient bits for the day's work, all of which he can carry in his pocket. In addition, such small pieces can be more easily and more correctly heat treated than the tip of a long bar of steel.

We cannot say that the detachable bit is a success. In the dovetail type, poor registering of the bit with the shaft throws the axis of the bit out of line with the axis of the shaft. This causes the bit to rotate eccentrically an therefore to throw the wear all on one side and produce a hole of larger gage than the bit diameter. This would cause binding in the hole when such an eccentric bit followed one which was true. In either type of bit, the joint produces a reduced section and notches where breakage may well occur, particularly if the bit is not supported at all points by being well fitted to the shaft. All the joint ends of the drill-steel shafts will necessarily have to be heat treated as well as the bit. The matter of costs will, of course, determine the feasibility of this innovation. Insofar as we are able to judge the detachable bit has not shown a favorable balance in costs, but since it is to be considered at present in the experimental stage of development judgment must be withheld.

It is claimed that the detachable bit makes a saving in the amount of steel discarded. In actual tonnage this may or may not be so. It appears to us that if the claim that the detachable bit will stand three sharpenings by grinding before it has lost a prohibitive amount of gage, there is a discard of about 5 oz., about the weight of a bit, for every three runs. The average mine drill steel may be taken as a 6-ft. length, which weighs about 3 lb. per foot, or 18 lb., which is equivalent in weight to 58 detachable bits. Each detachable bit stands three runs, so that the 18 lb. of ordinary drill steel should stand 174 sharpenings before it is discarded as under 2 ft. in length. The 174 sharpenings should use up, then, 4 ft. of drill, or each run and resharpening should consume 0.275 in. In other words, to equal the discard of detachable bits a little less than four runs should be had from an inch of ordinary drill steel. It is on such a basis as this that comparison of waste should be made.

As to handling, it is inconceivable that the circulation of long lengths of drill steel between underground and the smith shop will be entirely done away with because some of the joint ends of the drill-steel shafts will have to be redressed from time to time and the shank end will not last indefinitely.

The success of the detachable bit may result from the use of an alloy that will combine hardness with great resistance to abrasion. Such bits used on a shaft, perhaps also of alloy steel carefully heat treated throughout and possessing strength and toughness, would give nearly ideal service. The alloy that may prove best for bits may not be the best for shanks, so that an entire alloy steel drill of the type now in use might not be successful. Were the shaft and bit separate pieces, the shaft having been subjected to proper heat treatment at the start, would not need to be reheated as often as an ordinary drill steel, which is reheated for every sharpening of the bit. In general, the detachable bit principle carried out with the proper kind of alloys for bit and shaft would reduce the amount of smithing and the efficiency of the steel would be greatly increased.

Even if the mechanical features of the detachable bit are perfected, it is our opinion that it will take close figuring to show a saving in costs for carbon-steel detachable bits as against ordinary carbon-steel drills. Convenience in handling may, however, recommend it in spite of a possible little or no saving.

DRILL-STEEL HANDLING

Some drill steel, perhaps not much, is lost underground. To check up such losses and to keep good performance records a systematic method of drill-steel handling is followed in some mines. In one of the most systematic methods observed, a certain number of steels held in a ring are sent underground with each machine, stamped with the machine number. Each day the drilled steels from each machine are sent to the smith shop, held together by this ring. The steels are sharpened, hardened and returned to the ring in which they are sent underground and unloaded at the level where the machine is working to be carried to it by a nipper. In this way each miner is made responsible for his tools.

The general idea in most mines is to make sure that the miner has sharp drills at all times and so to obviate any charges of holding up the drilling on that account. No attempt is made to keep track of the individual steels and it is not generally considered necessary.

Central distribution stations underground are often used and new steels sent to the faces by nippers, who collect dulled steels for the return trip, keeping advised of the needs of the driller. The dull steels collected at the different distributing stations are sent up and new steels brought down from the smith shop.

The opinion was expressed by more than one mine superintendent that those immediately in charge of drill-steel sharpening and hardening are not well informed concerning the effects of heat on steel. The theory of hardening steel is unknown to them. The information on the subject is really not available because it is usually put in terms with which they are totally unfamiliar. One of the needs appears to be a treatise on the heat treatment and working of steel written in non-technical language.

STANDARDIZATION

There were thirteen sections of drill steels used in forty-eight mines tabulated. Steels were in round, hexagonal, quarter-octagon, and cruciform shapes and were ⅞, 1, 1⅛, 1½ in. in diameter. Such a condition shows the importance of standardization; we noted five sections in use in one mine. Standardization has been adopted by some mines, however, and other mines are watching their results. The following is a record of the number of mines in which the different shapes were found in use.

Round			Hexagonal		Quarter-octagon			Cruciform				
1 in.	1⅛ in.	1¼ in.	⅞ in.	1 in.	⅞ in.	1 in.	1⅛ in.	⅞ in.	1 in.	1⅛ in.	1¼ in.	1½ in.
2	9	32	41	8	5	19	1	1	8	12	15	1

Oddly enough the two companies using 1-in. round are standardizing on that section; six mines are standardizing on quarter-octagon, two on ⅞-in. hexagon, and one on 1¼-in. round.

Where fitchery ground is encountered, the round steels with lugged shanks hold sway because of the necessity of a positive method of withdrawal from the hole. In other places the shaped steels, quarter-octagon and hexagon, are preferred. The cruciform section is in little favor and there are strong indications that it may be dropped. It is a light section and gives more bar length per pound, which reduces drill-steel costs slightly but in the long run this is offset by the greater breakage. There seems to be no argument against ultimately reducing shapes to perhaps three at the most: for jackhamers, ⅞ in. hollow hexagon with collars; for heavier machines 1¼ or 1 in. round, and 1-in., quarter-octagon.

There is some evidence that considerable hammer breakage has been experienced in using the 1-in. round section in certain types of machines, but a change in machines has apparently done away with the trouble. The machine adopted, however, is of an obsolete type. In another section of the country, 1-in. round has been adopted as standard without, as far as we know, any trouble. It is the general opinion that a lighter section is necessary for use with jackhamers; yet there are mines standardizing throughout on one section, altering their machines where necessary in order to make this possible.

One argument advanced for round steel is that it is not retarded, as shaped steels are, by bearing on the periphery of the hole during drilling, particularly in stoping. Against the round section is the question of forging lugs, which requires additional smithing. By using an anvil-block chuck, the shaped steels may be used without lugs, which is not possible with the round steels. Yet if a lugless steel were adopted, there are indications that there would be a reduction in the breakage so common in the region where the lug is formed.

Standardization of drill-steel shapes would be desirable both from the manufacturers' point of view and from the users'. Changing rolls in the rolling mill is no small item in the cost of drill-steel manufacture, and a reduction in the number of shapes carried in stock is an important consideration. The advantages to be gained by the mines and by the drilling-machine makers, also, are considerable.

In connection with the subject of standardization, attention is called to the following reports:

Digest of Paper on Suggestions for the Standardization of Hammer Drills and Accessories, by George H. Gilman, pp. 104–111, Standardization Bulletin of the American Mining Congress, 1921.

Preliminary Investigation on the Standardization of Drilling Machines and Drill Steel, prepared under the direction of Arthur Notman, superintendent mine department, Copper Queen Branch, Phelps Dodge Corpn., pp. 112–154, Standardization Bulletin of the American Mining Congress, 1921.

Progress Report of Committee on Drilling Machines and Drill Steel, pp. 186–191, Standardization Bulletin of the American Mining Congress, 1922.

Standardization of Drilling Machines and Drill Steel, by Geo. H. Gilman, pp. 198–206, Standardization Bulletin of the American Mining Congress, 1922.

Summary

The general survey of sixty mines west of the Mississippi River suggests investigations having for their purpose the following:

1. The designing of a furnace of sufficient capacity, which can be operated with a reducing atmosphere and in which drill steels can be heated under pyrometric control. Ideally the pyrometer should measure the temperature of the steels themselves. Such a furnace should be designed so that first costs and upkeep are not prohibitive to the mine handling say 300 steels per day.

2. The determination of the proper carbon content of plain carbon drill steel.

3. The determination of the proper structural condition from end to end of drill steels: (*a*) What is the proper hardness for drill shanks? (*b*) How may the sharp structural change at the bit and shank ends of

drill steel be eliminated? (*c*) What is the best condition of the steel in the shaft portion of drill steels for the reduction of breakage? (*d*) Under ideal heat treatment conditions, is drawing of the bit end desirable?

4. Determination of the best size and shape of cross-section of drill steel.

5. Investigation of alloy steels for use in rock drilling.

Finally, we suggest that some authoritative agency, perhaps the Bureau of Standards or the Bureau of Mines, publish a brochure setting forth, in a non-technical manner, the principles involved in the production of good forgings and good heat-treated products, for the benefit of the practical man daily engaged in steel treatment and also for the use of men holding positions of responsibility in whose training the metallography of steel has had no place.

(*Discussion of this paper begins on page* 677.)

Hardness and Heat Treatment of Mining Drill Steel Shanks*

By Charles Y. Clayton,† Rolla, Mo.

(New York Meeting, February, 1923)

The shank, to give good service, should not upset nor should it cause excessive wear on the various parts of the machine. To fulfill these requirements, the steel must have a certain hardness—that is, resistance to deformation. The hardness is expressed in conventional terms, as Brinell or scleroscope number. Before deciding on the hardness that is proper, several factors should be considered; the most important are: type of drill steel used (whether hollow or solid), length of steel, nature of ground, air pressure, and hardness of hammer or anvil block (usually 600 Brinell). These are not all, but they are the most cogent ones. Without doubt, it would be better to use a shank that is too soft than one that is too hard. Most satisfactory service will be obtained from shanks that are just hard enough to resist any tendency to upset.

To express the proper hardness in terms of Brinell numbers would mean a great many tests over a long period of time, or a study of drill steels that are being used successfully in the various districts. To obtain some information along this line, the persons making this investigation requested a number of companies to send to them drill shanks that had been given the usual treatment at that mine. The results of the hardness tests are given in Table 1. The hardness values given in Table 2 are those of shanks known to have been quenched in water and drawn. These results show that where water quenching and drawing are in vogue the hardness is rarely under 400 Brinell.

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* Work done by Metallurgical Department of Missouri School of Mines in coöperation with the United States Bureau of Standards and the United States Bureau of Mines.

† Professor of Metallurgy and Ore Dressing, Missouri School of Mines and Metallurgy; Metallurgist, U. S. Bureau of Standards; and Consulting Metallurgist, U. S. Bureau of Mines.

TABLE 1.—*Results of Brinell Hardness Tests on Shank Ends of Drill Steel*

Quarter-octagon		Cruciform		⅞-in. Hexagonal		1¼-in. Round		1¼-in. Round		⅞-in. Round		1⅛-in. Round		1-in. Round		1¼-in. Round		
Drill No.	Hardness Value	Drill No.	Hardness Value	Drill No.	Hardness Value	Drill No.	Hardness Value	Drill No.	Hardness Value	Drill No.	Hardness Value	Drill No.	Hardness Value	Drill No.	Hardness Value	Drill No.	Hardness Value	
																	On End	On Lug
OO-1	656	8-28	652	9D-1	782	SU-3	656	EBC-1	554	11-24	652	SKC-1	656	PD-1	512	SU-3	656	242
B-1	600	8-32	652	8-31	713	2-Gr-1	600	HM-1	554	11-17	600	MCC-1	600	PD-2	512	2-Gr-1	600	
BO-H	511	L-5	652	5-1	713	2 Gr-2	600	BH-1	476	11-20	600	OM-1	511	ICC-1	445	2-Gr-2	600	
SU-2	511	L-6	652	MCC-3	600	WEC-2	554	MM-4	476	11-18	555	YK-1	510	ICC-2	445	WEC-2	554	387
MM-2	511	OM-1	600	9D-3	600	2-Gr-3	512	MM-3	415	11-25	555	SKC-2	366	8	444	2-Gr-3	512	
SU-4	476	L-2	600	5-2	600	2-PD-3	512	HB-1	387	11-21	444					2-PD-3	512	387
SU-1	415	L-4	600	6-1	600	WEC-1	511	MV-1	366	11-23	418					WEC-1	511	340
MCC-2	387	8-27	600	8-29	555	IMM-1	511	BM-1	366	11-19	387					IMM-1	511	286
T-1	366	L-7	555	9D-2	555	IMM-2	511	HO-1	340							IMM-2	511	713
L-1	366	L-3	512	19-1	430	17-1	477	BO-3	320							17-1	477	430
NSM-1	352	MM-1	415	4-1	418	EM-6	477	NB-1	320							EM-6	477	
NSM-2	352	HB-2	342	12-2	402	PCG-1	476	NS-1	304							PCG-1	476	255
EM-1	342	OM-2	302	12-1	364	14-1	444									14-1	444	351
T-2	320	BM-2	228	4-2	364	14-2	444									14-2	444	269
BO-T	285			4-3	364	14-3	430									14-3	430	375
				4-4	364	18-4	430									18-4	430	293
				9D-4	364	14-4	418									14-4	418	430
				9D-5	255	15-2	418									15-2	418	387
						OD-1	415									OD-1	415	
						15-1	402									15-1	402	351
						13-1	402									13-1	402	402
						18-3	402									18-3	402	255
						18-1	387									18-1	387	269
						10-B-1	387									10-B-1	387	
						10-B-2	387									10-B-2	387	
						16-1	387									16-1	387	402
						EM-3	366									EM-3	366	387
						EM-4	366									EM-4	366	364
						EM-1	364									EM-1	364	321
						1-PD-1	364									1-PD-1	364	340
						13-2	364									13-2	364	286
						PCG-2	342									PCG-2	342	179
						18-2	340									18-2	340	241
						2-PD-2	340									1-PD-2	340	387
						EM-2	302									EM-2	302	321
						2-PD-4	302									1-PD-4	302	387
						3-2	269									3-2	269	269
						3-1	196									3-1	196	286

TABLE 2.—*Brinell Hardness Numbers of Shanks Quenched in Water and Drawn*

Form of Drills				Company	Remarks
Leyner	Cruciform	Quarter-octagon	Hexagon		
Brinell Hardness Numbers					
555				17	
402				17	
	512			18	
	512			18	
600				19	
555				19	
512				22	
512				22	
				26	Shank so hard and brittle that it shattered during test.
	364			30	
444					
477	600	386		34	
600				35	
	364			40	These were broken ends of stoper steel taken out of the chucks at one of the drill repair shops.
	555			40	
	418			40	
	477			40	
	418			40	
	477			40	
	512			40	
	364			40	
	600			41	
596		510		43	
		550			
510				45	
512				8	
512				8	
712		416	600	12	

A study of the tabulated data shows: (1) There is a wide variation in hardness between the drills treated at the different mines; oil-quenched shanks vary between 300 and 400 Brinell, water-quenched shanks vary between 386 and 600. (2) There is a great variation in the hardness of the drills treated at the same mine but by different blacksmiths. (3) A large number of the drills show excessive hardness; that is, over 500 Brinell. (4) The hardness of quite a few drills indicates no heat treatment. (5) In most cases the shanks are hard at the end and for as much as an inch from the end. (6) Most of the drills show that an attempt has been made to harden them back to and including the lug.

PRESENT PRACTICE

The hardening methods in vogue in a large number of mines are given in Table 3.[1] A careful study of these methods show conclusively that insufficient attention is paid to this phase of the drill steel treatment. A brief summary of the methods follows:

Steel with Lugs.—Of 50 shops treating lug steel, 37 were attempting to harden the lugs, the other 13 were hardening for a distance of ½ to 2 in.

Of 50 shops treating lug steel, 37 were quenching the shank in oil while the others were quenching in water and drawing to a certain color.

Stoper Steel.—Of 46 shops treating stoper steel, 32 were hardening in oil by a direct quench, while the others were quenching in water and drawing to some color.

Jackhamer Steel.—Of 43 shops treating steel for jackhamers, three were hardening so as to include the collars while the others were hardening for about ½ to 1 in.; 34 shops were using an oil quench, 8 were quenching in water and drawing, and 1 was hardening by holding shank in a jet of compressed air.

Tanks for Oil for Shank Treatment.—None of the shops visited had adequate means for maintaining an even temperature in the oil used for hardening shanks. The general practice was to harden a few shanks, then wait until the oil cooled before attempting to treat others. The oil should not be allowed to get hot.

Oil Used for Hardening Shanks.—Of 30 companies where the kind of oil was noted, 10 were using fish oil, 3 were using standard A, 4 were using transformer oil, 1 was using waste oil, 7 were using a soluble quenching oil, and 5 were using Atlanta red.

Several blacksmiths reported that shanks hardened in oil were not hard enough to stand up. There is no doubt that soft shanks were due to the nature of the oil used. Little if any attention is paid to the kind of oil supplied to the shop, as the average smith does not know that different oils have different properties when used as a quenching medium. A standard grade of quenching oil or fish oil should be used.

Use of Cyanide for Imparting Additional Hardness to Shanks.—The term cyanide as used here may mean anything from potassium cyanide to potassium ferricyanide; some smiths were even using potassium carbonate. Twelve shops out of 55 reported the use of cyanide. Careful watching of some of the hardening operations showed that the cyanide was in contact with the steel for about 15 sec. Cyanide should not be used as it imparts no additional hardness to the steel.

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[1] Taken from the report of Clayton and Burnholz to the Bureau of Standards.

TABLE 3.—*Methods of Hardening Shanks*

Co.	Round Steel with Lug			Steel Other Than Round, without Collar			Steel Other Than Round, with Collar		
	Length of Shank Treated	Temperature	Operation	Length of Shank Treated	Temperature	Operation	Length of Shank Treated	Temperature	Operation
1				End	Cherry red	Quenched in water and drawn at 575° F. in oil			
2				3 in.	Cherry red	1 in. quenched in water drawn to a blue and quenched in oil			
3				1 in.	Cherry red	Quenched in oil			
4				End	1425° F. est.	Quenched in oil			
7				2 in.	Cherry red	½ in. quenched in water drawn to a straw and quenched cold			
8	To include lug	Cherry red	End quenched in water drawn to a blue then quenched in oil						
9	To include lug	Cherry red	1 in. quenched in water drawn to a straw and quenched to about 150° F.						
10	8 in.	Cherry red	3 in. quenched in oil, drawn to a light straw and quenched cold in oil						
11			Quenched in oil			Quenched in oil			Quenched in oil
12	8 in.	Cherry red	¾ in. quenched in water, drawn to a straw, and quenched in oil		Cherry red	Quenched in oil		Cherry red	Quenched in oil
13									
14									
15	¾ in.	Cherry red	Quenched in water and drawn to a dark straw	¾ in.	Cherry red	Quenched in water and drawn to a dark straw	¾ in.	Cherry red	Quenched in water and drawn to a dark straw
	¾ in.	Cherry red	Quenched in standard-A tempering oil	¾ in. 6 in.	Cherry red Cherry red	Quenched in standard-A tempering oil Quenched in waste oil	¾ in.	Cherry red	Quenched in standard-A tempering oil
	To include lug	Cherry red	Quenched in standard-A tempering oil.	1½in.	Cherry red	Quenched in standard-A tempering oil	1 in.	Cherry red	Quenched in standard-A tempering oil

19	To include lug	Bright cherry red	2 in. quenched in water, drawn to a blue and quenched cold						
20				1½ in.	Bright cherry red	Quenched in standard-A tempering oil	1½ in.	Bright cherry red	Quenched in standard-A tempering oil
21	¾ in.	Bright cherry red	Quenched in fish oil	¾ in.	Bright cherry red	Quenched in fish oil	¾ in.	Bright cherry red	Quenched in fish oil
22	End and lug		End quenched in water and drawn to a copper; lug reheated to a bright yellow sprinkled with cyanide and quenched in oil	3 in.	Cherry red	1 in. quenched in water and drawn to a copper	3 in.	Cherry red	1 in. quenched in water and drawn to a copper
23	To include lug	Cherry red	Sprinkled with cyanide and quenched in fish oil	½ in.	Cherry red	Dipped in cyanide and quenched in fish oil	½ in.	Cherry red	Dipped in cyanide and quenched in fish oil
24	7 in.	Cherry red	Quenched in No. 2 quenching oil	7 in.	Cherry red	Quenched in No. 2 quenching oil	7 in.	Cherry red	Quenched in No. 2 quenching oil
25				1½ to 2 in.	Cherry red	Quenched in fish oil	1½ to 2	Cherry red	Quenched in fish oil
26	1 in.	Cherry red	Quenched in water and drawn to a blue	1 in.	Cherry red	Quenched in water and drawn to a blue	1 in.	Cherry red	Quenched in water and drawn to a blue
27	To include lug	Cherry red	Quenched in transformer oil	1 in.	Cherry red	Quenched in transformer oil	1 in.	Cherry red	Quenched in transformer oil
28	6 to 7 in.	Bright cherry red	Quenched in transformer oil.	½ in.	Bright cherry red	Quenched in transformer oil	½ in.	Bright cherry red	Quenched in transformer oil
29	1½ to 2 in.	Bright cherry red	Quenched in transformer oil	1½ to 2 in.	Bright cherry red	Quenched in transformer oil			
30	1 in.	Bright cherry red	Quenched in fish oil	1 in.	Bright cherry red	Quenched in water and drawn to a blue	1 in.	Bright cherry red	Quenched in fish oil
31	6 in.	Bright cherry red	Quenched in fish oil	1½ in.	Bright cherry red	Quenched in fish oil	1½ in.	Bright cherry red	Quenched in fish oil
32							1 in.	Bright cherry red	Quenched in oil
33	1½ in.	Bright cherry red	Quenched in oil	½ in. of cruciform 1½ in. of octagon	Cherry red Cherry	Quenched in water and drawn to a blue Quenched in oil	1 in.	Cherry red	Quenched in oil
34	6 in.	Cherry red	Plunged in water for 3 in. drawn to a blue and cooled on floor	½ to 1 in. of cruciform ½ to 1 in. of quarter octagon	Cherry red Cherry red	Quenched in water Set in pan containing 1 in. of still water	2 in.	Cherry red	1 in. quenched in water and drawn to a blue
35	For raising to include lug; for stoping 1½ in.	Cherry red Cherry red	Dipped in cyanide and quenched in oil Quenched in water and drawn to a blue						
36	6 in.	Cherry red	Quenched in fish oil	1 to 1½ in.	Cherry red	Quenched in fish oil	1 to 1½ in.	Cherry red	Quenched in fish oil
37	6 in.	Bright yellow	Sprinkled with cyanide quenched in oil	1½ in.	Bright yellow	Sprinkled with cyanide, quenched oil	1½ in.	Bright yellow	Sprinkled with cyanide, quenched in oil

TABLE 3.—*Continued*

Co.	Round Steel with Lug			Steel Other Than Round, without Collar			Steel Other Than Round, with Collar		
	Length of Shank Treated	Temperature	Operation	Length of Shank Treated	Temperature	Operation	Length of Shank Treated	Temperature	Operation
38	To include lug	Bright yellow	Sprinkled with cyanide, quenched in oil	1½ in.	Bright yellow	Sprinkled with cyanide, quenched in oil	1½ in.	Bright yellow	Sprinkled with cyanide, quenched in oil
39	5 in.	Bright yellow	Sprinkled with cyanide, quenched in fish oil	1 in.	Bright yellow	Sprinkled with cyanide, quenched in fish oil	1 in.	Bright yellow	Sprinkled with cyanide, quenched, in fish oil
40-A				1½ to 2 in.	Lemon yellow	Dipped in cyanide, quenched in oil			
40-B	To include lug	Lemon yellow	Sprinkled with cyanide quenched in oil	1½ to 2 in.	Lemon yellow	Dipped in cyanide, quenched in oil	1½ to 2 in.	Lemon yellow	Quenched in oil
40-C	To include lug	Lemon yellow	Held in rising current of water for 7 sec. and quenched in oil	1½ to 2 in.	Lemon yellow	Held in rising current of water for 7 sec. and quenched in oil	1½ to 2 in.	Lemon yellow	Held in rising current of water for 7 sec. and quenched in oil
40-D	To include lug	Lemon yellow	Sprinkled with cyanide, quenched in oil	1½ to 2 in.	Lemon yellow	Sprinkled with cyanide, quenched in oil	1½ to 2 in.	Lemon yellow	Sprinkled with cyanide, quenched in oil
40-E	1½ in.	Lemon yellow	Sprinkled with cyanide quenched in oil	1½ in.	Lemon yellow	Sprinkled with cyanide, quenched in oil	1½ in.	Lemon yellow	Sprinkled with cyanide, quenched in oil
40-F	To include lug	Cherry	2 in. quenched in water, drawn to a blue and quenched in oil	1½ in.	Lemon yellow	Dipped in cyanide, quenched in oil	1½ in.	Lemon yellow	Dipped in cyanide, quenched in oil
40-G				1½ in.	Lemon yellow	Dipped in cyanide, quenched in oil			
40-H	To include lug	Lemon yellow	Sprinkled with cyanide, quenched in oil	2 in. of quarter octagon	Lemon yellow	Quenched in oil	1½ in.	Lemon yellow	Dipped in cyanide, quenched in oil
				2 in. of cruciform	Cherry	1 in. quenched in water and drawn to a blue			
41	To include lug	Orange	Sprinkled with cyanide, quenched in oil	1 in.	Cherry red	½ in. quenched in water and drawn to a straw	To include collar	Orange	Sprinkled with cyanide, quenched in oil
42				½ to 1 in.	Bright cherry red	Quenched in fish oil	½ to 1	Bright cherry red	Quenched in fish oil
43	1 in.	Cherry red	Quenched in water and drawn to a blue	1 in.	Cherry red	Quenched in water and drawn to a blue	1 in.	Cherry red	Quenched in water and drawn to a blue

44	To include lug	Bright cherry red	Quenched in fish oil	1½i n.	Bright cherry red	Quenched in fish oil	To include collar	Bright cherry red	Quenched in fish oil
45	½ to 1 in.	Bright cherry red	Quenched in water and drawn to a blue						
46	1½ in.	Bright cherry red	Quenched in Atlanta red	1½ in.	Bright cherry red	Quenched in Atlanta red	1½ in.	Bright cherry red	Quenched in Atlanta red
47	To include lug	Bright cherry red	Quenched in fish oil	1 in.	Bright cherry red	Quenched in fish oil	1 in.	Bright cherry red	Quenched in fish oil
48	2 in.	About 1650° F. est.	Quenched in linseed oil	2 in.	About 1650° F. est.	Quenched in linseed oil	2 in.	About 1650° F.	Quenched in linseed

Recommended Hardness

An attempt to ascertain what is considered the proper Brinell number, from those who have had experience along this line, gave the following figures:

Rock-drill company A, Brinell, 400, scleroscope 70 to 75.

Rock-drill company B, Brinell, 350 for hollow steel and 300 for solid steel.

Rock-drill company C, Brinell, 310.

Steel company A, Brinell, 400 or less.

Steel company B, Brinell, 265 to 300.

Steel company C, Brinell, 400 to 500.

While these Brinell numbers cover quite a wide range, the shanks that show Brinell numbers within the limits mentioned should give good results. A hardness of 500 Brinell is greater than is necessary, while 300 is much too low; the writer recommends 375 to 400.

Recommended Heat Treatments

Rock-drill Company A.—Heat the shank for 3 or 4 in. below the collar to a cherry red (1475° to 1500° F.). Do not allow the steel to remain at this heat for any great length of time, but as soon as the color is uniform throughout quench the heated portion in oil (preferably cottonseed or fish oil).

When heating the steel for this operation, best results will be obtained by heating the main body of the shank to a low red (1400° F.), graduate the temperature so that the end of the shank will be about 1500° F.; the end of the shank will then be of the proper hardness and the body of the shank will have sufficient strength to resist wear and breakage.

Rock-drill Company B.—For average mining work, in moderately hard rock, where the penetration of the steels into the rock cannot be great, the practice is to heat the shank end of the steel to a dull cherry red, about 3 in. back from the end, then dip in water and immediately withdraw, letting the temper draw for 2 or 3 sec. in the atmosphere, and then dipping in oil and allowing the shank to stay in the oil until cool. Under some conditions, even this mild practice makes the shank too hard, in which case oil is substituted for the water bath, withdrawing immediately, but dipping two or three times before finally quenching in oil and allowing the shanks to remain there until cool.

Rock-drill Company C.—The hammer-drill shanks should be tempered in oil. Heat from 7 to 9 in. of the shank to be tempered to about 1600° F. (cherry red) and cool completely in the quenching oil. A soluble quenching oil is recommended.

Steel Company A.—Heat the shank end to about 1450° F., and quench in a tank containing about 2 in. of water and 6 in. of oil. Quenching the shank end in the 2 in. of water will harden the extreme end and the oil will cool off the shank. Do not draw the end after it is quenched.

Steel Company B.—After the shank is completed, place the steel in the furnace and reheat to the forging heat; then place in an iron box and cover with powdered lime and allow to cool off slowly. After this, reheat to 1450° F. and immerse the shank in oil. Great care must be taken to see that the steel stands perpendicularly, permitting the oil to come ¼ in. above the lug. The temper of this shank should then be drawn by reheating. The extent of this reheating will depend a great deal on the type of machine used and the hardness of the rock to be drilled. If the shank shows a tendency to be brittle, it might be desirable to draw the temper to about 600° F.

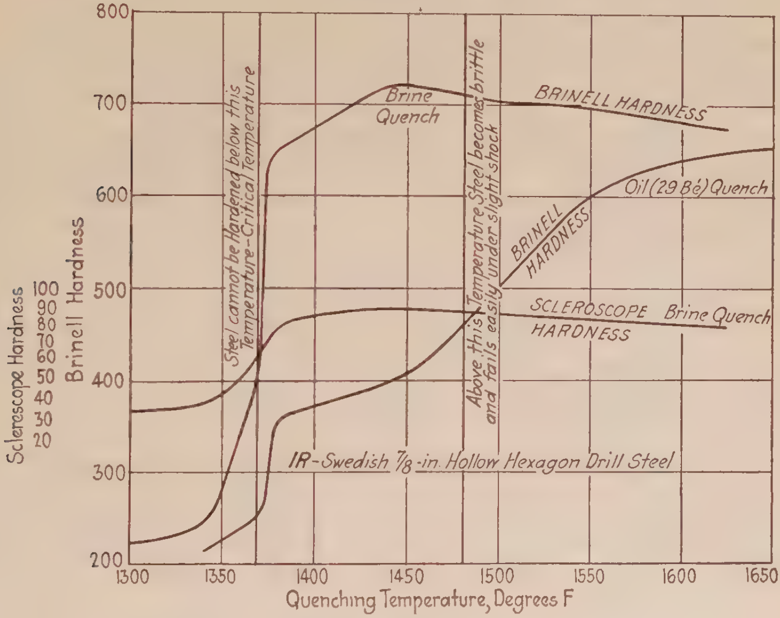

FIG. 1.

Steel Company C.—To temper the shank end, heat to about 1400° F. and chill in oil. This should leave the shank anywhere from 400 to 500 Brinell, or of such hardness that it will not upset or spall in service. The usual practice is not to draw the temper but leave it at about the hardness given by oil quenching.

Steel Company D.—Heat in muffle furnace to 1470° F.; quench in No. 2 soluble oil.

H. S. Brainerd.[2]—Drill shanks should be hardened at 1470° F. in quenching oil, such as has been described. This temperature is recommended for the ⅞-in. hollow hexagon steel, but for shanks with a larger section a little higher temperature should be used to take care of the increased section of the steel.

A study of these recommendations shows that the quenching temperature varies from 1400° to as high as 1600° F. It is possible that steel quenched from these temperatures will give equally satisfactory results, if hardness alone is considered.

Brainerd gives some interesting curves (Fig. 1) showing the effect of quenching temperature on the hardness of shanks. These curves indicate that the proper temperature lies between 1400° and 1450° F. Experiments carried on by the writer indicate that the hardness temperature curve is not as steep as the one given by Brainerd, that is, there is not much difference between a shank quenched from 1400° and one quenched from 1500° F.

Mining company A is now hardening at about 1550° to 1575°, pyroscope reading, with the following results for six shanks: 364, 340, 387, 364, 364, 351. The blacksmiths of this same company, using their eyes and quenching from a bright cherry red, obtained the following results: 364, 340, 340, 444, 387, 387. The drill steel shanks of this company are giving very satisfactory service.

There is no doubt that steel should be quenched in oil; quenching in water necessitates a draw to mitigate the hardness. Drawing by color —that is, to a straw or a blue oxide—will not give satisfactory results. A proper draw can be obtained only by the use of oil baths maintained at proper temperature.

The actual length of the shank that should be hardened has not as yet been satisfactorily determined. There is a wide difference of opinion on this question. Some shops harden the shank for about 1 in. while others harden to include the lugs. It is said that certain types of drilling machines cause excessive wear on the lugs and that other types do not. Even this point has not yet been satisfactorily settled.

Undoubtedly, there is much to be gained by reclaiming the part of the shank end of the steel that has been heated for the lugging or collaring operation. The most effective way of accomplishing this is to interpose between the forging and hardening of the tip of the shank an annealing heat that would be carried over the entire portion that had been heated for the forging, and allowing this portion to cool to black in air. The heating for this operation should not be done before the steel has cooled to black from the forging heat. Nothing is accomplished by having the cooling take place under retarded conditions, such as in the furnace or in lime. Temperature to be used should be 1450° F.

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[2] Drill Steel—Its Forging and Heat Treatment. *Compressed Air*, **27**, 304–311.

There are several questions that can be answered only by careful study and observation in practice; they are:

1. What is the proper hardness of shanks: (*a*) lug steel? (*b*) jackhamer steel? (*c*) lugless steel used with anvil block chuck?
2. Should stoper shanks have a greater hardness than other shanks?
3. Is it necessary to harden the lugs?

Discussion of these questions is invited.

DISCUSSION

(*Includes also discussion of paper of Foley, Clayton, and Burnholz, p.* 648).

N. B. Hoffman, Pittsburgh, Pa.—I understand the authors to say that all drill steel should be normalized; thoroughly normalized, the steel would have a Brinell hardness of possibly 155 to 165; would that be stiff enough for the shaft?

F. B. Foley, Rolla, Mo.—What do you mean by normalizing?

N. B. Hoffman.—I mean annealing.

F. B. Foley.—It would have to be heat treated to be made stiff and resistant to shock.

N. B. Hoffman.—That is, it would be annealed and heat treated afterwards to get the desired results?

F. B. Foley.—The entire heat treatment might consist of a simple air cooling.

N. B. Hoffman.—Heated to above the critical temperature and air cooled?

F. B. Foley.—The rate of cooling should be that which would produce the highest resistance to impact stresses.

N. B. Hoffman.—Do you know what that rate is?

F. B. Foley.—No; I think it would be air cooling.

N. B. Hoffman.—What hardness would result from such treatment.

F. B. Foley.—It would be greater than 200 Brinell.

N. B. Hoffman.—If a 1¼-in., quarter-octagon, solid bar is tested for hardness as it comes from the rolls, without any heat treatment, a difference of 50 Brinell numbers may be found between the center and the edge. This means that there is a variation in grain structure in the bar. This condition cannot be overcome by heating the steel to or through the critical temperature without sacrificing too much hardness or stiffness in the shaft. The bars should be given a "process" or semi-anneal, which would produce a fairly uniform grain structure and would not sacrifice too much strength.

F. B. Foley.—Of what does such a treatment consist?

N. B. Hoffman.—The bar is heated to just below the critical point and then let cool.

F. B. Foley.—Such treatment gives a harder bar than heating to above the critical temperature?

N. B. Hoffman.—No; if the bars were heated through the critical range and then slowly cooled they would have a Brinell hardness of approximately 150 to 165. But if these bars were tested for hardness as they come from the rolling mill, without heat treatment, they would show a hardness on the surface of 230 to 340, with decreasing hardness toward the center. The hardness would depend on the finishing temperature of rolling and the cooling conditions afterward. It is customary to use drill steel as it comes from the rolls without any heat treatment.

F. B. Foley.—Could not the same hardness, but more uniform results, be obtained by reheating these bars to above the critical temperature and air cooling? They are air cooled in both cases are they not?

N. B. Hoffman.—Yes, but if the bar were heated to the critical temperature and cooled naturally, you would sacrifice too much strength for softness. It would be better to heat to just below the critical temperature and cool in air. By this treatment, the working strains would be eliminated and the grain structure equalized without sacrificing an undue amount of hardness.

W. H. Schacht, Painesdale, Mich.—No mention has been made of the quality of the drill steel. I would place the quality first and the treatment second in importance.

As to the proper treatment, the most important factor is heating to the proper temperature, for both forging and tempering. This can best be done in a furnace equipped with means for controlling and maintaining the temperature. Since we installed such furnaces, and improved the methods used, there has been a reduction in the steel consumption and a 50-per cent. reduction in the number of bits sharpened. In addition, a reduction in gage was made possible, which has greatly increased the footage per drill shift. The reduced gage makes possible the use of a smaller steel which makes it less objectionable for handling. We start with 1⅞₁₆-in. gage and use 1/16-in. changes.

In spite of our efforts, at times the steel breakage more than doubles, although no change may have been made in the shop practice or in the brand of steel used. To determine the cause of this increased breakage, we cut thirty test pieces 45 in. long from six different brands, including the brand in use. These bars were not treated, they were simply cut to length and the ends squared by grinding. They were operated as drills by absorbing the blows of the steel by letting it strike against a 2½-ton block. One end of the steel was supported in a Waugh model 60 drill machine, the same as is used in the mine, and the other in a 1$\frac{1}{16}$-in. diameter hole drilled $\frac{3}{4}$-in. deep in the striking block. The time, in minutes, required to operate the drill machine until breakage occurred was recorded for each piece.

For most of these brands, the time required for breakage averaged about 15 min. The results obtained on each brand were quite uniform. All of the breakages were at the striking-block end of the steel and two-thirds were less than 5 in. from this end. This fact is significant when we consider that by far the greatest steel breakage in everyday practice is from 3 to 4 in. from the bit end.

We found that the brand of steel we were using, and on which the breakage had lately doubled, was the poorest, the other brands outrunning it from two to ten times. A few years ago, however, tests showed this brand to be the best steel, and it was so considered by many mines.

Tests to determine the quality of the cutting edge were also run on ten steels of each brand. All bits were accurately machined and carefully treated by the best known method. They were then placed in a heading of uniform hardness and all drills run to the same degree of dullness. Holes were started with regular starters to insure square bottoms, and the rate of penetration measured, bits being examined at intervals of 6 in. Here, again, the steel in question averaged less footage than the other brands; the averages in two cases exceeded the performance of this steel by 65 per cent. This brings up the question as to whether there should not be some standard or tests by which drill steel may be compared or judged as to these qualities. In this way the poorer steels now being sold would either be eliminated or improved. A recommendation from the manufacturer for best heat treatment for his steel may also be advisable, rather than leaving the treatment to the blacksmith.

An investigation of alloy steels is suggested in this paper. Both carbon and alloy steels were used in this test, and the two standing highest were carbon steels.

Various heat treatments have been tried at our shops, such as heating in a chloride bath and quenching in oil, brine, or water; also, tests with further treatments by drawing temper have been made, including heating in oil to different degrees of temperature after hardening. These experiments proved that nothing was to be gained by treatment after hardening, and that water at 80° to 90° was the best quenching medium for our conditions.

It has been suggested that the bit breakage may be remedied, to some extent, by overlapping the forging with the tempering heat, followed by quenching the cutting edge of the bit (allowing the steel just back of this point to cool in the air until it loses its redness) and then quenching the remainder, in this way getting a refined grain in the steel at this point. The advantages gained by this treatment will be more than offset by the extra breakage in the bit proper, as a result of the spalling or breaking of the wings because of this severe treatment. A better way, if such refinement is desired, would be to follow the forging heat by a secondary overlapping or refining heat slightly over the critical temperature, and allowing the steel to cool in the air, following this by a short tempering heat and quenching the entire bit. Such tests were made in our shops, and reduced the breakage by less than one-fifth of that obtained when the tempering heat did not overlap. Where the annealing, or secondary, heat was given, 80 per cent. of the breakage occurred in the refined-grain section and 20 per cent at the critical point, or where the refined grain changes to "as-rolled" grain. Where the secondary heat was not given, there were two refined-grain areas—one about 4-in. from the end and the other at the extreme end. The first was the result of the forging heat, where the temperature at the end of this heat was around 1400° to 1500° F., and the latter was caused by the tempering heat. The area between these refined-grain sections had a slightly coarser grain, such as is produced by a temperature of 1500° to 1600°, but not as coarse as the "as-rolled" grain. Picking, at random, fifteen broken bits each from drills having received our usual treatment and from those having received the secondary annealing, by splitting the broken steels longitudinally it was found that two-thirds of the breakages occurred in the refined-grain sections, 20 per cent. in the intermediate or coarser grained section, and the rest at the critical point. Out of thirty broken bits examined in both tests, only five broke in the "as-rolled" grain and three in the semi-coarse grain.

We conclude from these results that the point of breakage has shown a decided tendency to avoid, rather than seek, the coarser grained structure and that, therefore, the finest grained structure may not necessarily offer the greatest resistance to shock. In this conclusion, however, we must not overlook the fact (established in the shock test on plain, untreated bars) that the tendency was to break at this relative point. As to which of these two factors predominates, we do not know. It is our intention to determine this by preparing a set of pieces, the same as were run in the shock test, which we will heat to 1450° F. at points 1 or 2 ft. from the end, allowing them to cool in air so as to produce a fine-grained structure similar to that at the bit end at these points, yet without affecting the structure at the end of the bar. If the breakage of these bars shifts to these treated areas, it will show that the grain structure produced by the shop treatment is less desirable than that of the "as-rolled" bar.

Our observations show that much bit breakage, especially when drilling in hard ground, results from insufficient clearance angle on the bit. It is always advisable to make sure of this point before looking for trouble elsewhere.

B. F. Tillson, Franklin, N. J.—We have operated an automatic heat-treating furnace for the past nine months and have been well pleased with its work. The mechanism is necessarily complicated, for the period of time necessary to get sufficient heat into a piece of drill steel, within the limits desirable for speed and shop operations, is so great that a progressive, intermittent flow of material is required through the heating and quenching processes. Another factor is the desirability of quenching the drill steel over a bubble, in which there is actual circulation of the cooling medium, thereby avoiding the tendency of the steam to accumulate upon the faces or between the angles of the bits. The mechanism must therefore be so arranged that the steel is held over the bubble sufficiently long for quenching before it is gradually immersed farther in the water.

It is our practice to have the indicating pyrometer show a higher temperature in the furnace than what we estimate is the proper temperature for quenching the steel. Steel set at 1450° would come out of the mine too soft; and so we heat it about 1500° or 1550°, as indicated by the pyrometer. This subject is possibly more of academic interest because the process is taken care of by a machine with a certain fixed rate of speed; that is, it is operated by a variable-speed motor and the rate of speed can be adapted for different volumes of bits. So that with the signaling pyrometer installed on this equipment, the blacksmith must simply watch his fuel-oil burner to see that the signaling pyrometer is kept at the normal temperature.

C. R. Hayward, Cambridge, Mass.—The first feature of the Gilman machine that impressed me was the fact that it would receive any length of drill steel and would immerse it in the furnace atmosphere always to a definite depth. Furthermore, as the piece of steel emerges from the furnace, the mechanism causes it to drop quickly over a jet of water, where it is held stationary a few seconds, after which it moves slowly through the bath, being gradually immersed more deeply until it has become completely chilled. The depth that the steel is held in the water over the bubble is but a small fraction of an inch.

My only criticism would be that there might be a variation in the depth, or along the drill shank, that the original sharpening had caused increased grain size; or perhaps the drill steel, as received, was weakened by having too large a grain size.

It may be found in practice that the drill should be inserted a little farther into the heating chamber; that is a detail which can be worked out by experience. Of course if the carbon content in the drill steels varied, unsatisfactory results would be produced. But a company that would buy one of these machines would have a standard quality of steel and would have determined the proper critical temperature to which the drill should be heated. Of course, if the grade of the steel is changed, it is comparatively easy to change the temperature of the chamber.

About 150 pieces of a good grade of hollow drill steel, treated in this machine for use with my metallography class, were sawn through one side and split, thus giving the fracture along the longitudinal direction in the piece. The sawn part was then polished and examined under the microscope in connection with the fractured part. The steel was exceedingly uniform and possessed the characteristics I would prescribe for a properly heat-treated drill steel.

Two of the specimens, when fractured, showed elongated defects, the cause of which could not be definitely determined by the eye. They might have been impurities in the steel; they might have been longitudinal cracks. A microscopic examination seemed to indicate that stock was clean steel and that these defects were caused by bubbles, or blowholes, elongated in the drawing. As this was selected stock, these specimens showed that imperfections may occur even in properly heat-treated steel, and might cause difficulty when that steel is put into use.

C. C. Waite, Jersey City, N. J.—Fundamentally, the process for heat-treating metals developed by Snead & Co., of Jersey City, consists in passing an electric current through the piece to be heat treated, the resulting heat being due to the resistance of the piece to the passage of this current; the temperature is determined by the expansion of the piece under treatment. The application of the process is confined to pieces having practically uniform cross-section, such as rods and tubes, and its inception and development are the results of our having to heat-treat a large quantity of 1-in. 20-gage steel tubes about 10½ ft. long.

The tube to be treated is held vertical by jaws actuated by air cylinders at its top and bottom. The jaws are insulated from the machine and current is supplied to the tube under treatment from the secondary of a transformer through the jaws. The lower carriage, on which the lower jaw is mounted is, free to move downwards as the tube elongates during heating. The upper carriage is fastened but is adjustable to different lengths of work. The movement of the lower jaw changes the position of the fulcrum of a pointer used to indicate the dilation of the piece under treatment so that indications of dilation for a given temperature are maintained constant regardless of the length of the piece. This permits calibrating the indicator dial in terms of temperature.

When the piece has reached the desired temperature, the jaws are opened and the piece dropped into a quenching tank. A treadle is used to return and hold the lower carriage in a zero position for inserting a new piece. When using the machine for heat-treating thin-wall tubing, it is possible to quench without removing the tubing by passing oil or other quenching media through the tube, after which the draw-back temperature may be applied.

The critical temperature of the steel under treatment need not be known because the indicator shows the contraction that occurs when the steel passes through the critical temperature. Our observation of this volume change, using this method of heating, places the transformation 50° to 100° F. lower than when heat treatment is performed by ordinary furnace methods.

For work within its range, the process offers many advantages; such as the low temperature just mentioned, the speed of heating, which in most cases is less than a minute, practically no oxidation, the elimination of pyrometers, can be operated by unskilled help, good working conditions especially in warm weather, and uniformly high physical characteristics. For some classes of work, especially tubing, it serves as a means of inspection, as any pronounced variation in wall thickness will be shown by uneven heating. For the heat treatment of steel, we generally use about 3 volts per foot of length, which will give a current density of approximately 8000 amp. per sq. in.; these figures, of course, being different for the various stages of the process on account of the change in resistance due to the temperature change.

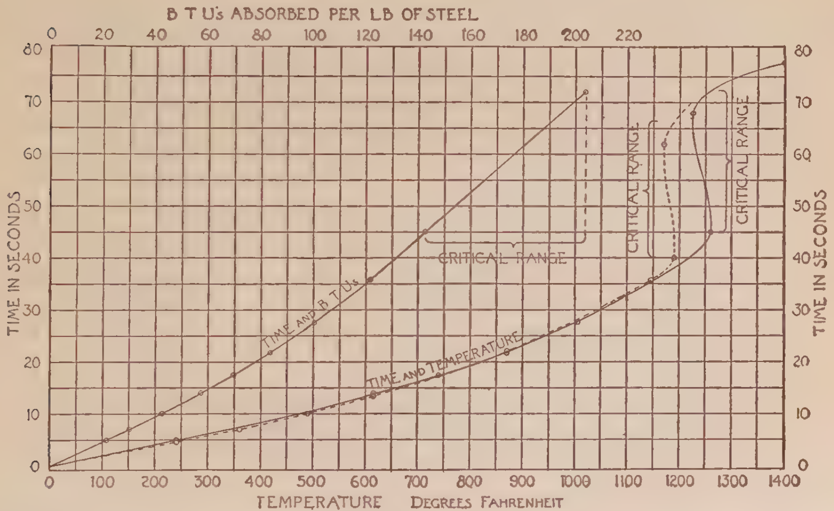


Fig. 2.—Average data obtained by heating ten rods of steel 3.45 per cent. nickel and 0.36 per cent. carbon.

Fig. 2 gives average data obtained by heating ten rods of steel having an average analysis of 3.45 per cent. nickel and 0.36 per cent. carbon. The ordinates represent time in seconds, the abscissæ, temperature in degrees Fahrenheit and British thermal units absorbed per pound of material. On the time and temperature curve, the critical range is well defined and the increasing slope of this curve before the critical period is interesting; the slowing up of the temperature rise, in part, is caused by radiation but to a greater extent by change in resistance, therefore, current density; the heating, of course, is proportional to the resistance and to the square of the current. The dotted curve represents the workings of the steel preparatory to a second quench and shows that the critical period occurs at a lower temperature. The temperature rises rapidly after it has passed the critical range, showing how easily the steel can be overheated in ordinary furnaces if a temperature-head is maintained. This curve also shows the necessity of having blacksmiths and other heat-treaters properly soak the material when using ordinary furnace methods to insure proper and complete transformation.

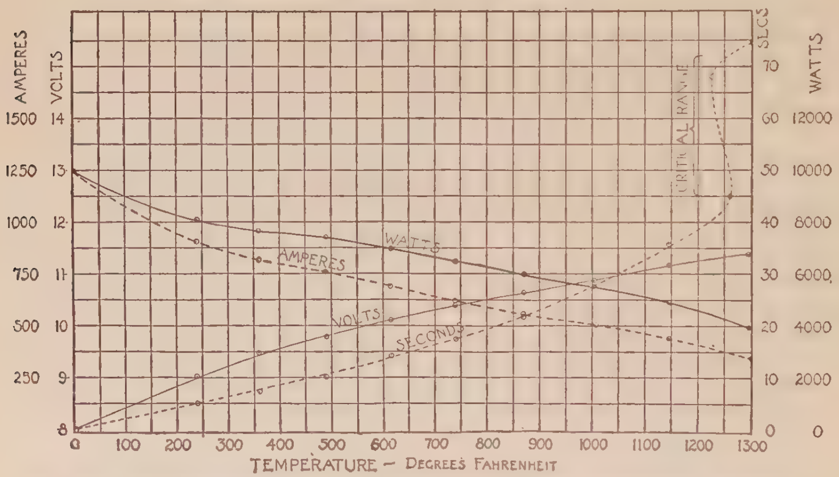


Fig. 3.—Power requirements of electrical heat-treatment method.

The time and British thermal unit curve shows the large part of the total energy that is necessary to carry the work through the critical period; of the 204 B.t.u. per lb. required to bring this metal through the entire heating range, 62 were absorbed within the critical period.

Fig. 3 shows the amperes, volts, time and watts required in the tests plotted against the temperature. This time is somewhat over one minute, as the process was slowed up to facilitate the taking of readings.

The uniformly high physical properties obtained are shown in Fig. 4, which compares the ultimate strength of S. A. E. No. 2330 steel, heat treated by this method, with the chart on page D-35 of the S. A. E. handbook, the steel having been quenched in oil and drawn at the temperatures indicated. A tensile strength of 237,000 lb. and an elongation of 10 per cent. was obtained by heat-treating a tube having an outside diameter of 2$\frac{3}{16}$ in. and a $\frac{1}{8}$-in. wall and a tube 1$\frac{1}{2}$ in. in diameter and a $\frac{3}{16}$-in. wall for the Naval aircraft factory of Philadelphia, the specifications calling for tensile strength 200,000 lb. and elongation 5 per cent. in 2 inches.

A series of 3.5 per cent. nickel, 0.35 per cent. carbon steel bars quenched and drawn at 630° F. (some having been double quenched and others carried well above the critical temperature or just past it and single quenched) gave the following physical properties:

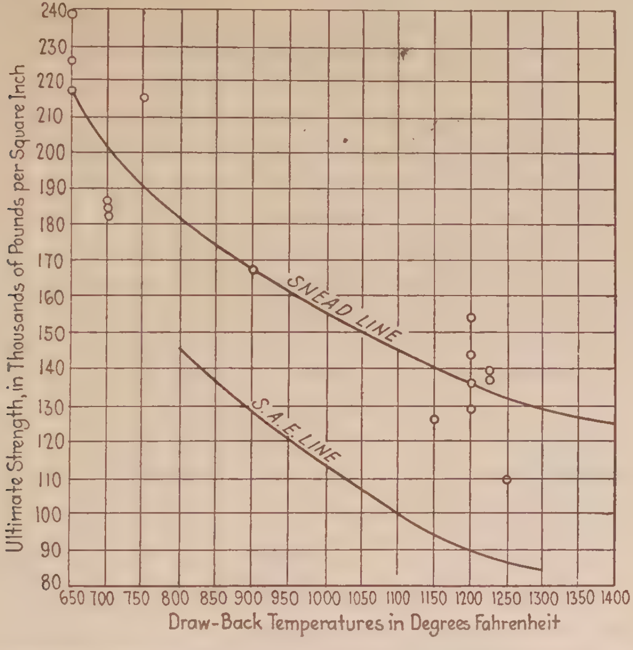


FIG. 4.—ULTIMATE TENSILE STRENGTH OF S. A. E. No. 2330 STEEL HEAT TREATED BY ELECTRICAL METHOD.

Ultimate strength, in pounds...........	229,250	238,900
Yield point, in pounds..................	197,600	210,350
Elongation in 2 in., per cent...........	9	11.5
Reduction of area, per cent............	52.1	54.0

The matter of cost is important and, as with all processes using electrical energy, the price paid for current is a large factor. However, we are able to obtain a high degree of efficiency as the machine has a minimum thermal capacity, so that practically the only losses, aside from those of the transformer and conductors, are due to radiation from the piece under treatment, and these are comparatively small for the time is very short. We require approximately 0.1 kw.-hr. per lb. of steel heated through its critical state. Even with moderately high-cost electrical power, the process will compare favorably with ordinary furnace methods on an over-all cost basis, especially where the work is put through in small lots or at irregular intervals. There are no stand-by losses, the machine being put into and taken out of service with all the ease and economical handling of an electric motor.

During the past week we received from Mr. Tillson several bars of drill steel but the short time before this meeting prevented our making more than a few simple experiments. Three of the bars were ⅞-in. quarter-octagon solid about 6 ft. long, supposed to be 0.90 to 0.95 per cent. carbon, that had been subjected to fatiguing percussive stresses by receiving the blows from a rock drill with the blunt end of the steel against a rock. This treatment was for a period of time slightly less than was necessary to break another piece of the same lot of steel. These bars were numbered 1, 2, and 3. Nos. 1 and 2 were normalized in our machine, No. 1 being carried well above the critical range and No. 2 just through it, both being allowed to cool in air. These two bars were returned to Mr. Tillson at Franklin for test. From the third one we cut about 1 ft. and normalized the remainder, the 1-ft. length being marked A and the normalized piece B. Tests and photomicrographs were made of these two specimens by the New York Testing Laboratory.

	(A)	(B)
Yield point, in pounds...........	78,200	55,400
Tensile strength, in pounds.......	124,100	102,200
Elongation in 2 in., per cent......	14	25
Reduction in area, per cent.......	18.5	39.4
	Crystalline dark core, flat fracture	Fine dark core, flat fracture

These photomicrographs we understand are consistent with the physical properties as shown by these tests. The time required to heat bar No. 1 was 3 min. 55 sec., bar No. 2, 3 min. 30 sec., the current ranging from 4960 to 4560 amp. giving current densities from 6760 to 6210 amp. per sq. in. A long time was required to handle these pieces because the transformer used was not of sufficient capacity. Although we have not had an opportunity to investigate, it has occurred to us that the time required to render a rock drill unfit for service through fatigue could be materially prolonged by giving the drill a full heat treatment, consisting of quenching and drawback, before it is put into service. The bars of drill steel were treated in a machine that held them in a horizontal position.

F. B. Foley.—Do you think that the annealing reduced the tensile strength of bar B to 102,220 lb. from 124,100 lb. by relieving the condition set up by this so-called fatiguing?

C. C. Waite.—Yes.

F. B. Foley.—The term fatigue as used here puzzles me. Steel is a crystalline substance with a definite arrangement of the atoms in planes, certain of which are weaker than others. When stressed beyond a certain point, called the elastic limit, there is a permanent offsetting of these planes in the crystal structure.

C. C. Waite.—Is it necessary to exceed the elastic limit?

F. B. Foley.—At least locally.

C. C. Waite.—That is what we believe.

F. B. Foley.—In steel that is heterogeneous, whether by reason of physical faults or chemical variations, there are local areas or spots that are weaker than others. Stresses below the measured strength of the section as a whole cause failure at these weak points. The discontinuity that results may be quite small at first but much less work is required, once such a fault has formed, to cause it to spread. It has been shown experimentally, a number of times, that stresses far beyond those that one might compute mathematically exist about the corners of notches in a stressed piece. In other words, stresses are not uniformly distributed in a uniform piece.

Professor Clayton and I have been examining drill steels that have been in use for a long time, trying to determine why, in a certain mine, short lengths fail more frequently than long ones. The shorter lengths are usually the pieces broken from longer steels or they have become short by the normal process of wear and redressing; in any event they are the pieces longest in service. We found small transverse cracks coming out from the water hole. Certain of these cracks spread during the constant battering to which the steel is subjected. Some bars are full of these cracks, sometimes occurring in patches and at other times distributed along the length of the steel. Apparently some are manufacturing faults and some have been produced since the steel was put in service.

B. F. Tillson.—Would not the working of the metal change the tensile-strength records?

F. B. Foley.—Yes, but the stress must be above the elastic limit.

B. F. Tillson.—Is not that another indication, possibly, that the work done by the steel has increased its tensile strength, as compared to its normalized condition?

In regard to fatigue: where we are at odds is possibly in the definition. As generally used, the term means that after a piece of steel has done so much work, it is less able to do more work, no matter what may be the cause. You say fatiguing is stressing beyond the elastic limit of the steel. It is just as well to say that any incipient flaw in steel reduces the area so that there is a greater tendency for the rest of the material to be stressed beyond its elastic limit; and as that flaw grows, the stress in the area or section being reduced is continually aggravated.

Is it not rather difficult to call drill steel a homogeneous structure? Is it not rather heterogeneous? It is composed of many alloys the conditions of which may vary with manufacturing service. Let us assume that a certain amount of work has been done on steel. Some of the alloys have a greater strength than the others. As all must receive the same uniform stress, unless there is some concentration of stress because of change in section or impurity in the steel, which merely aggravates it, can we not conceive that some of those alloys were stressed beyond the elastic limit while the others were not?

A day or two ago we tried to get some data regarding the effect of this method of heat treatment. Some 6-ft. bars of 7/8-in. quarter-octagon, solid, drill steel, containing 0.90 to 0.95 per cent. carbon, were sent to the Snead Co., one or two having been tested, before they were sent, by running them in an air-feed stoping machine until they broke. One piece, after running 85 min., broke into three pieces. Former tests show that these pieces could be run for 45 or 50 min. before breaking; but because of lack of time, these pieces were only tested for 35½ min. Two were then sent to the Snead Co., who heated one to about 150° above the critical temperature and then cooled it in air while the second was heated to the critical temperature and cooled in air. The one that had been heated 150° above the critical temperature, with the same drilling machine and same rock in which the other test was made, ran 83 min.; in other words, after being so heat treated, it ran a total of 118½ min. as against 85 min. that it ran in the state in which it was received from the manufacturer. It then broke 7 in. from the end of the shank.

The second steel broke after 61 min. of drilling, giving a total of 96½ min. of drilling, against 85 min. of the blank sample. The break in this case was 12 in. from the end of the shank, and showed a gradual, so-called, conchoidal fracture in the break.

Mr. Schacht said that in their accelerated tests they drilled against an anvil in a rock. I doubt if the work done on steel is properly indicated by drilling against so dense a medium as steel; it might be safer to make our accelerated tests with some other material. That is why we have tried to fracture the steel by drilling with the blunt end against rock.

I am curious to know if making a Brinell test on a piece of steel injures its service qualities; for instance, if the mines make continuous tests on any bars of their steel, will the resulting slight indentations tend to cause breakage at that point?

Charles Y. Clayton, Rolla Mo.—Inasmuch as manufacturers are making tests of automobile axles and articles of that kind, which are really delicate, I do not think that the tests would hurt drill steel.

B. F. Tillson.—Through the courtesy of A. R. Chambers, C. B. Archibald, and R. G. Watson, I have obtained data as to the use the Nova Scotia Steel & Coal Co., Ltd., New Glasgow, Nova Scotia, is making of a Thomson electric butt welder for the recovery of broken drill steels.

Mr. Chambers, who is consulting engineer, states that they have used steel with as many as fourteen welds in one piece. He does not think that any heat treatment has been attempted, as "they have not had success in heat treating steel at the mines, for it was difficult to get the furnace to give correct heats throughout its length; it is also difficult in the more or less isolated locality of the mine to interest men with a knowledge of the proper heat treatment. The welds stand up very well indeed," and he does "not know of many cases where steel welds have been broken in the weld. The way the breaks occur is usually because of small checks which start on the inside of the steel, due to the bending strain by heavy pressure between the machine and the face. These checks, once started, soon develop into breaks, and after a time there are so many in the steel it does not pay to weld the broken pieces." He "was in hopes at one time that a method might be arrived at to overcome these checks by means of butt welding, but nothing was attempted along that line."

Mr. Archibald, general superintendent, says, "It is difficult to form a definite idea as to the service performance of welded drill steel in comparison with new steel; it all depends on how long the welded steel had been in service before welding was necessary, or in other words, it depends on the crystallization of the used steel.

"In comparing the new steel unwelded with the new steel welded, we found no difference in the length of service. On several occasions we have had one or two steels out of a lot of new steel broken accidently after one day's service, and after being welded found them to stand up just as long as the remainder of the steel that went into service at the same time. Furthermore, in making up new steel we frequently have a lot of short ends left; these we weld together to make steel from 6 to 10 ft. long, and have found them to give equal service with any of the unwelded new steel.

"With regard to the welding of old steel, less than 5 per cent. of the breaks in this steel occur in the weld, the remainder of the breaks occurring between welds due to checking in the steel, caused largely by fatigue. We have 10-ft. steels working today with as high as twelve welds in them and they are still giving good service. If it were not for the welding, these steels would have had to have been discarded 6 mo. ago. Generally it has been found that welding has reduced our yearly requirements of new steel slightly over 50 per cent."

The cost of this electric butt welding is given by R. G. Watson, as follows:

We are figuring on 0.27 kw.-hr. per weld. This includes grinding both ends of the steel concave and dressing the weld when completed. Our best output for a week so far has been 500 welds. Considerable time, however, was spent in sorting out the broken steel in the steel dump, that is arranging two or more broken pieces to make up steels in 2 ft., 4 ft., 6 ft., 8 ft., or 10 ft. lengths. After the steel has been sorted, it takes from 3 to 4 min. to square the ends of steel, make the weld, and dress the weld when completed (3 min. if the fracture to be ground off is square, and 4 min. if the fracture is beveled). The actual weld is made in about 35 sec., that is from the time the current is switched on until completion of weld, and uses approximately 0.18 kw.-hr. I tried a piece of solid steel 1⅛ in. in diameter, and found the time required to make the weld is practically the same as for the 1¼ in. hollow steel. From this data the following may be calculated: 17 completed welds per hour equals 170 welds per day of 10 hr. 170 × 0.27 kw.-hr. = 45.9 kw.-hr. 45.9 kw.-hr. at 5 c. per kw.-hr. = $2.29 per day for power and with one operator at $4 per day, the total for welding would be $6.29 per day. This divided by 170 welds, gives 3.7 c. per weld plus whatever proportion of overhead should be added to this operation."

Mr. Chambers says, regarding the grinding of the ends, "If the ends are ground convex, the protruding portion, on welding, swells to the center of the steel tending to block the hole; the concave method overcomes this difficulty. To prepare the concave end, the steel is held against an ordinary sharp-faced emery wheel and rotated in the hands of the operator." Of course the diameter of the emery wheel should be small enough to produce the proper curvature of concavity.

The Nova Scotia Steel & Coal Co. has also supplied the following description of the method used in welding hollow rock drill steels. The method described has been found, by actual working tests of welded steels, to give very satisfactory results.

The ends to be joined should first be ground off fairly square and the rust or scale, etc., cleaned off fairly well for 3 or 4 in. back, in order to insure good electrical contact between dies of welder and drill steels. Then the two ends are countersunk with an 80° countersink to a depth of approximately ¼ in. This countersinking prevents the "flash" and the metal pushed up during completion of weld from filling or clogging the water hole.

The steels are then clamped in the dies of the welder, so that approximately ¾ in. projects from the inside edge of each pair of dies; or in other words so that there is about 1½ in. opening between the dies when the ends of the steel are brought together. There is no need of accuracy in this for ⅛ in. either way makes no difference. The steels should of course be in quite accurate alignment; this can be obtained by adjusting the front or rear, or both dies of either right or left clamp by the screws on both front and rear of clamp bodies. The weld is made by what is known as the part flash, part slow, butt method.

Arthur Notman, New York, N. Y. (written discussion).—The desirable qualities in drill steel might be listed as follows, in the order of their importance: (1) Maximum resistance to abrasion, resulting in the longest possible life to cutting edge and gage; (2) minimum weight per foot that will give a finishing size large enough to insure economy in explosives; (3) maximum resistance to breakage compatible with (2).

With the first quality present, two conditions result that increase the economy of drilling operations: Changes in gage can be reduced, thus decreasing the size of the starting bit necessary to reach the required finishing size, and the amount of rock cut by the bore hole; the length of run per change can be increased and the number of changes per unit of depth reduced.

With the first and second qualities both present, the least labor possible will be consumed in transporting steel to and from the working place and in handling to and from the machine at the face. For example, the weight of 1¼-in. round hollow drill steel is 3.98 lb. per ft., while that of 1 in. is 2.48 lb. per ft. Assuming that a face will require thirteen holes to break 5 ft., and that six sets of four steels each will be needed to reach the desired depth, using a 3-in. starter and 15-in. run, we would have, in the first case, 465.7 lb. of steel to be transported to and from the shop and lifted in and out of the machine; and, in the second case, only 290.2 lb. or a saving of 37½ per cent. in weight handled. A reduction from 1 in. round to ⅞ in. hexagon would show a saving of 19.1 per cent. in weight.

Similarly, if through improvement in the quality of the steel, the change in gage can be reduced from ⅛ to $\frac{1}{16}$ in. and the length of run increased from 15 to 18 in., the following economies would result: Assuming the same conditions as before, the reduction in gage would save 11.3 per cent. of the ground to be cut to reach the same finishing size. Only five sets of steel would be required for the round on account of the increased run, or a reduction of 16⅔ per cent. in the amount of steel to be handled.

The need for temperature control in the shop cannot be questioned but, so far as I am aware, no satisfactory pyrometer-controlled furnace is on the market. There is little literature dealing with the effects on drill steel of the various types of furnaces, fuels, atmospheres, temperatures, and routine used in its preparation. Naturally, the scope of the authors' report prevented their doing much more than carefully observing and noting these questions. An optical pyrometer for checking forging and tempering heats is better than only the eye of the operator. Perhaps the solution of the problem lies in the electric furnace; here both temperatures and atmospheres might be successfully controlled.

An article, which I think was published in *Compressed Air Magazine* a year or so ago, seemed to prove that no particular harm was done to drill steel of 0.80-0.90 per cent. carbon by forging up to 1950° F. If this is true, the allowable range for forging heat is greater than the probable error of the optical pyrometer; my own experience seems to support this.

In the matter of breakage, it is difficult to secure accurate figures, and it is doubtful whether the end would justify the expense. Breakage is largely in the control of the shop. For instance, in two mines using the same brand of steel of the same section at the same time, less than 2 per cent. of the steel sharpened in one mine was returned broken, while over 50 per cent. was returned in the other. Whatever the inaccuracies in the method of securing the records may have been, the obvious explanation is good shop practice in one mine and very poor practice in the other. In the first mine, the average drilling speed was 7½ in. per min., while in the latter, it was about 10 in. Possibly poor shop practice in the second mine was aggravated by using the steels after the bits had become dull, the ground being so soft that the drills could still drive them into the rock. No steel will last long after penetration stops. One frequent cause of breakage at or near the bit is the failure to keep dies, dollies, and gaging blocks in good shape. There is no economy in poor tools in the shop.

The handling of drill steel underground, in a large operation, is a vexatious problem, and greater attention is usually given to furnishing an adequate supply to faces rather than the development of a plan for carefully distributing the steel to each face. The lack of standardization in sizes and sections of steel used for the various types of machines is another difficulty, which no doubt, tends to increase the amount of steel lost.

The electric furnace seems to offer attractive possibilities for the forging and tempering of drill steel. It should be remembered that, in most cases, it will cost as much to deliver steel at the faces as it does to sharpen and temper. At the Copper Queen, handling 600 pieces a day through the shop, mining 30,000 tons of ore and driving 1,800 ft. of development work per month through three shafts and one adit, these costs are about evenly divided at 7½ c. per steel, each, for manufacture and transportation. At $5 per day for 7½ hr. in the face by the miner, the hourly rate is 66⅔ c. Money can be lost here very rapidly through ill-advised economies in the quality of the steel purchased, treatment at the shop, or poor distribution.

Copper Queen Practice.—Several years ago, at the Copper Queen, we realized that our steel was in poor shape. The first improvement was to insist that dies, dollies, formers, and gaging blocks should be kept up to specifications continually. We were indebted to our neighbors, the Calumet & Arizona Mining Co., for valuable suggestions in this regard. At that time, we were using the following sizes of steel: 1¼-in. round hollow lugged, 1-in. solid cruciform shankless, ⅞-in. hollow hexagonal collared, and ⅞-in. hollow hexagonal shankless. Experiments were begun to determine the most desirable design for our conditions. Some of the steel companies were strongly advocating the quarter-octagon section. While my knowledge of the cost of manufacturing hollow drill steel is very limited, I believe that this is the cheapest section to roll and also the most difficult in which to secure a uniform hole. We were convinced that a well-centered circular hole of uniform diameter was essential. The fitchery ground encountered in the Bisbee limestones requires a positive method of withdrawing the steel and the reduction of hole friction to a minimum.

With these conditions in mind, we decided to experiment with the 1-in. round, hollow, lugged section to determine whether it could be used in all types of wet machines without loss of economy. Theoretically, the round section is the strongest possible per unit of weight; to retain this advantage, it must not be weakened by careless work in forming the lugs. In the drifting machines using 1¼-in. steel, there was a great increase in drilling speed with the lighter steel, no increase in breakage, and a large saving in weight. In the plugger type there was no apparent loss in drilling speed in changing from the ⅞-in. hollow, hexagonal, collared steel to the slightly heavier 1-in. round hollow lugged. In fact, the rotation was improved through the better alignment resulting from the longer front thead required and the lessened hole friction as a result of the round section.

The use of stopers at the Copper Queen is confined almost entirely to the driving of raises. We were advised by the drill companies that the use of lugged steel in the stopers would result in great wear from the sludge in the front end; our experience has not confirmed this opinion, and we are thoroughly satisfied with the outcome.

Other changes that resulted from improved shop practice were the reduction in gage change from ⅛-in. to 1⁄16 in. and standardizing on a 15-in. run. The proper finishing size is still undetermined, further experiments being necessary to determine what effect the size of the bottom of the hole may have on the efficiency of the explosives used. This point is undoubtedly a matter of compromise between efficient drilling and efficient breaking.

Another point that is still in doubt, but on which we have some interesting preliminary results, is the relative life of cutting edge and gage of some competitive brands of drill steel now on the market. The results of these tests are shown in the accompanying table. Quantitative conclusions cannot be based on so few tests, but they are significant.

The tests are quoted from the report of Chief Engineer, H. M. Ziesemer:

> The first tests, or following figures, were carried on in 7 S.W. 11 drift, the ground being classed as A+ ground, consisting of 80 per cent. silica, cut by a very small stringer of iron oxides. This ground is fairly uniform and should determine the make of steel that would stand up best in our hardest class. Each length of steel was run to the point where the bit was either broken or the gage completely worn so as to make further drilling almost impossible.

	Time	Depth, Inches	Rate per Minute, Inches	Dullness	Remarks
			Steel No. 1		
Starters	1.10	5½	5.00	2	
Starters	0.07	1½	2.14	3	
Starters	0.87	2½	2.87	3	Shank broken
Seconds	1.05	4	3.81	3	One edge chipped
Seconds	0.95	4	4.22	4	Two edges broken
Seconds	0.54	2½	4.63	3	
Thirds	1.43	7½	5.24	4	One edge chipped
Thirds	1.43	9	6.30	3	One edge chipped
Thirds	1.62	12	7.41	3	
Fourths	1.02	4½	4.42	4	One edge chipped
Fourths	1.44	9	6.24	3	
Fourths	0.90	3	3.33	3	One edge chipped
Total	12.42	65	5.23		
			Steel No. 2		
Starters	0.39	1½	3.85	2	
Starters	0.97	3½	3.51	3	One edge chipped
Starters	0.87	2	2.53	3	One edge chipped
Seconds	1.03	4	3.94	2	
Seconds	0.52	2	3.85	2	
Seconds	0.60	3½	5.83	4	Two edges gone
Thirds	1.60	9½	5.93	2	
Thirds	1.07	5½	5.13	3	Two edges chipped
Thirds	1.32	6½	4.93	4	Three edges chipped
Fourths	1.73	4½	2.61	2	
Fourths	1.40	½	0.36	3	Two edges chipped
Fourths	Not run				
Total	11.50	43	3.74		
			Steel No. 3		
Starters	0.70	4½	6.43	2	
Starters	2.60	3	1.15	2	
Starters	1.70	2½	1.47	2	
Seconds	1.35	5½	4.07	2	
Seconds	2.82	10	3.53	4	Three wings gone
Seconds	0.40	4	10.00	2	
Thirds	0.50	1½	3.00	2	
Thirds	1.05	1½	1.43	2	
Thirds	0.84	3	3.57	2	Two edges chipped
Fourths	0.95	3	3.16	4	Two edges chipped
Fourths	1.05	2	1.11	3	Three edges chipped
Fourths	1.50	1½	1.00	3	One edge chipped
Total	15.46	42	2.72		

	Time	Depth, Inches	Rate per Minute, Inches	Dullness	Remarks
		Steel No. 4			
Starters	0.95	3½	3.69	2	
Starters	2.19	10	4.57	4	Three edges chipped
Starters	Not run				
Seconds	1.05	6	5.72	4	Two edges chipped
Seconds	1.25	5	4.00	4	Three edges chipped
Seconds	1.07	6½	6.07	4	Three edges chipped
Thirds	1.56	8½	5.45	4	Three edges chipped
Thirds	0.88	7	7.95	2	
Thirds	Not run				
Fourths	1.95	16	8.20	2	
Fourths	0.89	2	2.22	4	One edge chipped
Fourths	1.80	7½	4.17	4	Three edges chipped
Total	13.59	72	5.30		
		Steel No. 5			
Starters	1.15	6½	5.66	2	
Starters	1.75	9	4.62	4	Two edges chipped
Starters	Not run				
Seconds	0.83	4½	5.43	4	Two edges chipped
Seconds	1.25	9½	7.60	2	
Seconds	Not run				
Thirds	0.75	6	8.00	4	Two edges chipped
Thirds	0.90	6½	7.23	3	Two edges chipped
Thirds	0.80	4	5.00	3	One edge chipped
Fourths	1.20	8½	7.09	3	
Fourths	1.70	9½	5.58	3	One edge chipped
Fourths	Not run				
Total	10.53	64	6.08		

SUMMARY

	INCHES PER MINUTE	INCHES PER BIT
Steel No. 1	5.23	5.42
Steel No. 2	3.74	3.91
Steel No. 3	2.72	3.50
Steel No. 4	5.30	7.20
Steel No. 5	6.08	7.12

Dullness.—

2, denotes bit evenly worn, gage gone.
3, denotes bit evenly worn and chipped on wing or edges, gage gone.
4, denotes bit completely gone.

From these results, the choice would seem to lie between steel No. 4 and steel No. 5. Photostats of these bits arranged in the order to correspond with the figures accompany this report. These two makes of steel were then tested in uniform limestone, B ground, containing about 30 per cent. silica, where it was thought that the loss in gage could be determined together with the drilling speed. The loss in gage on all bits was so small that tests failed to show any results. The average drilling speed for steel No. 4 was 7.33 in. per min. for 169.25 in. drilled; for steel No. 5, 7.10 in. per min. for 162.5 in. drilled. Steel No. 4 shows a greater drilling speed per minute, which must have been due to some outside governing condition, as no noticeable wear could be seen in either steel as to gage or cutting edges. These same bits were then tested out in 7 S.W. 11 D, where the first runs were made. Each bit was allowed to drill 3 in., after which the loss in gage was measured.

	Steel No. 4			Steel No. 5		
	Gage Start, Inches	Gage Finish, Inches	Loss, Inch	Gage Start, Inches	Gage Finish, Inches	Loss, Inch
Starters	1.733	1.717	0.016	1.730	1.714	0.016
Starters	1.733	1.710	0.023	1.728	1.716	0.012
Starters	1.738	1.718	0.020	1.715	1.700	0.015
Average	1.7347	1.715	0.0197	1.7243	1.710	0.0143
Seconds	1.675	1.654	0.021	1.677	1.663	0.014
Seconds	1.656	1.638	0.018	1.677	1.655	0.022
Seconds	1.663	1.626	0.037	1.672	1.655	0.017
Average	1.665	1.6393	0.0257	1.6753	1.6577	0.0176
Thirds	1.632	1.597	0.035	1.630	1.606	0.024
Thirds	1.633	1.597	0.036	1.624	1.608	0.016
Thirds	1.627	1.605	0.022	1.630	1.607	0.023
Average	1.6307	1.599	0.0317	1.628	1.607	0.021
Fourths	1.556	1.517	0.039	1.558	1.524	0.034
Fourths	1.547	1.513	0.034	1.555	1.527	0.028
Fourths	1.551	1.525	0.026	1.555	1.523	0.032
Average	1.5513	1.5183	0.033	1.556	1.5247	0.0313
Average loss of gage per bit			0.0273			0.0211
Average loss of gage per inch drilled			0.0091			0.0070

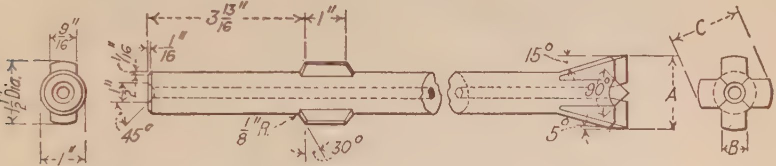
Breaking tests were run on these two makes of steel by allowing them to be continuously hammered by a machine against an anvil block until the breaking time was determined. The following results were obtained:

	Steel No. 4, Minutes	Steel No. 5, Minutes
1	20.38	30.00
2	28.00	12.00
3	20.20	114.00
Average	22.86	52.00

A test was also run on three lengths of steel furnished by the No. 6 steel company; these showed an average gage loss of 0.061 in. per bit, 0.0203 in. per in. drilled, and a drilling speed of 1.68 in. per min. in the ground where the other steel was tested. Granted that this steel was given the proper treatment, it compared very unfavorably with steel Nos. 4 or 5.

Referring to the summary of the first tests, steels Nos. 2 and 4 furnished by the same manufacturer, were of identical composition. No. 4, however, was given a normalizing treatment after rolling. Apparently the result of this treatment was to increase its drilling speed, during its life, 41.7 per cent., and its life 84.1 per cent. Steel No. 1 was also furnished by this company and was of the same composition, but it was subjected to an additional heat treatment after rolling.

Steel No. 5, which gave the best all-round results, was of Swedish manufacture. The centering and uniformity of diameter of the hole were superior to the three brands of American steel tested. Its good performance in these tests was checked by that of a 2-ton lot, the record of which was carefully kept for 10 months' continuous service in the mine. The Copper Queen is planning, in the near future, to confirm these tests with greater elaboration and a larger number of domestic and foreign brands of steel.



SIZE OF BITS

Length	A	B	C
2'-6"	1¾"	11/16"	1¾"
3'-9"	1 11/16"	"	1 11/16"
5'-0"	1⅝"	"	1⅝"
6'-3"	1 9/16"	"	1 9/16"
7'-6"	1½"	"	1½"

FIG. 5.—STANDARD, 1-IN., HOLLOW, ROUND, LUG STEEL FOR PHELPS DODGE CORPN. COPPER QUEEN BRANCH.

It is obvious that given good shop practice, the ordinary drill-steel troubles can be greatly reduced and that the final conclusion of the authors of this paper offers the basis for the first step in this direction. In other words, the management of the mining industry has failed to appreciate the importance of the drill-steel problem, and consequently, the shop foremen have not been given the benefit of technical investigation and advice. Second, although perhaps this may be a little premature, the drill-steel manufacturers (or some at least) have been content to make what their salesmen could sell, rather than try to determine for the mining industry the best possible steel for this purpose.

The American manufacturers of machine drills have been keenly alive to the problem of breaking rock, and to them is due much of the present agitation for better steel. Doubtless little more advance can be made in the art, unless the improvement in drill steels can be made to keep pace with that in the machines that use them.

R. G. WAYLAND, Lead, S. D. (written discussion).—In our work, we find that the main source of trouble is breakage just behind the bits, at the point at which the forging merges with the unworked steel. There is a change of structure here shown by a rougher, coarser fracture and by less hardness than that on either the bit end or the unworked steel. If the committee can determine how the sharp structural change at the bit and shank ends of the steel may be eliminated it will go a long way toward solving our drill-steel problem. In order that the accompanying diagrams, Figs. 6, 7, and 8, which show some of the results at our property, may be better understood, I will explain our practice briefly:

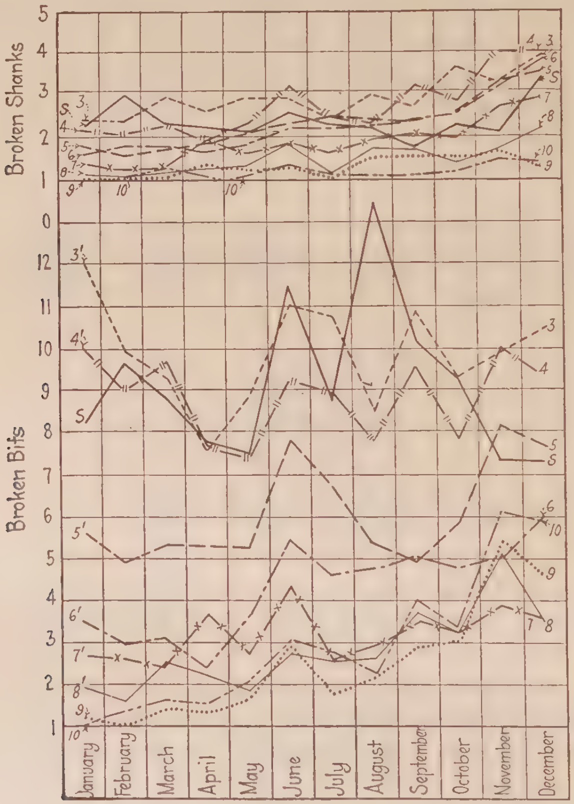


FIG. 6.—PERCENTAGE OF DRILLS, OF EACH SIZE, SHARPENED.

Drilling in drifts and stopes is done by mounted hammer drills, weighing 160 to 170 lb., equipped with anvil-block chucks. The steel is 1-in. quarter-octagon section and no lugs are required on the shank end. The gage of the starters is 2⅜ in. and the change for each succeeding length is ⅛ in. The second drill is 3 ft. long and each succeeding drill is 1 ft. longer than the preceding one. This makes the change in gage ⅛ in.

and the run of each drill 1 ft. In nearly all cases, the gage is worn off and the bit dulled in drilling 1 ft.; so that each sharpening represents 1 ft. drilled. The behavior of each length of steel is shown on Fig. 6. The total number sharpened ranges from 20,000 to 30,000 per month.

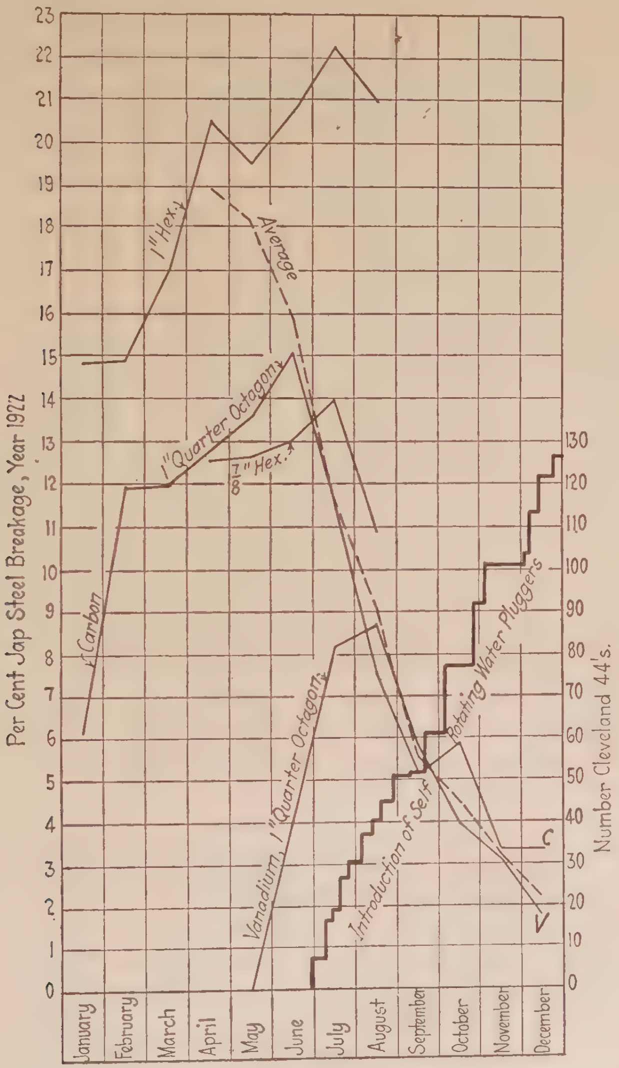


FIG. 7.

The steel known locally as "jap steel" is used in light unmounted self-rotating water drills that do the block holing of ore broken into blocks too large to pass through the chutes or into the cars. This steel is also 1-in. quarter-octagon, without lugs, and is made up chiefly of pieces too short for starters for the large machines. Every piece of steel 11 in. or longer but too short for starters is made into jap steel, using a 1⅜-in. rose bit.

Fig. 7 shows the percentage of jap steel broken each month of 1922. The 1-in. and ⅞-in. hexagon drills were replaced by 1-in. quarter-octagon during the first part of the year. During the year, the old hand-rotated dry pluggers were replaced by self-rotating water pluggers; the change began June 28 and was complete at the end of the year. In May, the total number of jap steel sharpenings was 46,000 and the total breakage was 8400. In December, the total number of sharpenings was 23,000 and the total number of drills broken 470. During this period considerable improvement was made in the drill-shop practice and the fins around the bit were eliminated, which undoubtedly helped reduce the breakage. When using hand-rotating machines, the men will allow the steel to bear on the side of the hole and throw undue stress upon it under impact; while they must hold the steel free in using self-rotating machines to prevent it from binding. I believe that this is the main reason for the reduction in breakage.

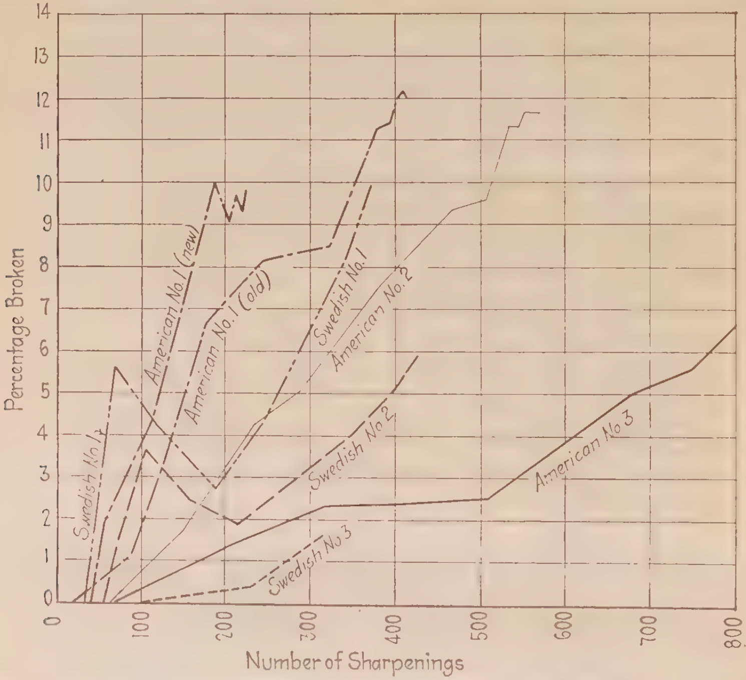


FIG. 8.—SPECIAL TESTS ON TEN BARS EACH OF SEVERAL BRANDS OF 1-IN., QUARTER-OCTAGON, HOLLOW, DRILL STEEL, SHOWING PERCENTAGE BROKEN IN RELATION TO TOTAL NUMBER SHARPENED. TESTS BEGAN NOVEMBER, 1921, AND WERE STILL IN PROGRESS ON SWEDISH STEELS JANUARY, 1923.

Fig. 8 represents the result to date of several test runs, of ten bars each, of various brands of hollow 1-in. quarter-octagon drill steel. These lots were each cut into about 25 pieces of steel in the 3-, 4- and 5-ft. lengths and used in extra hard ground until the breakage in a month reached 50 per cent. or more of the number of sharpenings. The curves are plotted on a basis of the percentage of the total breakage from the beginning of the test to the total number of sharpenings, which also is the total number of feet drilled, approximately.

In regard to breakage of the long steel at points between bit and shank, we have found that, in practically all cases, the corner of the steel has become nicked, through contact with rock or other steel, and that the break has occurred at that point.

For our conditions, we are not inclined to draw the temper of the bits after hardening, because we need all the hardness we can get by plunging in order to drill 1 ft. in the hard rock. Our practice is to allow the steel to cool in air after sharpening and afterward heat about ¼ in. of the point of the bit and plunge.

W. V. DeCamp, Jerome, Ariz. (written discussion).—An effort should be made to standardize the temperature of heating within certain limits, both for forging and tempering. Whether these temperature limits are fixed by pyrometer or otherwise is of no particular consequence provided a definite range is adopted. A definite method of tempering shanks with a definite type of oil and definite length to be tempered would be advisable, as well as instruction relative to maintaining a constant temperature for tempering oil.

There is great variation throughout the country in the treatment of drill steel; and the greatest difficulty will be met in trying to overcome the prejudice that exists among the various operators for their particular method of work. A large amount of literature might serve this purpose, but a demonstration in one's own shop by competent men would do more toward standardizing methods than any other one thing.

P. N. Hoorweg, The Hague, Holland (written discussion).—Could not an oil-treated steel, carbon or alloy, be used for the whole bit? By this treatment the whole bar is made tough, without losing its strength, while the hardening of one end of the bar will not affect the toughness of the other part, as we know from drilling bits in the oil fields. I would suggest selecting the kind of steel for these bits in this way:

Determine, first, the maximum force at which the different lengths of steel bars (full or reduced size) will break by torque. Say that one half this force is allowed, giving a certain strain in the steel bar with sufficient safety. Try the steel on a repeated-torque machine, which has a spring device for releasing the strain every time the force reaches one-half the maximum. This might be done with a speed generally run by the bit; count the revolutions until the bar breaks by fatigue. Do this with different kinds of steel and choose the material according to these results. These machines can be very simple and the laboratory can give us, in a cheaper way, the best steel that can be used. After having found the quality in this way, determine the best way of hardening it. We must, of course, consider both the percussion and the rotating movement of the bit. For the rotary movement the torque must be considered and for the percussion movement we must consider the tensile strength and shock.

STEPHEN M. HOLDEN, Sheffield, England (written discussion).—The forging temperature of 1475° F. appeared to me to be far too low, and without disclosing any of the facts I asked a prominent Sheffield firm (Samuel Osborn & Co., Ltd.) for a copy of heats on rock-drill steel. This company recommends a forging temperature approximately (950° C. or 1710° F.) in agreement with my deductions. A previous communication[3] furnishes abundant proof that with a temperature from 900° to 950° C. for forging it is possible to upset the bits with less power consumption and without injury to the steel, probably the contrary; and if water quenched and drawn, as outlined in that paper, would give satisfactory service.

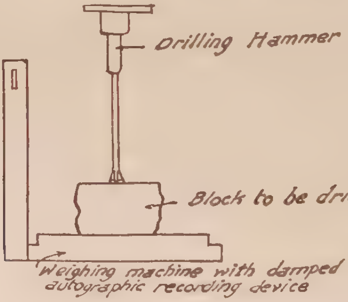


FIG. 9.

A temperature described as cherry red, and varying between 1450° to 1600°, leaves a big margin for any preconceived ideas of the smith, but many of the irregularities could be got over or eliminated by allowing the drills to cool and adjust internal strains, followed by reheating to approximately 1200° F.

In hollow steels, capillarity may tend to produce invisible water cracks unless the steel is quickly immersed below the length that has been heated.

In hand-guided drills, it is possible that many breakages are caused by wedging in the holes and the part played by vibration is of little consequence; but some useful information could be gained by an experimental rig, like that shown in Fig. 9, using various lengths of steel and varying the number of blows per minute to find the relation between points of maximum pressure on the weighing machine and the maximum pressure caused by the coalescence of vibrations and blow. Samuel Osborn & Co. recommend a Brinell hardness of 375 for shanks and 555 for bits.

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[3] J. H. Hall, A. E. Nisson, and Knox Taylor: Heat Treatment of Cast Steel, *Trans.* (1920) 62, 353.

E. F. Kenney, Philadelphia, Pa. (written discussion).—It is undoubtedly good practice to normalize the lugged shank, after upsetting, by an air quench after reheating to 1500°. We prefer 1500°, if the steel is not to be held, as at 1450° the mobility of the constituents of the steel is less and more time is required to effect same results. The amount to be heated for hardening will depend on circumstances. If lugs are not wearing, probably 2 in. will be all that is needed to prevent upsetting; if the lugs wear unduly, they should be included in this step.

It is useless to use cyanide in any form; it gives only a most superficial hardness, which is of no value Heating should be from 1425° to about 1475°, depending on the section, lower temperatures being used for light cross-sections.

Quenching should be in a good quality oil—cottonseed, fish, mineral quenching oil, or so-called soluble quenching oil. The oil must not be so thick as to prevent rapid transfer of heat, and should be maintained at reasonably constant temperature. This is easily attained by having the oil receptacle in a circulating water bath. Quenching should be vertical and the steel can be allowed to cool in the oil to its temperature.

For shanks, 375 to 425 Brinell is about right. If greater hardness is desired on the extreme end of shank, it is better to quench in a bath of oil, floating on an inch or two of water, as recommended by steel company A, than to heat the steel much hotter for quenching in oil alone.

H. S. Brainerd, Phillipsburg, N. J. (written discussion).—Table 1 shows the great variation in hardness actually existing in the field. No doubt, the difficulties some mines are encountering are attributed to the poor quality of the drill steel and the drilling machine, when the difficulties may be eliminated by correct heat-treating practice.

One of the essentials is the heating operation. The atmosphere of the furnace should be slightly reducing and the temperatures for forging and hardening carefully controlled. If the shank end of a drill steel is as hard as the piston, damage to both will result. Drill shanks of ⅞-in. hexagon hollow steel should be hardened at 1470° F. upon a rising heat in oil; for shanks of larger section a trifle higher temperature should be used. A Brinell hardness of 425 should result if the reading is taken from the end of the shank. For hardening, the shank should be heated for the same distance from the end that it was heated for upsetting, and then quenched entirely in oil. The overlapping of the forging heat by reheating removes the large grain size left after that operation, reducing the tendency to failure through the collar.

The right grade of oil must be used for quenching. Oils that are too heavy require too high a quenching temperature to obtain the correct hardening, while oils that are too thin make the steel too hard.

The curve for the oil quench, in Fig. 1, shows that for high temperatures, a ⅞-in. hexagon, hollow, drill-steel shank will become quite hard. As the extreme end of the shank, facing the striking end, becomes harder than the body of the shank, the curve was plotted from the hardness values obtained on this end, so that it might be compared with the hardness of the piston. The reason the author's experiments show a curve for hardness not so steep as mine, is that his Brinell readings were taken from the side of the shank instead of the end.

I should like to correct the author's statement that the oil-quench curve in Fig. 1 indicates that the proper temperature lies between 1400° and 1450° F.; my experience is that 1470° F. is the proper temperature for quenching shanks.

Charles B. Officer, Chicago, Ill. (written discussion).—I agree with the recommendation that the shanks of drill steel should be of just sufficient hardness not to upset in the drill; the proper hardness ranges between 375 and 400 Brinell. At one gold mine, there was considerable steel breakage on the shanks with high drill repair expense and less footage from the drills than could reasonably be expected. An investigation showed that the drill-steel shanks were being hardened in nearly the same manner as the drill-steel bits. After the method of tempering the drill-steel shanks was changed so as to produce a hardness about 400 Brinell the drill-steel breakage and the repair expense were greatly reduced. Under the old conditions, a hammer-drill piston would last between 3 and 4 weeks; since the method of treatment was changed, which was about 6 months ago, there has been no piston breakage.

Although better results may be obtained by varying the hardness of the shanks used in different style drills because of the variation in grade of steel of the same size, the human equation in the blacksmith shop and other factors of this nature, better results can be generally obtained by having only one hardness for all types of shanks and one method of treatment for obtaining this hardness.

When outlining a method for tempering the drill shanks, too much emphasis cannot be put on the selection of proper equipment and its correct use by the blacksmith. The equipment required is a suitable furnace, with a pyrometer for indicating the temperature, and a quenching tank with oil as a quenching medium. Blacksmiths should not be handicapped with furnaces that cannot be quickly and readily adjusted to give out varying amounts of heat nor with furnaces that do not overheat the steel but, because of excessive air passing through them or type of fuel used, will oxidize the steel or impart impurities to it. Pyrometers should be installed and the furnace run to obtain only the temperature desired. Without a pyrometer, the steels will be removed from the furnace at different temperatures from that desired, which will result in different hardness being given the shanks, as it is impossible with the changing light during the day to judge temperatures by colors correctly.

Frank H. Kingdon, Claremont, N. H. (written discussion).—In comparing the methods of heat treatment and temperatures used as shown in the tables, the absence of the chemical analysis of the various steels is a handicap. The conclusion that a Brinell hardness of 375 to 400 is best for drill-steel shanks will meet with general approval. A soluble quenching oil will assure best results, although it may be practical to use steaming hot water for the quenching bath. Annealing and air cooling after forging are important. Air cooling from the proper temperature gives the steel a structure usually corresponding to that of the body of the drill steel; however the forging temperature should not be too high.

When discussing the drill-steel shank, one must bear in mind that the life of the shank, even when properly forged and heat treated, is dependent on the cutting and reaming edges, shape and hardness of the drill-steel bit. Results of tests and experiments show that the dull bit is the cause of most of the trouble of the drill-steel shank, and also in turn the cause of most of the breakage of the drill-machine parts.

Heating and Cooling Curves of Large Ingots

By F. E. Bash,* Ch. E., Philadelphia, Pa.

(New York Meeting, February, 1923)

About three years ago, the writer presented a paper[1] on the rate of heating and cooling of a 24-in. round ingot. The present paper deals with work done on larger ingots at the plant of the Allis Chalmers Mfg. Co., Milwaukee, Wis., in August, 1919. Sufficient data on ingots of all sizes were desired to enable one to calculate in advance the rate at which any ingot could be raised to a forging temperature without injury to the steel, determining also the temperature gradient between different sections of the ingot and the length of time necessary to heat to the center. This has now been experimentally determined for 24-in. and 45-in. ignots by the writer and for a 18-in. cube by E. F. Law.[2] J. F. Harper has also made tests on heat-treating temperatures on 30-in. ingots; which data are included in this paper.

No attempt is made here to work out a formula for the heat penetration of steel, for too many factors are involved concerning which we have not sufficient information, as for instance, the specific heat of steel up to 2300° F. It is hoped that eventually a set of formulas will be developed so that one may calculate maximum allowable temperature differences within an ingot for any particular kind of steel and the rate of absorption of heat and temperature gradient within the ingot for a definite temperature head.

Experimental Ingot

The ingot tested was 45 in. in diameter, having been forged into the shape shown in Fig. 1 from 54 in. diameter. Seven ¾-in. holes were drilled 30 in. deep into the end, as shown in Fig. 2. The holes were made 7½ in. deeper than the length of the radius of the ingot so as to lessen the effect of heat penetration from the end on the inserted thermocouples, the hot junctions of which were at the bottoms of the holes. Holes 1 and 7 were drilled about 1 in. under the surface of the ingot, 4 was in the center, and the others were equally spaced between the surface and the center. The length of the ingot in the furnace was 8 ft. 6 in. The composition of the ingot was as follows: Carbon, 0.26 per cent.; manganese, 0.52 per cent.; phosphorus, 0.040 per cent.; sulfur, 0.039 per cent.; silicon, 0.19 per cent.; nickel, 3.57 per cent. A test was made with differential thermocouples on a sample cut from the ingot to determine the critical points.

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* Research Engineer, Leeds & Northrup Co.

[1] Forging Temperatures and Rate of Heating and Cooling of Large Ingots. "Pyrometry," 614. A. I. M. E. 1920.

[2] *Proc.* Iron and Steel Inst. (1918).

FURNACE

The furnace was oil-fired with regenerative chambers; the plan is shown in Fig. 1. Peep holes were provided at convenient locations, also holes for insertion of the thermocouples into the ingot. Directly above the ingot and also in the back wall, a 2-in. closed end "Usalite" porcelain tube was inserted as a protecting tube for a platinum-platinum rhodium thermocouple to determine the furnace gas temperatures. Two steel rods with $\frac{1}{32}$-in. nichrome sheet targets were welded in such a position that temperature readings could be made on them through the peep holes with an optical pyrometer. These readings were used as a check on

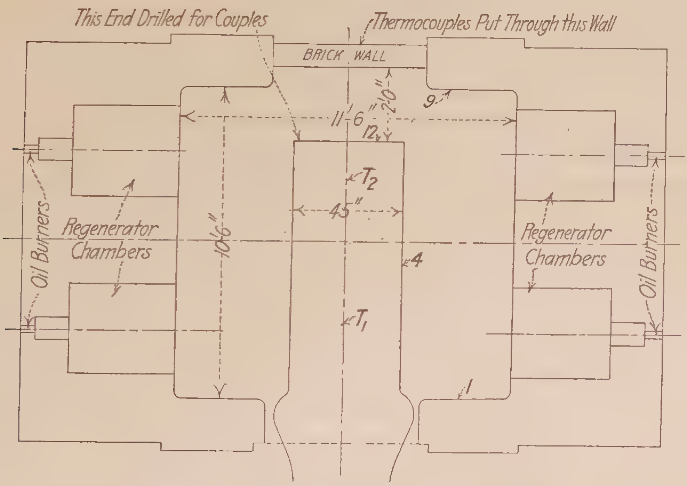

FIG. 1.—PLAN OF HEATING FURNACE.

the gas temperatures. The arrangement of targets and points where optical pyrometer readings were taken is also shown in Fig. 1.

APPARATUS

The thermocouples used were prepared from ⅜-in. extra heavy, wrought-iron pipe and No. 8 B. & S. gage pure nickel wire, welded inside the pipe and protected with porcelain insulators. The nickel-iron couple was used in place of the usual iron-constantan in order to be able to reach forging temperatures. It gives a fairly straight calibration curve with approximately 30 millivolts at 2400° F. The thermocouple temperatures were recorded with Leeds & Northrup potentiometer recorders and the surface of the ingot, target, and inner wall temperature were taken with a Leeds & Northrup optical pyrometer.

TESTS

The ingot was placed upon blocks while the furnace was cold and the thermocouples inserted and connected to the recorders. In addition to the couples in the ingot, two were laid on top of it, one near each end. In order to be able to withdraw the couples from the ingot when it was up to heat and also to protect them, sleeves made from 1-in. pipe were slipped around them and up against the ingot, where they were luted in place with fireclay and rigidly set in the brick wall of the furnace.

The furnace was started cold and the flames reversed every 20 min. during the whole test. The schedule of heating was worked out and the heater was instructed to follow certain lines on the recorder paper. This he managed to do fairly successfully. He was able to hold an average gas temperature within approximately ± 50° F. The log of the test is given in Table 1.

Little difficulty was experienced in running the test. A few of the thermocouples open circuited when the temperature got over 2000° F. but they were replaced.

It took approximately 17 hr. to bring the ingot from room temperature to forging heat, at which time the center was at 2130° F. and the skin temperature was 2260° F. It was held at approximately the same temperature for a few hours, until the day crew came on, when the thermocouples were withdrawn and the ingot pulled from the furnace and laid to cool in the open air. As couples 1 and 7 had oxidized heavily and, because of the settling of the support blocks, had been bent at the entrance to the ingot so that they could not be withdrawn, it was possible to get only the temperatures of points 4, 3, 2, and a contact point against the under side of the ingot, which gave outside skin temperatures that are somewhat lower than point 1 (1 in. under the skin) would have been. At the end of 18 hr. the temperature inside the ingot was between 600 and 700° F. and the outside skin temperature was 350° F.

TABLE 1.—*Log of Test on a 45-in. Nickel-steel Ingot*

Time	Minutes	Temperature, Degrees F.	Position	Remarks	
8:30 a.m.	0			Start	
2:48 p.m.	378	1903	T_1	On target	
2:49	379	1960	12		
2:51	381	1863	T_2	On iron rod—cut flame	
4:08	458	2075	T_1	On target	
4:09	459	1990	12		
4:10	460	1995	T_2	On rod	
4:10	460	1965	9	On edge of arch	
5:04	514	1975	T_1		
5:05	515	1903	1	On arch	Temperature of gas dropped 80° during these readings
5:06	516	2030	12		
5:07	517	1930		Floor under ingot	
5:08	518	1970	T_1	On target—dark	
5:08	518	1980	Pt.C		
6:47	617	2116	T_1		
6:48	618	2107	12		
6:49	619	2084	1	On arch	Temperature of gas dropped 30° during these readings
6:50	620	1965	4	On ingot	
6:51	621	2160	T_2		
6:52	622	2148	P	Back wall beside Usalite tube	
11:31	901	2378	T_1		
11:32	902	2477	1 arch		
11:33	903	2411	12 floor		
11:34	904	2463	back wall		
3:27 a.m.	1137	2249	T_1		
3:28	1138	2249	1 arch		
3:29	1139	2233	4		
3:29	1139	2160	12		
3:30	1140	2205	T_2		
3:31		2193	P	On Usalite tube	
9:40				Pulled ingot	

RESULTS

The heating curves are shown in Fig. 2, also temperatures taken with the optical pyrometer which, it will be noted, check closely with the platinum-thermocouple temperature curve *P* for the gas temperature. The gas temperature, after rising quickly to approximately 1200° F. at a point in the furnace directly above the ingot, was steadily increased at about the rate of 2° F. per min. to 1800° F. in 5 hr., at which time the ingot started to pass through the critical range. The gas temperature was then increased at the rate of 1° F. per min. until shortly after the whole mass was completely through the critical range, after which the

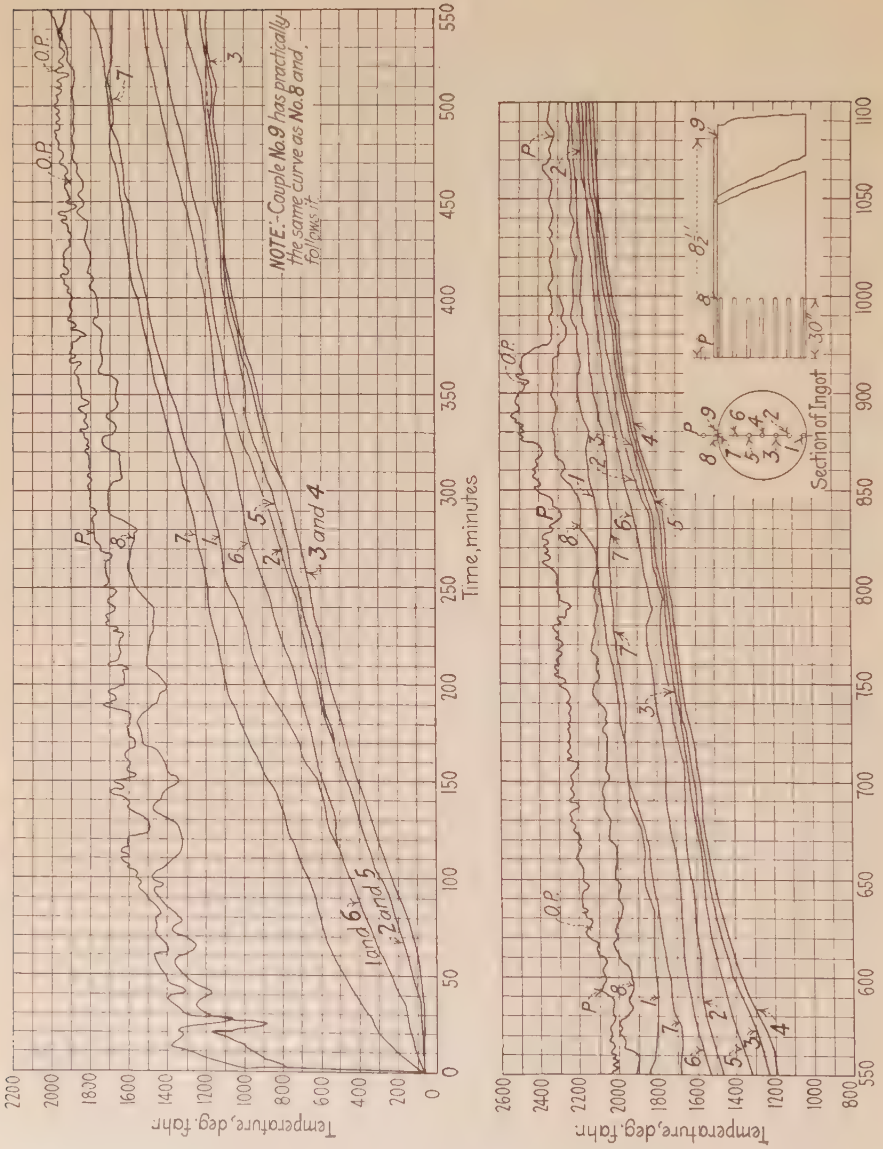

FIG. 2.—HEATING CURVE OF 45-IN. NICKEL-STEEL INGOT.

temperature was increased at the rate of 1.5° per min. up to 2500° F., when it was dropped to 2350° F. and held there until the ingot was ready for forging.

Curve 8 shows the variations of the temperature of the end of a thermocouple lying on top of the ingot and directly above and on line with the couples in the ingot. It follows the gas temperature much more closely than that of the ingot and apparently gives no indication of the critical point.

Referring to the heating curves of the couples just under the skin, the couple on top, 7, heated more rapidly at first although at the end the lower one, 1, became the hotter. These two couples give indications of passing through the critical range 5 hr. before the center (curve 4) was completely through. It is interesting to note also that the Ar_1 point on the small sample corresponded very closely with the Ac_1 point of the large ingot, which occurs at 1200° F.

In cooling, the curves of which were also plotted, the final passage through the critical point was approximately 200° F. lower than in heating. The cooling in the air from 2200° to 660° was at the rate of 1.5° per min. while the average rate of heating from room temperature to 2200° F. for the ingot was 2° F. per minute.

Oil was burned at the rate of 38.92 gal. per hr., making, a total for the 17 hr. in which the ingot was coming up to heat, of 661.64 gal. and 91,512,840 B.t.u. at the approximate heating value of 18,500 B.t.u. per lb. The oil had a specific gravity of 26.0° Bé., 230° F. flash point, and 330° F. fire point.

Heating Tests on 30-inch Ingots

In Fig. 3 are shown the heating curve and arrangement of couples for a 30-in. shaft which was brought up to 1500° F. in 20 hr. in an oil-fired furnace at the plant of the Allis Chalmers Mfg. Co., Milwaukee, by J. F. Harper.

Curve 5 shows the rate of heat penetration to the center of the ingot, the thermocouple being inserted in a hole 15 in. deep. Curve 3 is for a couple 5 in. deep and curves 4 and 9 are for contact couples near each end, as shown. In the cooling curves for the same ingot cooled in the furnace, the passage through the critical range is well marked on both heating and cooling for couple 5 but is not so clear for couple 3. The probable reason is that the outside diameter of couple 3 was only ¾-in., while the hole was 1½ in. diameter, and only 5 in. deep, which would allow the hot gases of the furnace to enter.

In a heating curve for a similar ingot with one couple 15 in. deep in the end and others in contact at different positions as indicated, the critical range is noticeable on curve 5 but not particularly marked by contact couples. However, on the cooling curve for the same ingot the passage through the critical range is very clearly shown. It apparently depends on how carefully the couple is held against the work as to the sharpness of the break in the curve.

These curves give a good idea of the difference in temperature of the skin and the center of an ingot of this size and the length of time necessary to bring them all to a constant temperature.

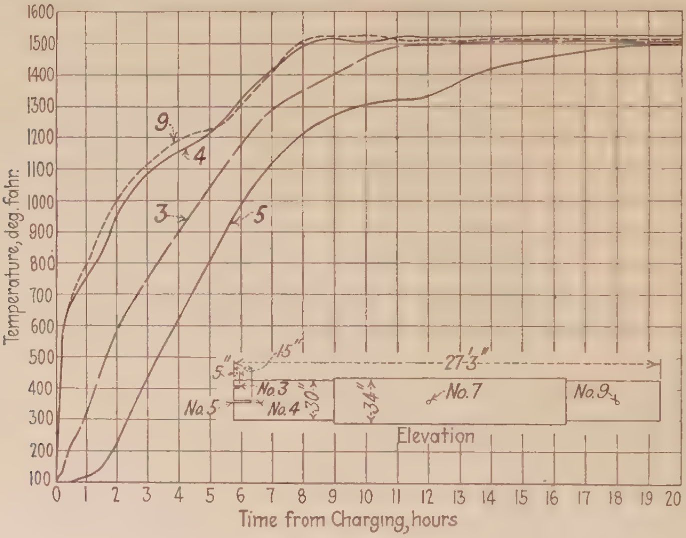

Fig. 3.—Heating curve of 30-in. ingot.

In concluding the writer hopes that these data may serve as a basis for further experiments and calculations, which will eventually enable the steel man to determine exactly how fast he can heat any size ingot without injury. This information is known empirically for certain practices but should be scientifically worked out for all conditions and kinds of material. The writer wishes to thank Mr. J. F. Harper, Mr. H. Freeman, and the Allis Chalmers Mfg. Co. for their kind assistance and coöperation and Mr. R. C. Drinker for his interest and suggestions.

DISCUSSION

William J. Priestley, Midland, Pa. (written discussion).—The fact that the experiment was made with a 45-in. bloom, that had been forged from a 54-in. ingot, should be borne in mind when comparing the heating curves with those where corresponding masses are heated in the ingot form. In the bloom used, it is quite probable that the original coarse crystal grains of the piece had been broken up in forging from 54 to 45 in. in diameter, and more rapid heating might be safely done than in the case of an ingot 45 in. in diameter.

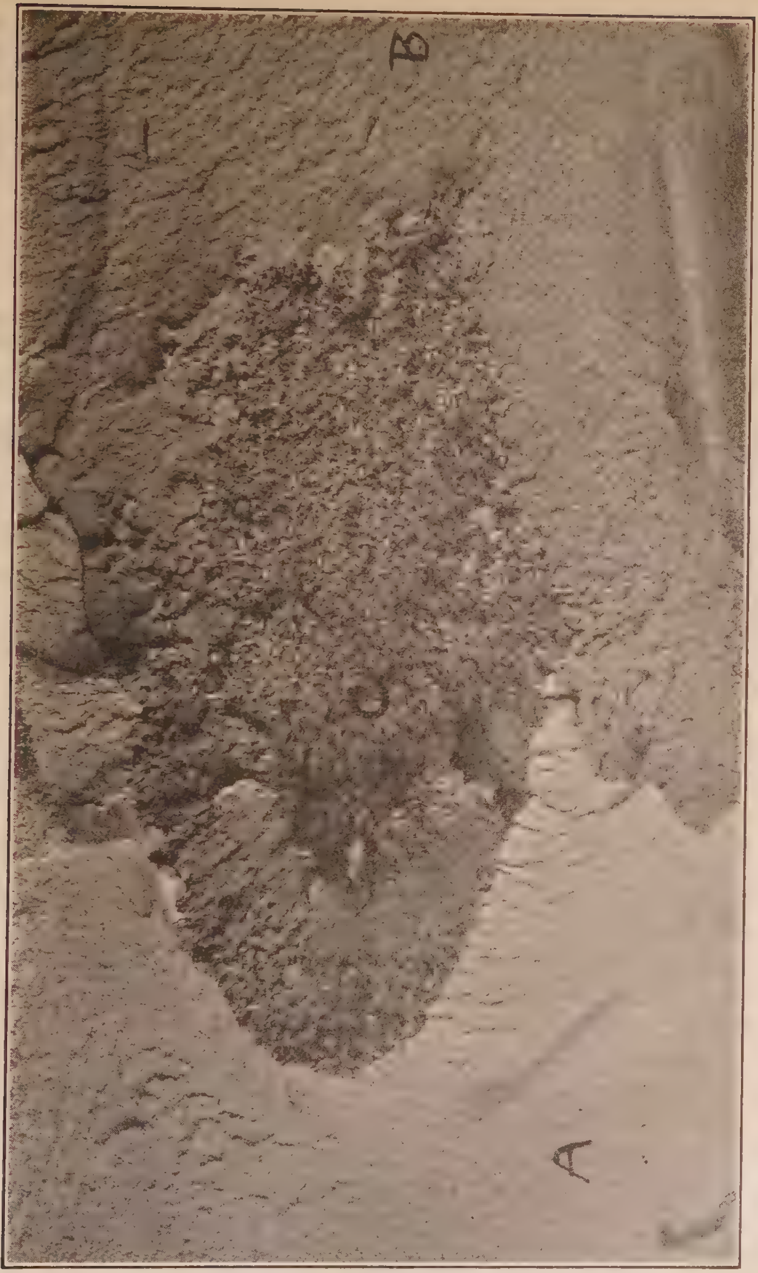

FIG. 4.—SHATTER CRACK IN 20-IN. ARMOR PLATE.

Too rapid heating of large ingots of 3½ per cent. nickel and other alloy steel frequently results in internal tearing in the center. These transverse tears are caused by the exterior of the ingot expanding more rapidly than the interior and are commonly called "shatter cracks." When an ingot is forged, these shatter cracks are elongated and, when detected in shafts of blooms, are not uncommonly mistaken for secondary pipe. It is understood this is what the author had in mind when he refers to the injury done to steel by too rapid heating. This injury is not as likely to occur in heating blooms for reforging.

The writer has seen machine-finished shafts 16 to 24 in. in diameter fall apart as a result of too rapid heating of cold ingots. A good safeguard against this condition may be taken by bore searching through a 4 or 5-in. hole drilled through the center of the forging. This was recently done on the large 30-in. forged steel columns 60 ft. long, for the 14,000-ton forging and bending press erected at the Naval Ordnance plant.

It will be noted on the curve, Fig. 3, there is a difference of 500° F. between the inside and outside of the forging after 5 hr. heating, and 400° F. difference after 10 hr. This may be satisfactory for heating reforgings, but it seems undesirable for safe heating of large alloy-steel ingots. It would be more cautious to raise the furnace temperature more slowly. If raised at the rate of 100° F. per hr. up to 1200° F. and held at this temperature until the ingot became thoroughly "soaked out," there would be less difference in temperature between the center and the outside of the ingot and less danger of producing internal tearing. After soaking the ingot uniformly at 1200° F., the temperature might be increased with safety at the rate of 1° F. per min. up to the final forging temperature.

In forging 58 and 68-in. nickel-steel ingots, it has been found most satisfactory to charge into the heating furnaces as soon as possible after removing from the molds. With the furnace at about the same temperature as the ingot, heating may safely be done at the rate of ½ hr. per in. of radius, or in terms of temperature at the rate of 1° F. per minute.

When cold ingots of alloy steel of this size are heated for forging, greater care should be taken. It has been found most satisfactory to heat no faster than approximately 1 hr. per in. of diameter. The heating up to 1200° F. should be slow and gradual so as to have as little difference in temperature as possible between the outer and inner parts of the ingot. From 1200° F. up to the forging temperature, the rate may safely be 1° F. per min., with a period of thorough soaking out before forging.

Fig. 4 shows a shatter crack in an armor plate 20 in. thick that developed during the heating of a 36 by 120-in. ingot; *A* and *B* represent the metal in the exterior surface of the ingot, *C* the pocket formed by the shatter crack in the interior of the ingot. Note the fine-grained structure at *A* and *B* and the coarse-grained ingot crystals at *C*.

Continued Discussion on the Physics of Steel

By William R. Webster, Philadelphia, Pa.

(New York Meeting, February, 1923)

The unusual interest taken in the papers on steel at the New York (1922) meeting showed that the time is ripe for the renewal of the general discussion of the physics of steel, on the same lines that were used formerly and produced such good and lasting results. The curves shown in Figs. 1 to 4 and the appendix were added to my paper[1] after the February meeting, but, unfortunately, Figs. 1 and 3 were transposed; the curves on page 252 are for bessemer steel and belong on page 254; while those on page 254 are for open-hearth steel and belong on page 252. These errors are evident at once on referring to the data in Tables 1 and 2, pages 227 and 228.

The curves for each 0.05 per cent. of carbon and manganese in basic open-hearth steel are given in Figs. 1 and 2; these lines for the lower carbons and manganese are comparatively uniform.

Figs. 3 and 4 give similar lines for carbon and manganese in bessemer steel. The large break in the carbon lines from 0.55 per cent. to 0.60 per cent. is caused by much higher silicon in the steels of 0.60 per cent. carbon and over that are used for springs, etc.

The estimated ultimate strength of pure iron, and the increase in strength by carbon, phosphorus, and manganese, with the different views regarding the same, were given and discussed. All of this can be considered as an introduction to the practical use that is now being made of such data in selecting the heats of steel best suited to give the required physical tests called for in the finished rolled material.

The chemical composition of the steel, of course, tells only a part of the story, as many other factors that have an influence on the physical properties of the steel must be considered, and controlled as far as is possible. Some of these have already been referred to, but there are other important ones to be considered and discussed.

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[1] Application in Rolling of Effects of Carbon, Phosphorus, and Manganese, on Mechanical Properties of Steel. *Trans.* (1922) **67**, 220–242.

All of these matters are now much better understood than formerly and many of them are being standardized in practical every-day work by the general use of better heating furnaces and the control of temperatures by the use of pyrometers instead of by the eye. The writer in his early work found the necessity of taking into consideration the changes in structure and physical properties of steel in rolling and forging. He gave great importance to this in his paper before the Iron and Steel Institute in 1894. The following[2] is quoted from the written discussion of this paper, as it has a direct bearing on the important matters that we are now considering in our discussion:

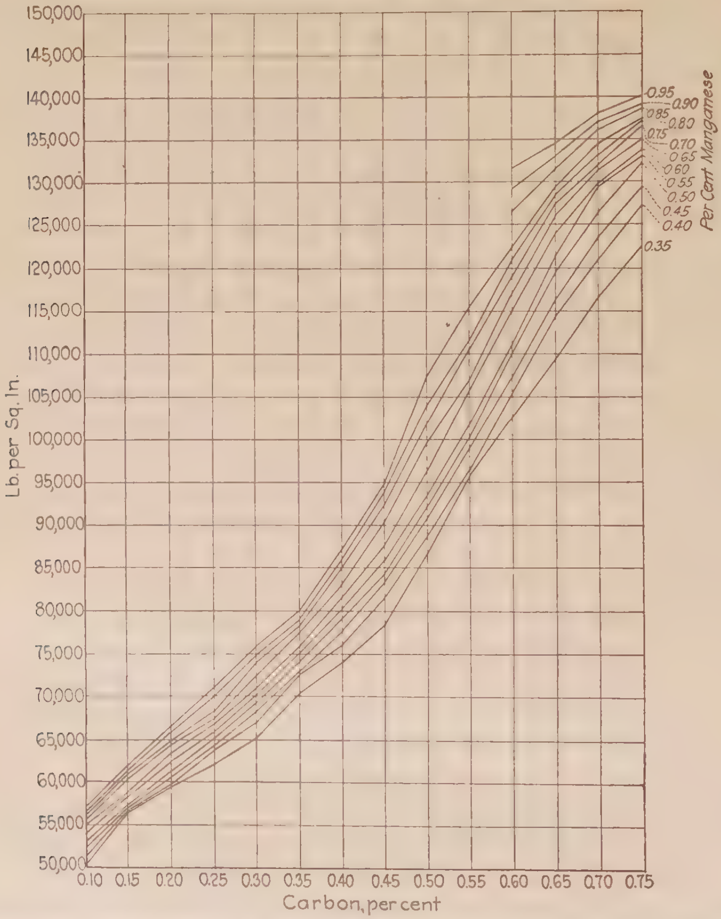

FIG. 1.—AVERAGE RESULTS OF TESTS ON ¾-IN. ROUND BARS ROLLED FROM TEST INGOT OF BASIC OPEN-HEARTH STEEL WITH PHOSPHORUS UNDER 0.031 PER CENT.

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[2] *Jnl.* Iron and Steel Inst. (1894) No. 1, 328.

Mr. P. C. Gilchrist, Member of Council, considered that the paper was one of the most important ever received by the Institute. It drew attention in a scientific manner to a great fact that had been well known to all observers who had been connected with the production and manipulation of steel, the great fact in question being that within certain limits the chemical composition of a piece of steel was of less importance than its physical state. In plain English, that meant that in the case of a plate you could make it hard and, in an extreme case, as brittle as a carrot by improper heating or rolling, and that you could make another piece from the very same plate almost as tough as leather.

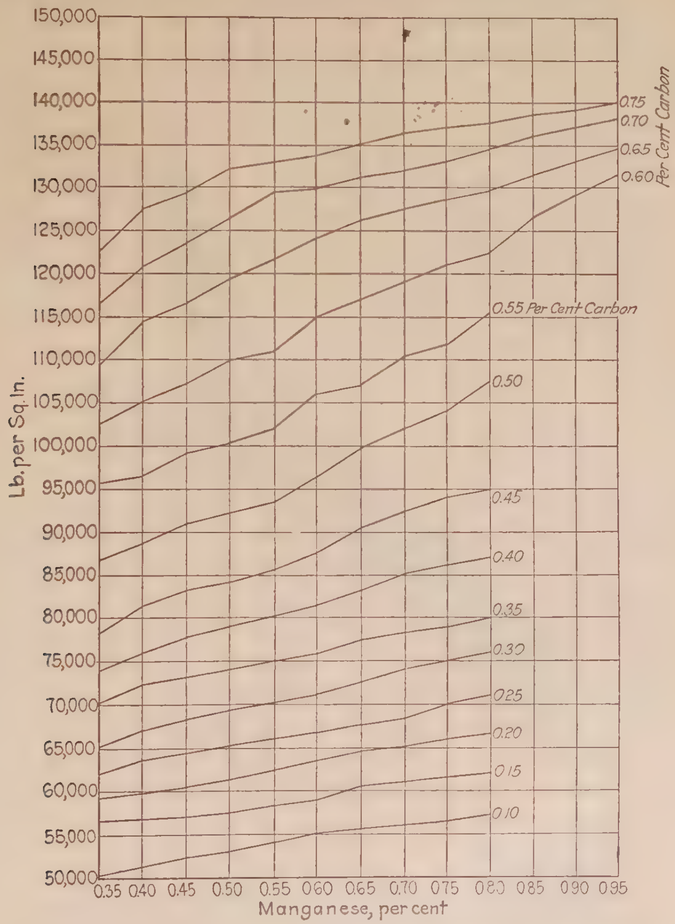

FIG. 2.—RESULTS OF CARBON VARIATIONS IN STEEL SHOWN IN FIG. 1.

STEEL MANUFACTURERS' PRACTICAL WORKING TABLES USED IN GRADING STEEL

In most of these tables, no attempt has been made to consider separately the effects of each element, but the combined effects of carbon, phosphorus, and manganese on the tensile strength and other physical properties of the steel are considered. These tables are now generally based on the results of tension and bending tests of rolled steel of different weights and thickness that have given satisfactory results and complied with the physical and chemical requirements of the standard specification on which they were rolled. Thus by referring to the tables, for similar new orders, the chemical composition of the heats of steel to use are known at once. This was made possible, as has been explained, through the standardization of our steel specifications by the A. S. T. M. in coöperation with other societies.

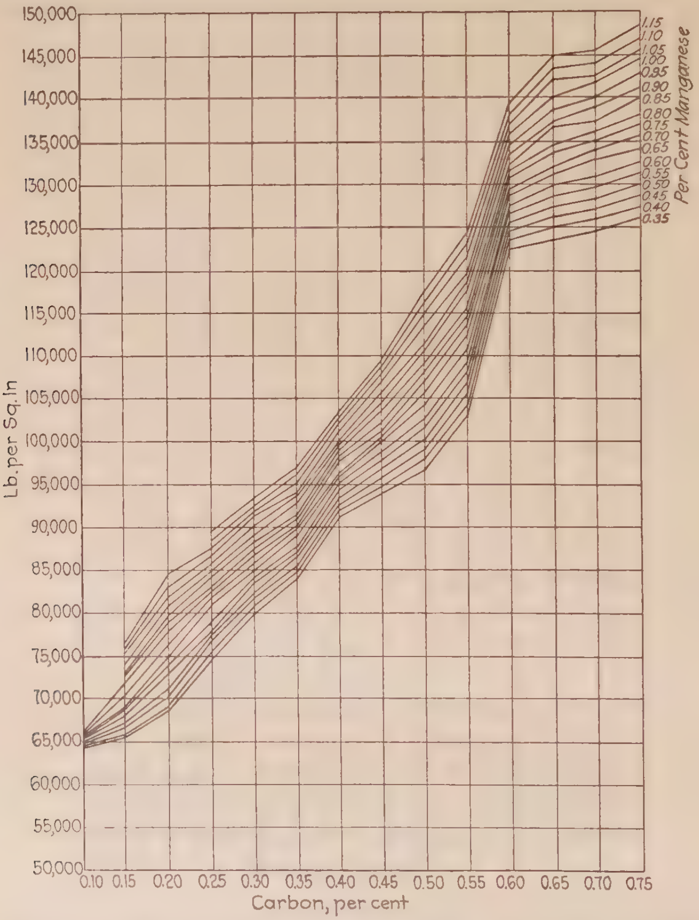

FIG. 3.—AVERAGE RESULTS OF TESTS ON ¾-IN. ROUND BARS ROLLED FROM TEST INGOTS OF ACID BESSEMER STEEL.

These tables only tell a part of the story and generally will apply only to the rolled material of the mill from which the results on which they were based were obtained. This is now well understood and every endeavor is made to keep all of these working conditions standardized as far as possible. Yet, notwithstanding all this, at times every mill has trouble with its steel; generally the cause of such trouble can be located, but in some cases it cannot.

We now need for permanent record some of these working tables as used in order to show just what advances have been made along these lines in the past thirty years; in addition to this, information as to how these tables were arrived at, the necessary changes that have been made in them from time to time and—most important of all—some details of the troubles manufacturers now have with their steel for which the causes cannot be located. This would add very much to the interest of the subject and give us valuable matter for discussion and further research.

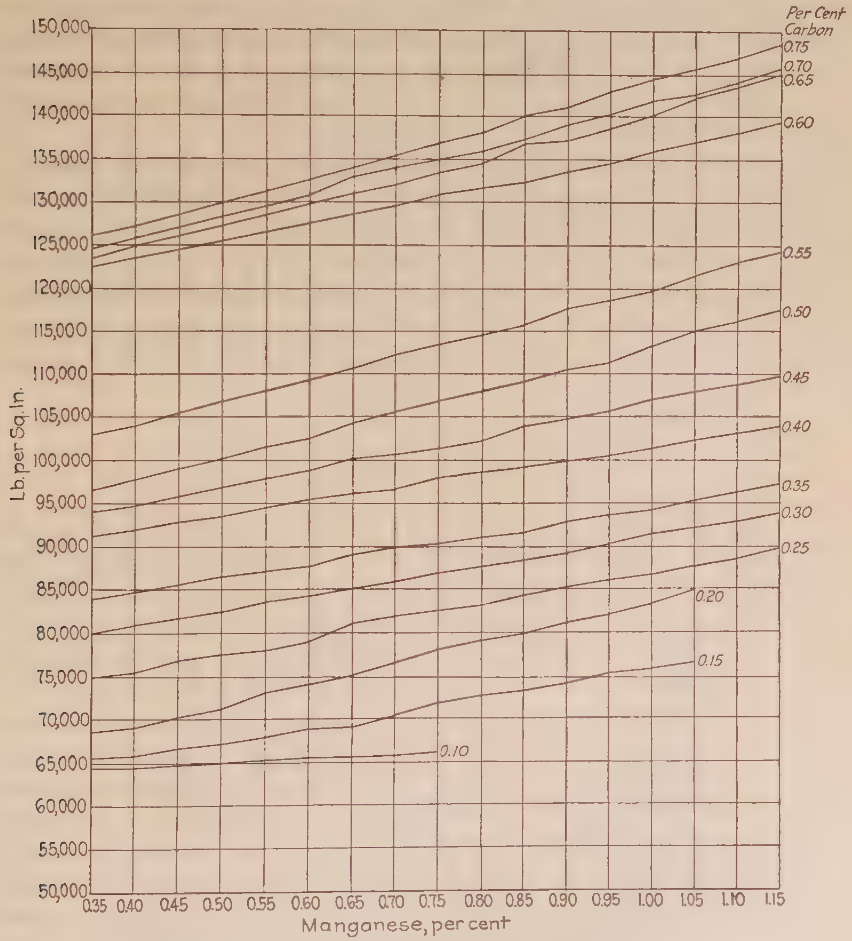

FIG. 4.—RESULTS OF CARBON VARIATIONS IN STEEL SHOWN IN FIG. 3.

Some of these matters were treated in a general way at the session on steel last February, others were treated in detail, giving what was really necessary to improve the quality of open-hearth steel. In addition to this, the advantages of the electric furnace in producing a better and cleaner steel than could be produced by any other process was fully covered. We naturally look for additional information from those who have been using the electric furnace and are gradually applying the information acquired from its use to the other makes of steel at their works and thus materially improving the quality of same. This important matter will, no doubt, be taken up and discussed by those who have detailed information regarding the same.

Up to the present time, the papers referred to have not been discussed as fully as they deserve but when this is done, much valuable information will be brought out, which, in turn, will result in additional papers as a very wide field of research work has been opened up. Of course, we are still handicapped by the need of good, quick methods for determining the oxides and gas content of the steel, but it is to be hoped that these will soon be developed and then we can look forward to very much greater improvements in the manufacture of steel.

DISCUSSION

Carle R. Hayward, Cambridge, Mass.—When preparing test specimens for various researches, I have found that pieces of ¾-in. rolled steel taken from the same bar and given identical heat treatment, varied in tensile strength sometimes more than 10 per cent. With a thoroughly normalized steel, almost identical results can be obtained; but with specimens that are quenched or quenched and drawn, it is difficult, even with very careful procedure, to get three specimens that would give absolutely check results. Whether this is caused by some slight variation in the heat treatment procedure or by slight imperfections in the tensile specimen, or by something in the steel itself, I am unable to say. This has occurred particularly with medium- or low-carbon steels. If this is true in small specimens that were carefully treated, it would be magnified in the product of the mill; and if we cannot depend on our tensile specimens, how are we going to arrive at definite conclusions in regard to the properties of the output of any given mill?

A. E. White, Ann Arbor, Mich.—What were the compositions of the steels tested; that is, were the tests made on steel with 0.10, 0.15, and 0.20 per cent. carbon, etc.? Was the steel in the as-rolled condition when tested? What was the initial size of the ingots into which the metal was cast, and the degree of reduction that it underwent in the rolling operations?

George K. Burgess, Washington, D. C.—My impression is that the specimens were in the as-rolled condition. I think it is a safe assumption that the actual percentages of composition could not have been exactly one-tenth, and so forth, but the specimens were regrouped and apparently curves drawn to the average values.

Albert Sauveur, Cambridge, Mass.—Unless saturated ingots of the same size are subjected to the same reduction, conclusions drawn from the physical properties of the different steels cannot have much value.

William R. Webster (author's reply to discussion).—Those who discussed the paper overlooked the fact that the plotted results are from 3/4-in. round bars rolled from test ingots, both for basic open-hearth and for acid-bessemer steel, as noted in the diagrams. These small bars, of course, were tested as rolled. A description of these bars is given in the original paper.[3]

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[3] *Loc. cit.*

Influence of Temperature, Time and Rate of Cooling on Physical Properties of Carbon Steel

By Henry M. Howe,* Bedford Hills, N. Y., Francis B. Foley,† Rolla, Mo., and Joseph Winlock,‡ Philadelphia, Pa.

(New York Meeting, February, 1923)

This investigation was undertaken for the purpose of determining, in a systematic way, the effect of the rate of cooling of steels, heated to above the transformation range, on their various mechanical properties. This paper gives the results obtained in the investigation of the heat treatment of carbon steels carried out under the auspices of the Committee on The Heat Treatment of Carbon Steels of the Engineering Division, National Research Council. The objects of this committee, of which Dr. Henry M. Howe was chairman, were to increase knowledge of the influence of heat treatment on the mechanical properties of carbon steel. The method pursued was to subject specimens of steel of the carbon contents most used for engineering work, 0.34 per cent., 0.52 per cent., and 0.75 per cent., to various heat treatments, and to test the mechanical properties thus induced. The steels were the same as were used by the committee on fatigue phenomena. This project was financed jointly by the U. S. Bureaus of Mines and Standards; industries and university laboratories have coöperated. A large part of the work was done at Doctor Howe's private laboratory.

The committee consisted of F. B. Foley, H. M. Boylston, H. C. Boynton, G. K. Burgess, H. J. French, J. H. Hall, Zay Jeffries, R. S. Johnston, F. C. Langenberg, H. F. Moore, E. H. Peirce, H. S. Rawdon, W. E. Ruder and H. J. Stagg.

The work was undertaken at the suggestion of Doctor Howe, who thought, at the outset, that steel in the sorbitic state might have properties which, for most engineering purposes, would be better than those of the usual or pearlitic state. By a systematic variation in the heating temperature (T max.), the time of holding at T max., and the rate of cooling it might be shown what method would prove most economical whereby to produce any given set of physical properties. It was Doctor Howe's work in collaboration with Arthur G. Levy, the results of which have been set forth in their paper,[1] that suggested to him the plan of this investigation. In that paper the rates of cooling were varied for a steel of eutectoid composition using heating temperatures of 800° and 900° C. Whereas, in the previous investigation the rates of cooling were usually interrupted at 650° C., in the present work most of the data have been obtained by uninterrupted coolings.

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* Consulting Metallurgist, U. S. Bureau of Standards. Died May 14, 1922.

† Metallurgist, Bureau of Mines, Mississippi Valley Experiment Station.

‡ Assistant Metallurgist, Bureau of Mines.

The committee is indebted to John A. Roebling's Sons Co. for supplying the steel and to the Carpenter Steel Co. for rolling it into bars. All of the heat-treating operations were performed at Doctor Howe's laboratory (Green Peace), those in the first part being performed by Francis B. Foley and the others by J. Winlock. Microscopic examinations were made there and at the University of Minnesota, by the Bureau of Mines. The tensile properties and hardness were carefully determined by Tom W. Greene, under the direction of R. S. Johnson of the Bureau of Standards. Impact test specimens were broken at the Watertown Arsenal by Dr. F. C. Langenberg. The machining of the test specimens was done in part at the machine shop of the Bureau of Mines at Pittsburgh, Pa. and in part at the works of the Bethlehem Steel Co., South Bethlehem, Pa., under the direction of R. M. Bird and G. C. Lilly, at the works of the American Tool and Mach. Co., Boston, under the direction of H. W. Woodworth; and at the Neverskip Works, New Brunswick, N. J., under the direction of George McCormick.

While the committee had the advantage of the guidance of Doctor Howe during the initiation and progress of the investigation, his death on May 14, 1922, deprived us of his assistance in the compilation and presentation of the work. Credit therefore is given Doctor Howe for the initiation of the program and for his untiring efforts in behalf of the work but he has been in no way responsible for the method of presentation of the results nor for the conclusions and statements made in connection with them, which are primarily those of the junior authors rather than of the committee.

STEELS CONTAINING 0.34, 0.52, AND 0.75 PER CENT. CARBON

The material treated consisted of three lots of acid open-hearth steel (acid open-hearth metal was selected for its greater freedom from impurities as compared with basic steel) of about 1300 lb. each made in

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[1] Notes on Pearlite. *Jnl.* Iron and Steel Inst. (1916), **94**, 210.

11½-in. square ingots and rolled to 4-in. square billets by the John A. Roebling's Sons Co. The billets were rolled to ⅞-in. round bars by the Carpenter Steel Co.

Observations made at the mill, during the final rolling, showed the temperature of the heating furnace to have been 1223° to 1260° C. (2234° to 2300° F.) and that the finishing temperature of the bars was within 14° C. plus or minus of 871° C. (1600° F.).

The Bureau of Standards' analyses of the three lots gave:

	Steel T Per Cent.	Steel U Per Cent.	Steel V Per Cent.
Carbon	0.34 0.34	0.52	0.75
Manganese	0.55	0.56 0.54	0.52 0.54
Phosphorus	0.030	0.031 0.030	0.031 0.031
Sulfur	0.032	0.029	0.027
Silicon	0.14	0.22 0.22	0.11

These analyses are a confirmation of results obtained at the plant. Samples were taken from each rolled bar for the purpose of confirming microscopically the analysis, to guard against mis-stamping; in every instance the ferrite was found to be banded.

Banded Ferrite.—It is not unusual to find banded ferrite in hypoeutectoid steels rolled to bars of this size; in fact some persons contend that this is most frequently the case. Examinations along a plane parallel to the direction of rolling are rarely made because it is much more convenient to prepare a transverse section, which gives no evidence of banding, and thus most cases of banding escape notice. The conditions of temperature and time necessary for the establishment of homogeneity are such as to establish in the steel a condition more unnatural than that which we endeavor to efface, therefore we proceeded with the treatment of the steel as we found it. To what extent this banding affects the actual physical properties resulting from our tests, we cannot say; however, we believe that the fundamental principles evolved are affected only in degree. The rolled bars were sawed into 5½-in. lengths, in which size they were heat treated.

The Furnace

A specially designed vertical cylindrical furnace of the "hump-process" type was manufactured for this investigation by the Leeds & Northrup Co., with special care so as to produce an even temperature throughout the heating chamber, which was 7 in. in diameter and 14 in. deep.

A section through the center line of the furnace is shown in Fig. 1, with the specimens and an extra sheet-iron muffle to aid in the restraint

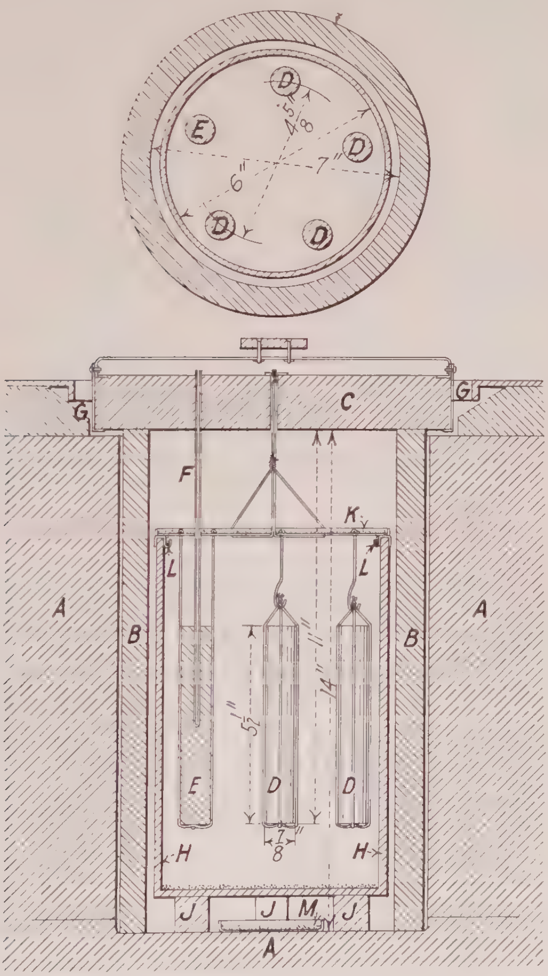

FIG. 1.—FURNACE SHOWING ARRANGEMENT OF SPECIMENS, MUFFLE, ETC., IN HEATING CHAMBER.

of oxidation. *A* is the solid lagging of the sides and bottom of the furnace. *B* is the heating element, which consists of a vertical cylinder of refractory material in which is embedded the resistance wire. *C* is the refractory material cover of the furnace; *D*, the specimens under treatment; *E*, a dummy of the same material as the specimens; *F*, the thermocouple, which has its hot junction at the middle of the length and in the axis of the dummy. *G*, an air space facilitating the entry of the cover *C*, this space was closed with asbestos cord and cloth. *H* is a sheet-iron muffle, which serves to restrain oxidation of the specimens; *J* are silocel

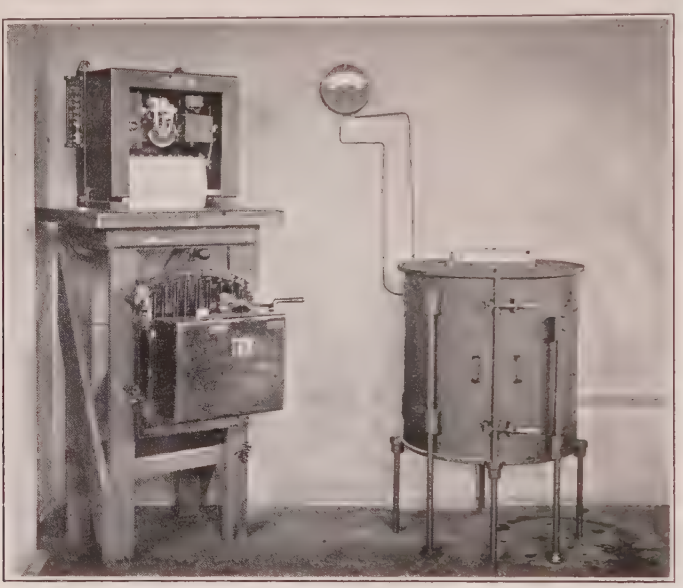

FIG. 2.—FURNACE DURING A HEATING.

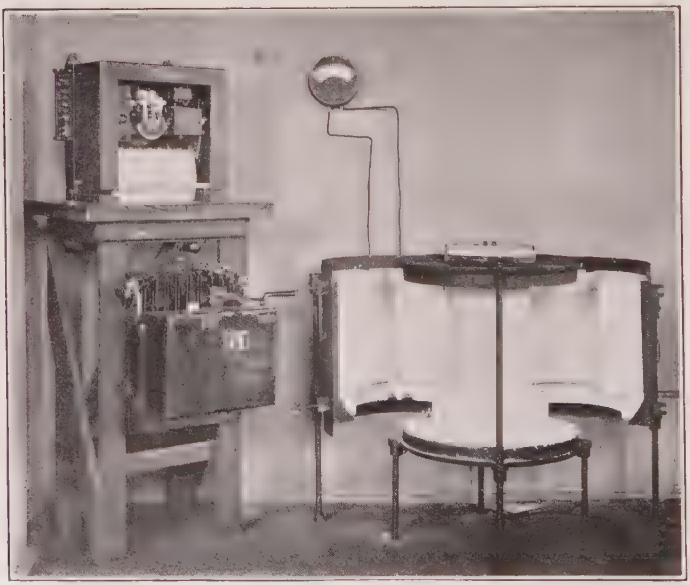

FIG. 3.—FURNACE WITH HEAT INSULATION THROWN BACK TO ACCELERATE FURNACE COOLING.

piers to hold the muffle at the desired height; *K* is the muffle cover (a round pan) in which carbon is spread and from which the specimens hang. *L* is a trough hung around the inside of the top of the muffle and kept full of carbon for the purpose of taking up oxygen about to enter the muffle at the joint. *M* is a small pan of carbon to take up oxygen from the heating chamber. The specimens *D* and the dummy *E* are suspended from the cover of the muffle, which in turn is suspended from the furnace cover.

The furnace was designed with the shell, which held the heat insulation, halved vertically and hinged in such a manner that it was readily thrown back, exposing the heating chamber in order, when desirable, to hasten the cooling. Fig. 2 shows the furnace during a heating and Fig. 3 shows the heat insulation thrown back as described.

The furnace is heated entirely from the sides, the top and bottom being heated only by radiation and convection; hence, the ends tend to be materially colder than the middle. To mitigate this defect, the makers set the heating wires closer together toward the ends than in the middle of the length of the chamber. Also, because this end cooling effect decreases toward the middle of the length, the heating chamber was made much longer than the specimens to be heated, 14 in. against 5½ in. Indeed, for the tensile specimens it is only the middle 2-in. measured part that is of importance, so that here the total length of the heating chamber is seven times that of the part which should be uniformly heated. In order that the measured length of the specimens during the heating may be placed where the temperature is the most even, it is important to determine the thermal gradient of the furnace.

Determining Thermal Gradient

A convenient way of determining the thermal gradient is shown in Fig. 4. The heating chamber is brought to a temperature within a range that includes a large number of the proposed heats and the current is adjusted so that the temperature rises quite slowly. The four specimens are inserted in the regular place, as in Fig. 1, because they tend to equalize the temperature through their thermal conductivity. To determine the thermal gradient, the thermocouple is inserted for 1 in. into a 2-in. length of ⅞-in. round bar and lowered in the line usually occupied by the dummy to within, say, 1 in. of the muffle bottom and held there for 10 min., to allow it to come to temperature, when a record of the time and temperature is made. It is then raised to points 1 in. apart and the time and temperature recorded after a 10-min. stop at each.

This operation is repeated, redetermining the temperature and recording the time at the same series of points as many times as seems desirable. At each of the redetermined points, the temperature is somewhat higher than before. Thus a number of time-temperature points are determined for each inch of the explored line and these are plotted as in Fig. 4, the numbers on the lines denoting the distance from the bottom of the furnace.

From this diagram, the thermal gradient that existed at any given moment is derived, as in Fig. 5, by plotting in a diagram, with the distance from the furnace bottom as the abscissas and the temperature as the ordinates, the intersection of the ordinate of time with the several position lines in Fig. 4.

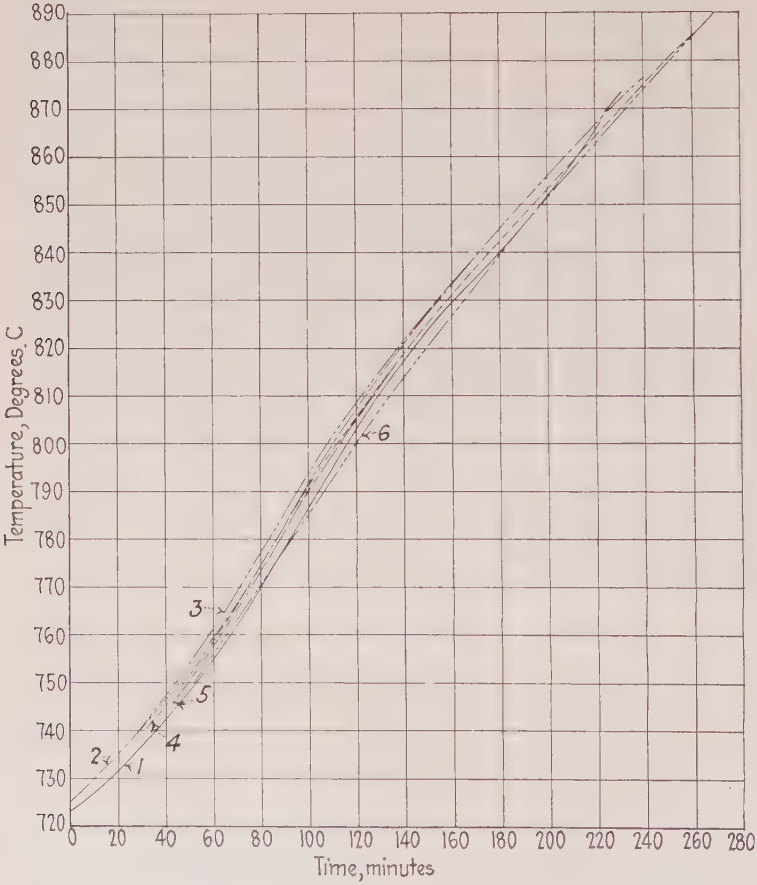

Fig. 4.—Determination of thermal gradient in heating furnace.

The temperature in the open furnace was somewhat more regular than that within the inner sheet-iron muffle in which the test pieces were set. But the muffle restrained the oxidation of the test pieces to a degree which outweighed the loss of regularity of temperature that it caused. Observations made indicated that the maximum difference in temperature at any given instant along the 2-in. measured length of the tensile specimens was about 5½° at 775°, 3½° at 805°, and 2½° at higher temperatures.

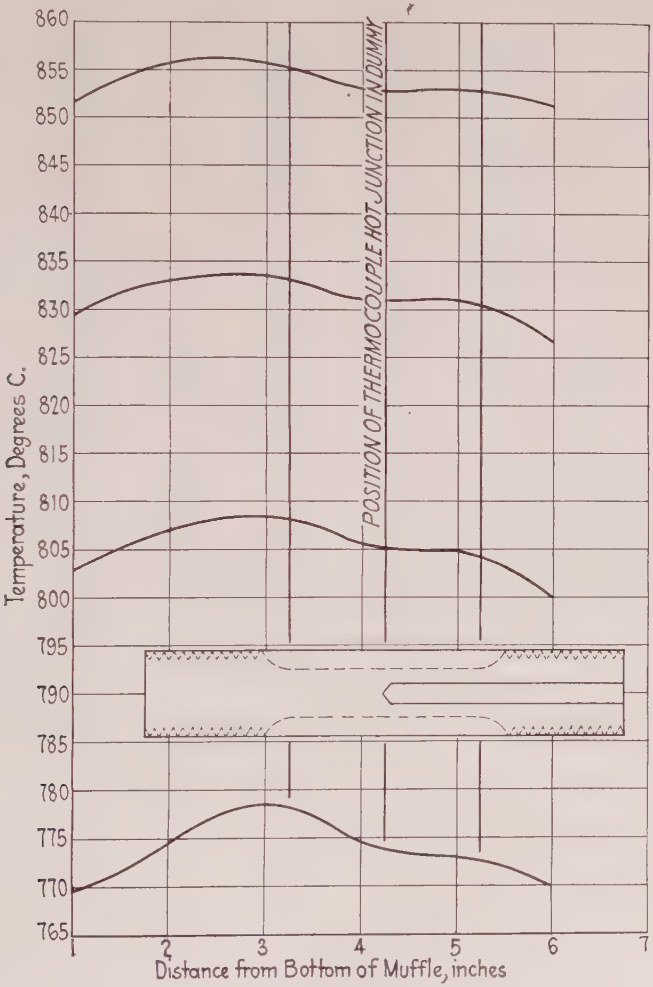

FIG. 5.—THERMAL GRADIENT WITHIN HEATING CHAMBER OF FURNACE.

Temperature Measurements

Temperatures were measured thermoelectrically, using a Hoskin's thermocouple of the usual nichrome-chromel type of No. 20 wire and, in a few instances, a platinum-platinum rhodium couple. Thermocouples were calibrated at the Bureau of Standards and the calibration checked at moderate intervals at Green Peace Laboratory, taking readings with the potentiometer used in making the routine observations during the investigations. In addition, the calibration was checked in every run by noting the temperature of Ac_1, the heatings all closely following an established rate, Fig. 5. The cold junction in every case was in a closed thermos bottle, which contained ice at all times.

Before immersing specimens in water, the water was vigorously stirred and it was experimentally determined that the water continued to circulate for fully a minute thereafter. At first the cooling was done in still water, but the results were not constant; it was only when the water was stirred that consistent results were obtained. The time of immersion in water was determined by counting from a stop watch or from a metronome. The cooling rate was determined from time and temperature observations made during the cooling of the dummy.

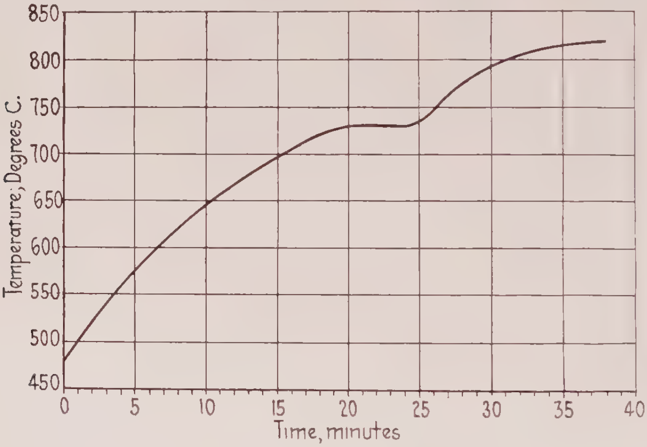

Fig. 6.—Standard rate of heating.

For cooling rates slower than 2° C. per sec. (fan coolings) when using a base-metal couple, the potentiometer was set at even 0.5 mv. in advance of the fall in electromotive force and the time was observed when the galvanometer needle passed zero, indicating a balance. Observations were made in the same manner during heating and a convenient rate of heating was established, Fig. 6, and followed closely at all times by having before the operator a record of the time at which each increment of temperature should be passed. During the passage through the *Ac* and *Ar*, readings were made every 0.1 mv. When using the noble-metal couple, readings of 0.2 mv. were usually made; and during passage through the transformation range, intervals of 0.1 mv. were recorded.

For coolings faster than 2° per sec., readings were taken at convenient intervals sometimes as great as 1.0 mv. apart. Time readings were made from a stop watch and no attempt was made to split seconds.

Experimental Methods.—The general procedure was to raise the furnace to about 600° C. The pan *M*, Fig. 1, containing wood charcoal was then placed on the bottom of the furnace. About this pan, there were set four silocel pedestals *J* to support the muffle. Some wood charcoal was sprinkled on the bottom of the muffle and the trough *L* was filled with it. The muffle was then placed on the silocel supports and in a short time the furnace was again at uniform temperature (about 600° C.). The lid of the furnace was then replaced by one carrying the specimens, as shown in Fig. 1. The furnace rheostat was set to pass a current of 27.5 amp. through the furnace winding and the bars were brought to the temperature desired.

The basal experiments for each composition consisted of a heating at an arbitrary but constant rate (Fig. 6) to 10° C. above the Ac_3, holding at that temperature for 10 min. and cooling at the various rates given.

Cooling Rates

For convenience in handling the data in tabulation the various cooling methods were given the following designations:

A, greatly retarded furnace cooling
B, retarded furnace cooling
C, plain furnace cooling
D, accelerated furnace cooling
E, retarded air cooling
F, plain air cooling
G, air cooling accelerated by fan
H, cooling in molten lead for a period of time and then in still air.
I, cooling in water for a period of time and then in still air.

The letter *H* alone denotes a 4-sec. immersion in lead and the letter *I* alone denotes a 2-sec. immersion in water. The addition of the figures 10 and 20 to the letter *H* denotes 10 sec. and 20 sec. time of immersion in the water and likewise the addition of the figures 3 and 4 to the letter *I* denote 3- and 4-sec. immersion in the molten lead.

All of the furnace coolings took place in the furnace, as shown in Figs. 1 and 2. For the greatly retarded furnace cooling *A*, considerable current was left in, giving an average rate through *Ar* of about 0.01° C. per sec., a smaller amount of current (giving an average rate through *Ar* of about 0.027° C.) was left on for the retarded *B* cooling and all current was shut off for the plain furnace *C* cooling.

In the accelerated furnace cooling *D*, the lagging *A*, Fig. 1, was thrown back as in Fig. 3, leaving the heating element exposed to the air. In the retarded air cooling *E*, the muffle fastened to the lid was drawn along with the specimens and was allowed to cool in the air while suspended from the gallows shown in Fig. 7. In the air cooling *F*, the specimens were suspended as in Fig. 7. In the accelerated air cooling *G* the specimens were suspended in the same way and their cooling hastened by a blast of air from the fan shown in Fig. 7.

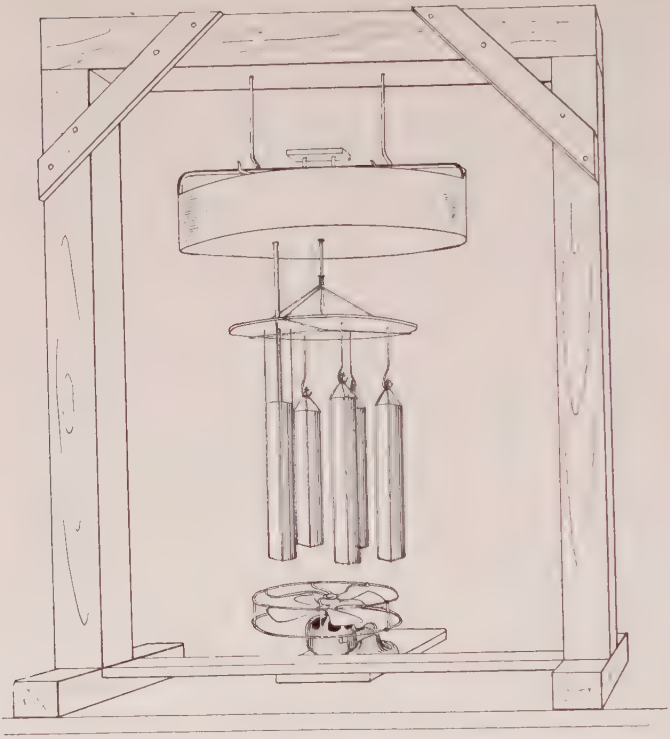

Fig. 7.—Method of suspending specimens during air cooling.

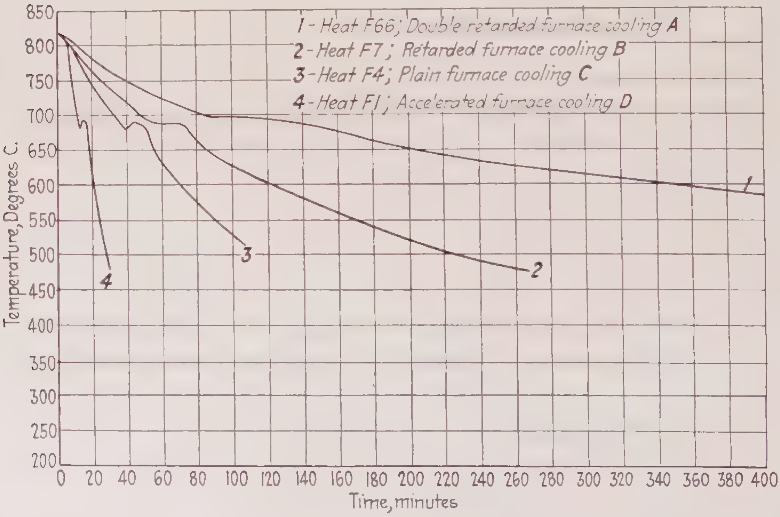

Fig. 8.—Cooling curves, steel T, 0.34 per cent. C.

In the lead *H* and water *I* coolings, the specimens, together with the furnace cover and muffle top, were lifted from the heating furnace and immediately lowered into a tank that had two bars across its top at such a height that, when the furnace cover rested on them, the specimens were immersed to the desired depth. In order to overcome the buoyancy of the lead, a poker set on a sheet-iron frame, fastened to the lower ends of the specimens during their transit from the furnace to the lead pot,

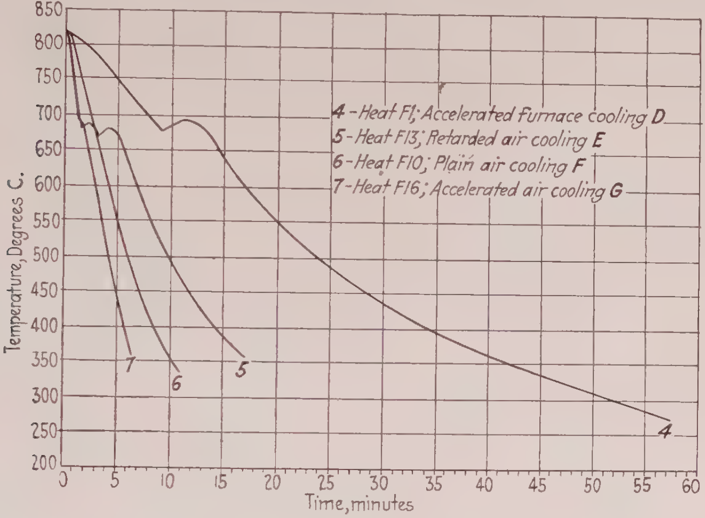

FIG. 9.—COOLING CURVES, STEEL T, 0.34 PER CENT. C.

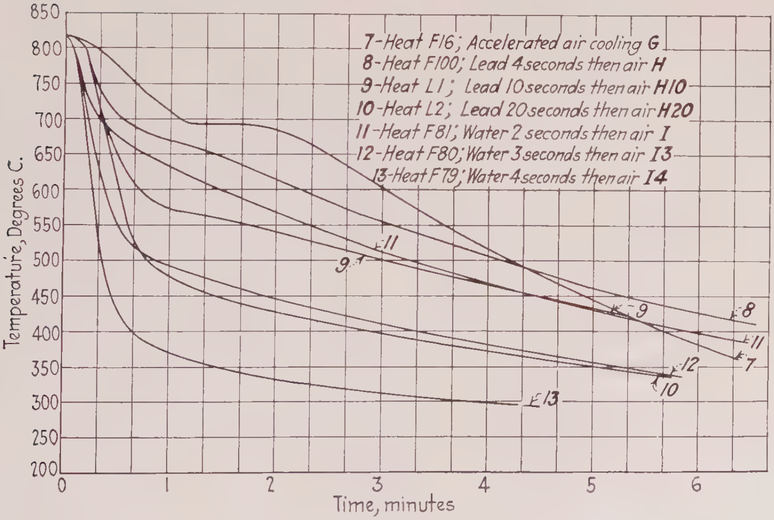

FIG. 10.—COOLING CURVES, STEEL T, 0.34 PER CENT. C.

was used to submerge them. The temperature of the lead bath was recorded by a Leeds & Northrup line-drawing recording potentiometer.

The tank of water was 16½ in. in diameter and 23 in. deep. The lead pot was 8½ in. in diameter and contained about 215 lb. of lead; the depth of the bath was 9 in. The transit from the heating furnace to the cooling bath occupied about 0.6 sec. The observed rates of cooling are shown in Figs. 8 to 10.

Temperatures Used

For the basal experiments the Ac_3 was exceeded by 10° C. The Ac_3 for 0.34 per cent. C steel was taken to be 807° C.; for 0.52 per cent. C steel, 770° C.; and for 0.75 per cent. C steel 741° C. These temperatures were adopted after quenching a number of small specimens of each steel in iced water after a 10-min. hold at temperatures above and below those given. The specimens were drawn at 400° C., polished, and etched with a solution of nitric acid in water, which revealed any ferrite that remained out of solution. The temperatures adopted agree closely with those which result from observations by Howe.[2]

In order to observe the effect of temperatures above and below Ac_3 some of the higher-carbon steels (0.52 and 0.75 per cent. C) were treated together with the 0.34 per cent. steel at 751°, 800°, 817°, and 900° C.

Time at T. Max.—The effect of time was obtained by holding at temperature for periods of 10, 60, 120, and 240 min. For convenience in tabulating, the 1-hr. holding at temperature is indicated in the tables of results by adding the figure 1 to the letters *A* to *G* designating the cooling rates and the 2-hr. holding is indicated by adding the figure 2. A study of the mechanical properties resulting from the various treatments suggest that a 30-min. holding might have fixed the effect of time at temperature better, for in a number of instances, as will be shown later, the properties had not been fully developed by the 10-min. holdings.

Tensile Specimens

The tensile specimens were of the A. S. T. M. standard type, as shown in Fig. 11. In reducing the diameter to 0.505 in., the outer 0.185 in. was removed, which should include all metal that could be decarburized during the heat treatment.

Tensile specimens were pulled in a small Emery hydraulic testing machine of 230,000-lb. capacity at the U. S. Bureau of Standards, and were carefully centered in the machine grips before application of stress. Elongations on the 2-in. gage length were measured by an Ewing extensometer, which was carefully adjusted before being used for this series of tests. The proportional limit was determined from the stress-strain curve, as that stress at which the ratio of the unit stress to unit deformation ceases to be a constant.

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[2] Discussion of the Existing Data as to the Position of Ae_3. *Trans.* (1913) **47**, 611.

The yield point was determined by the drop of beam of the testing machine; as the action of the Emery machine is extremely sensitive the yield point values are probably very accurate. The 0.75 per cent. C steels rarely showed a yield point as indicated by drop of beam. The tensile strength was determined by observing the maximum load carried by the specimen before rupture occurred.

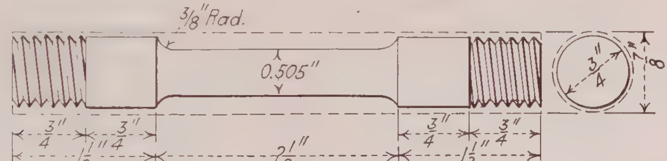

FIG. 11.—STANDARD TENSILE SPECIMEN.

The proportional-limit, yield-point, and tensile-strength values are based on the original cross-sectional area of the specimens.

On account of the sensitiveness of the Emery machine, it was possible to follow the falling off of the load as the specimen necked down and thus to determine the stress at rupture for the 0.34 and 0.52 per cent. C steels. For the 0.75 per cent. C steels, the falling off in load was so slight that usually it could not be followed even with a machine as sensitive as the Emery; therefore, where possible there is given the so-called stress at rupture, which is based on the load at rupture and the reduced cross-sectional area of the specimen, which was measured after rupture took place.

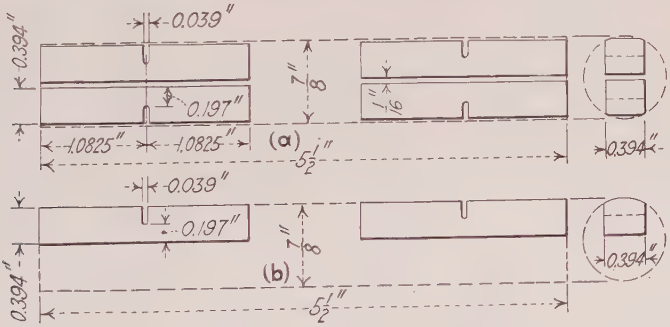

FIG. 12.—(*a*) METHOD OF MAKING FOUR IMPACT SPECIMENS FROM ONE BAR. (*b*) METHOD OF MAKING TWO IMPACT SPECIMENS FROM ONE BAR.

Charpy Impact Specimens.—Those specimens of which four results are given in Tables 1 to 5 were cut as shown in Fig. 12. By using a saw cut 1⁄16 in. wide, four Charpy impact specimens were made from each 5½-in.

bar, leaving a piece about 1.17 in. long at the middle of the bar for microscopic examination. In cutting the test pieces in this manner, their outside corners are trimmed very slightly. This trimmed portion is in the notched side of the specimen and so does not form part of the section tested and has an insignificant, if any, effect on the result.

In those cases where but two impact results are given in Tables 1 to 3, the impact specimens were cut as shown. The relation of these specimens to the remainder of the bar is the same as when four pieces were cut.

Experimental Results

The values of the different properties are shown in Tables 1 to 3. It is apparent, from a study of these values, that a critical rate of cooling exists, faster rates causing a falling off in extension, contraction, and impact resistance, and slower rates causing a drop in all the properties measured. This critical rate seems to be about 2° C. per sec. passing through the *Ar* transformation range. Such a rate has resulted from a 2 sec. quenching in water followed by air cooling.

As far as may be judged from the cooling curves (curve 11, Fig. 10), the *Ar* transformation occurs above 585° C. in steels cooled slower than the *I* rate. The next faster rate *H*10 (curve 9) has produced physical properties which indicate that the critical rate has been exceeded; the contraction of area and impact value have started to fall and the extension shows a marked drop.

With this critical rate as a maximum, a relationship between the cooling rate and the different physical properties has been derived from the basal experiments, which may be expressed by the following empirical formulas:

$$PL = 24{,}750\ (1 + \tfrac{1}{3}c) + [(115{,}225c - 101{,}100c^2 - 12{,}250)\ \sqrt[3]{R}]$$
$$YP = 25{,}000\ (1 + c) + [(89{,}680c - 86{,}185c^2 - 8{,}080)\sqrt[3]{R}]$$
$$TS = 50{,}000\ (1 + c) + [(7{,}023 - 11{,}450c + 50{,}780c^2)\sqrt[3]{R}]$$
$$SR = 118{,}000 + 33{,}200\ \sqrt{R}$$
$$Ext = 52.1 - 61.4\ c + 22.4c^2$$
$$Con = (79 - 61c)\ R^{.05}$$
$$I = (51.25 - 111.5c + 64.27c^2)\ R^{.29}$$
$$B = (97 + 80c) + 100c\ \sqrt[5]{R}$$

In which:

PL = proportional limit;
YP = yield point;
TS = tensile strength;
SR = stress of rupture;
Ext = per cent. extension in 2 in.;
Con = per cent. contraction of area;
I = impact resistance, ft.-lb. per sq. in. (Charpy);
B = Brinell hardness;
c = per cent. carbon;
R = rate of cooling degrees C. per sec. between Ac_3 and 585° C.

Stead gives the following values obtained by Arnold on an iron of 0.08 per cent. C., drastically annealed and very slowly cooled:[3] Yield point, 19,750 lb. per sq. in.; tensile strength, 41,000 lb. per sq. in.; extension in 2 in., 52.7 per cent.; contraction, 76.7 per cent.

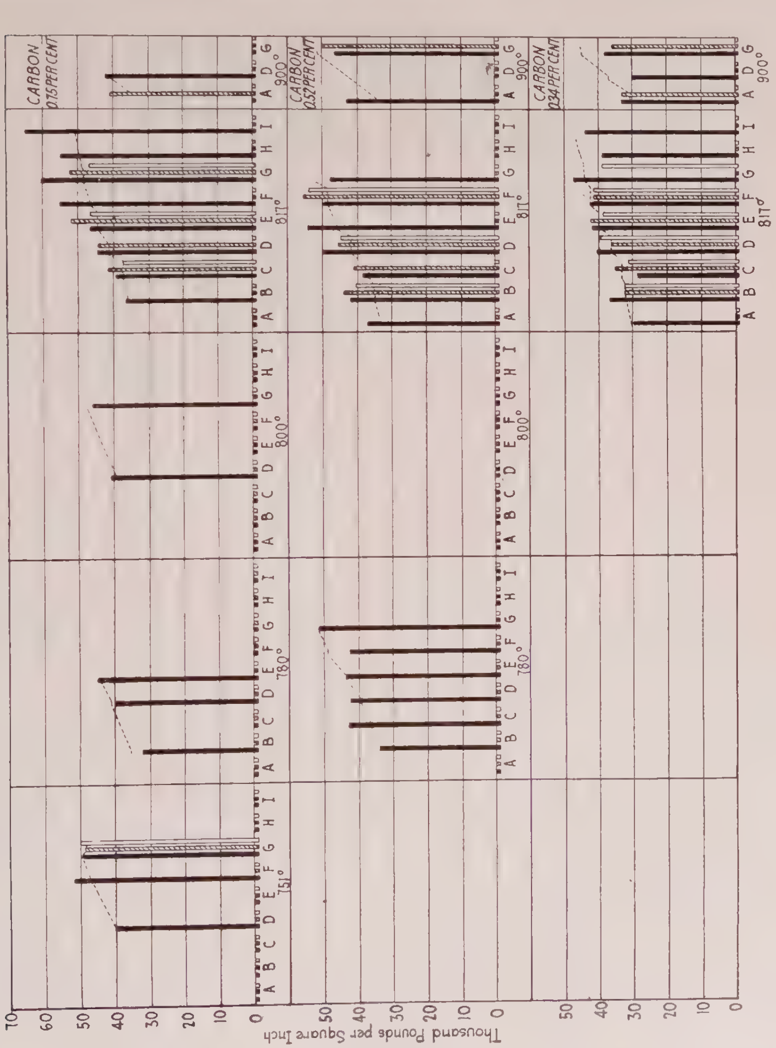

Fig. 13.—Proportional Limit.

The constants in the equations given indicate the following values for an iron containing about 0.22 per cent. Mn and 0.11 per cent. Si with 0.03 per cent. P and S: Yield point. 25,000 lb. per sq. in.; tensile strength,

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[3] Stead: Influence of Some Elements on the Mechanical Properties of Steel. *Jnl.* Iron and Steel Inst. (1916) **94**, 12.

50,000 lb. per sq. in.; extension in 2 in., 52.1 per cent.; contraction, 79 per cent.

Making a correction for the 0.08 per cent. of carbon in Arnold's iron, Stead gives the properties of pure iron as approximately: Yield point, 20,100 lb. per sq. in.; tensile strength, 38,900 lb. per sq. in.; extension in 2 in., 51.6 per cent.; contraction, 83 per cent.

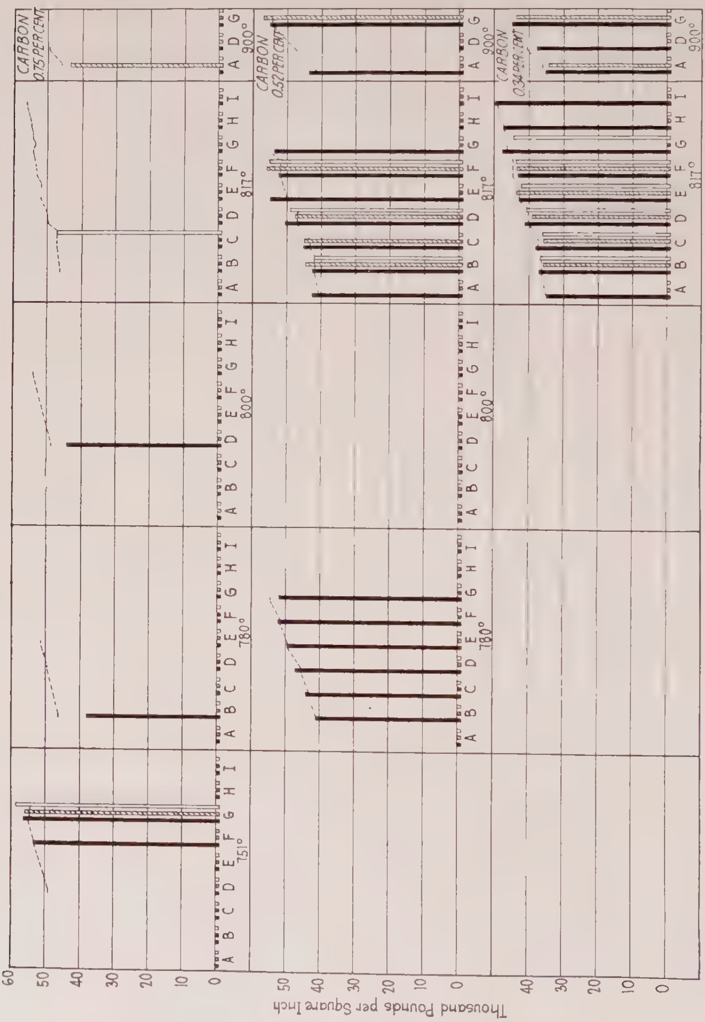

FIG. 14.—YIELD POINT.

In deriving our equations no attempt was made to establish constants that satisfied the foregoing values for pure iron but the agreement appears as close as could be expected, making allowance for the impurities.

Theoretical values have been calculated from these formulas for each of the specimens and are given in Tables 1 to 3. These theoretical values serve as a gage for measuring the effect of temperatures higher than those used in the basal experiment.

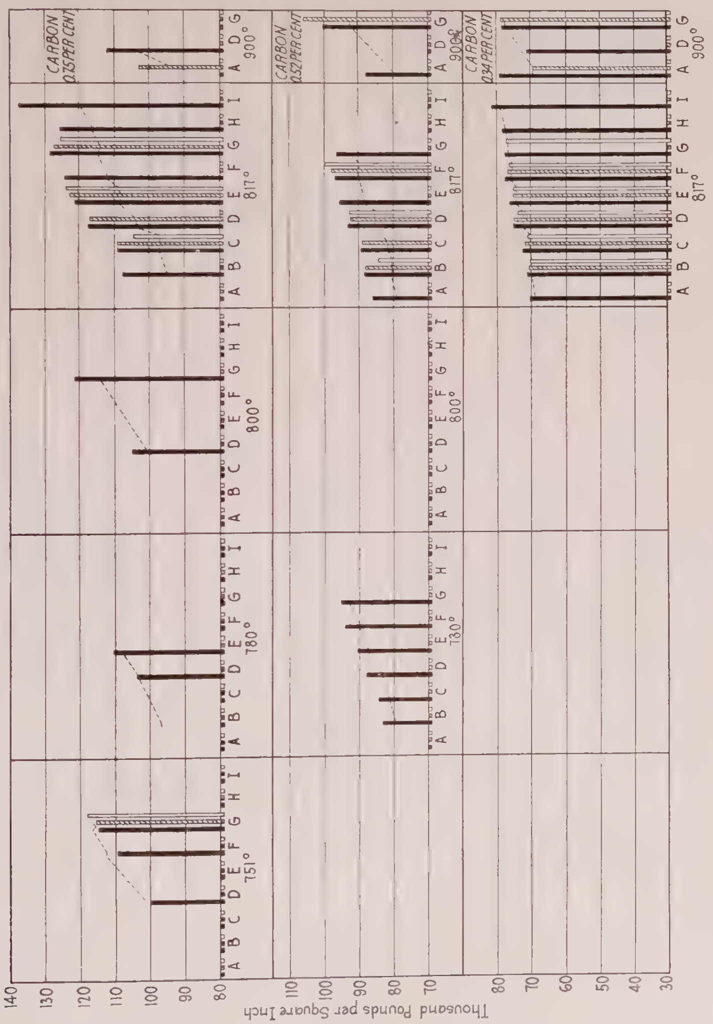

FIG. 15.—TENSILE STRENGTH.

The values are shown graphically in Figs. 13 to 20. The dotted line in each instance represents the theoretical value. The letters at the base of the columns refer to the methods of cooling. The time of holding at T max. is indicated for the 10-min. holding by a solid black column, for the 60-min. holding by an interlined column, and for the 120-min. holding by a blank column.

Discussion of Results

The following analysis is made of the mechanical properties obtained by coolings up to and including the critical rate *I* (2 sec. in water followed by air cooling) all of which rates produced pearlite (Figs. 27 and 28). Rates faster than this produced a sorbitic structure (Figs. 29 and 30) with an attendant increase in proportional limit, yield point, stress at rupture, and hardness, and a decrease in the per cent. of elongation, the per cent. reduction of area, and in impact resistance.

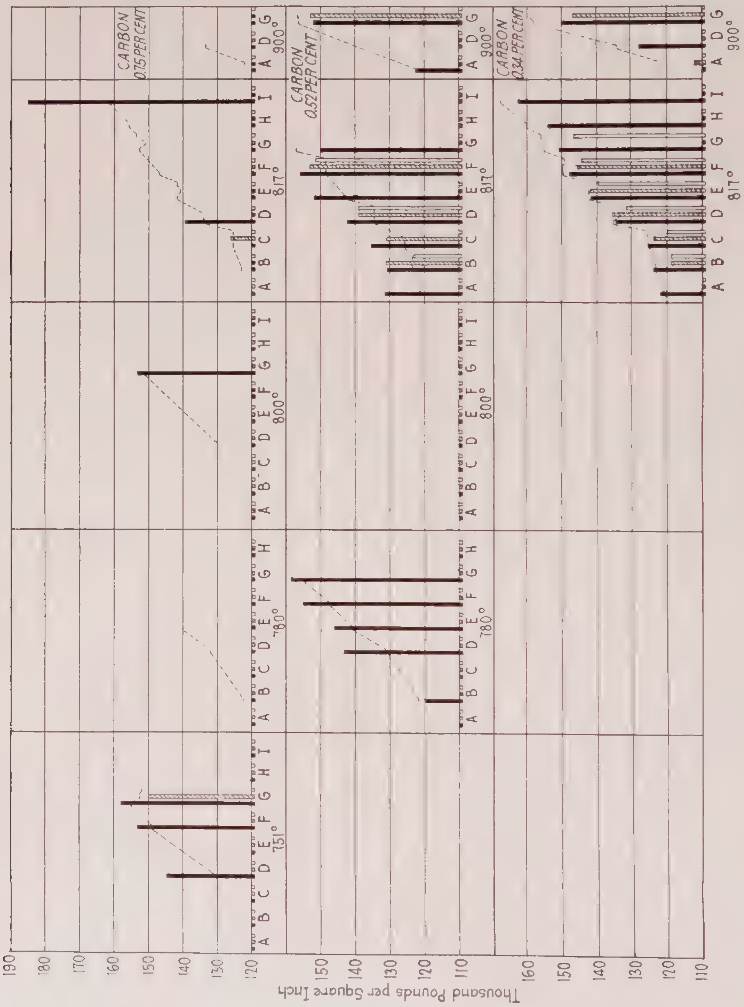

FIG. 16.—STRESS AT RUPTURE.

Proportional Limit.—The proportional limit is increased with increase in rapidity of cooling throughout the range of rates used in these experiments.

The 0.34 per cent. steel was heated to but one temperature above that used in its basal experiment; this indicated a drop in proportional limit with increase in T max. Increasing T max. for 0.52 and 0.75 per cent. C steels caused an increase in proportional limit which is more marked in the more rapid cooling rates.

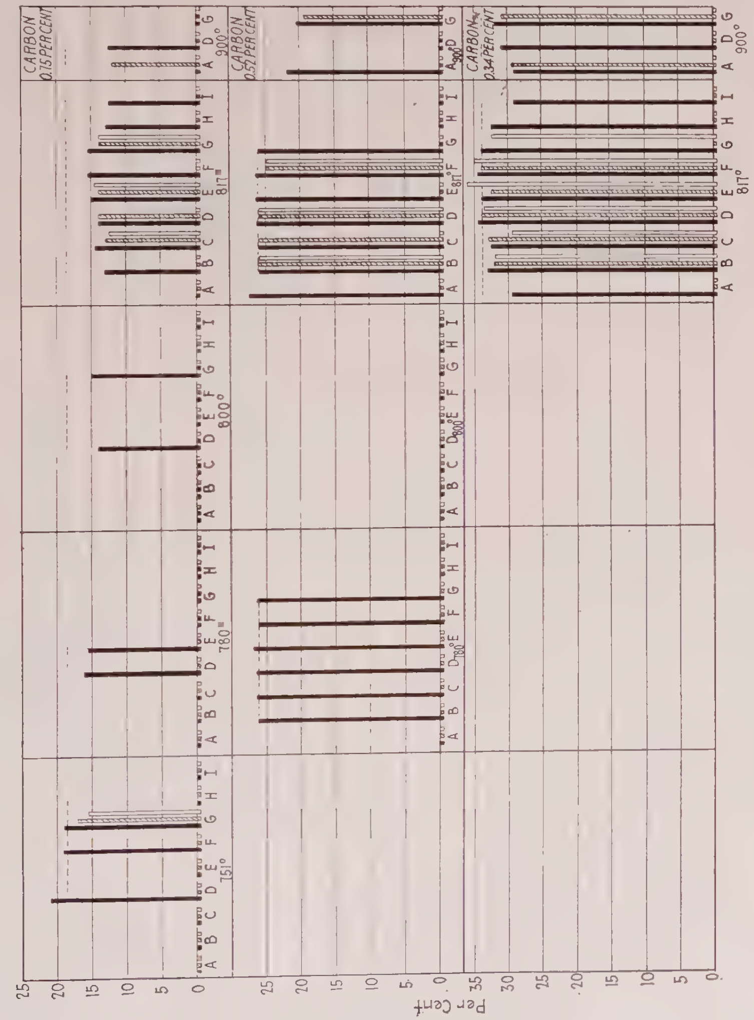

FIG. 17.—EXTENSION.

Undoubtedly there is a drop in proportional limit for steels held at temperature for 2 hr. There are but two instances where the proportional limit of steel held 2 hr. at temperature exceeded that of the steel from the corresponding treatment held at temperature 1 hr. (0.34 per cent. C steel, 817° C., *D* rate, and 0.75 per cent. C steel, 751° C., *C* rate). While there are instances where the 10-min. holding porduced a better proportional limit value than the other holding periods, there are instances where the proportional limit falls short.

Yield Point.—In general, it may be assumed that the effects of temperature, of rate of cooling, and of time at T max. are the same for yield point as for proportional limit.

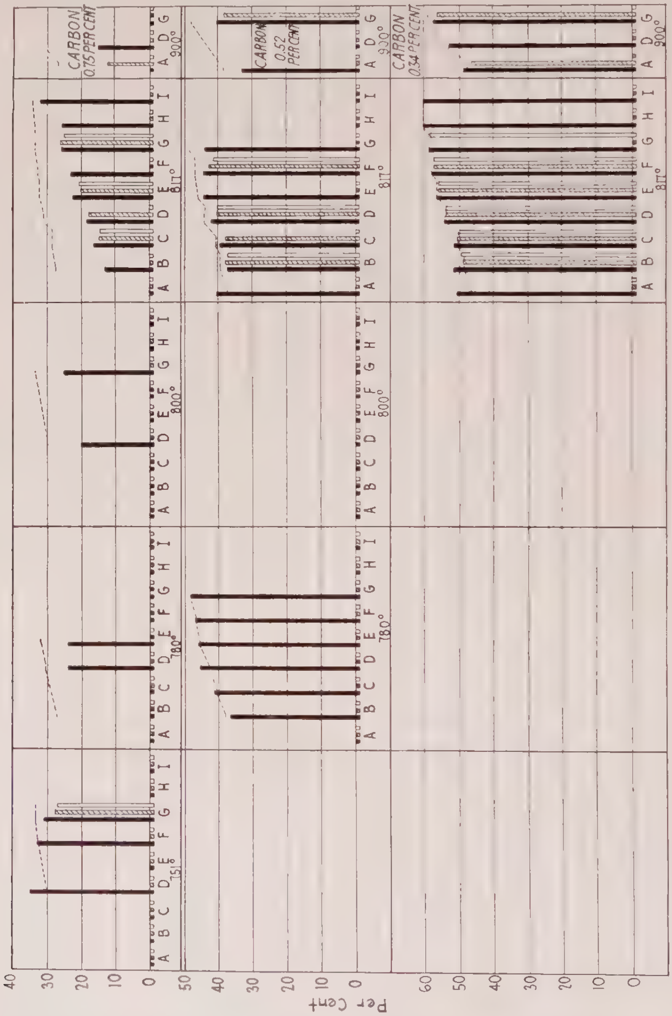

FIG. 18.—CONTRACTION OF AREA.

Tensile Strength.—Tensile strength is increased by increasing the cooling rate.

For steels with 0.75 and 0.52 per cent. C, increasing T max. above the Ac_3 increases the tensile strength but the difference does not become greater with increase in cooling rate, as is evident in the case of the proportional limit (Fig. 13). There is no evidence of change in tensile strength brought about by overheating 0.34 per cent. C steel to 900° C.

Time of holding at T max. uniformly decreased the tensile strength of 0.34 per cent. C steel. The tensile strength of the higher carbon steels was usually lowered by lengthening the period of holding.

Stress at Rupture.—Stress at rupture is increased by increasing the cooling rate of all the steels used.

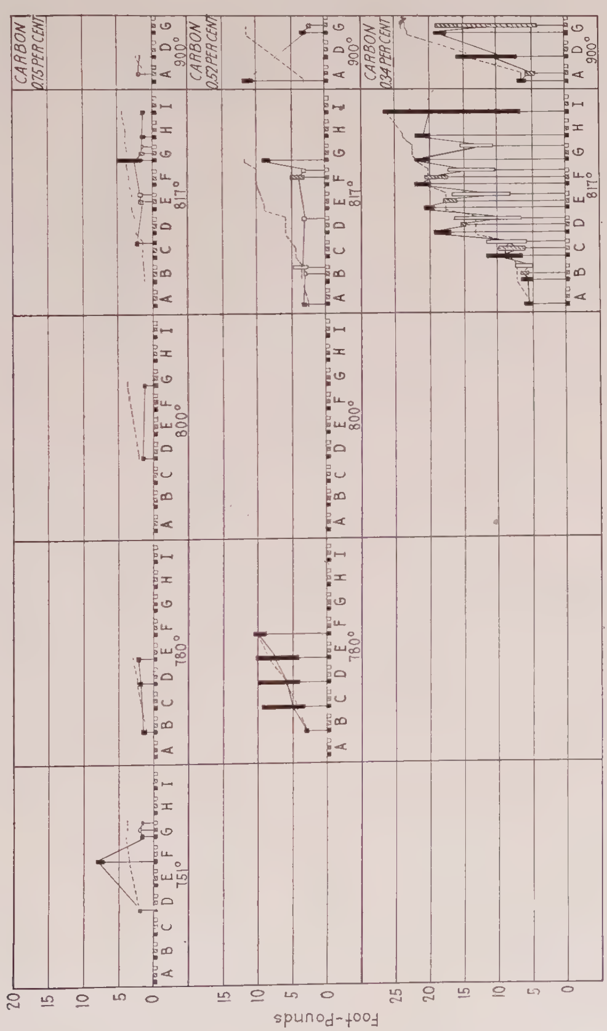

FIG. 19.—IMPACT VALUES.

Increasing T max. probably increased stress at rupture within the range of temperature used for 0.52 and 0.75 per cent. C steels. There was an apparent decrease in stress at rupture at 900° C. for 0.34 per cent. C steel.

Stress at rupture is lowered with increase in time of holding at T max. for all the steels and for all the temperatures used.

Extension.—Variations in the rate of cooling have a minor, if any, effect on the per cent. of extension.

Increasing the temperature above Ac_3 decreases the extension.

The per cent. extension decreases with increase in time of holding at T max.

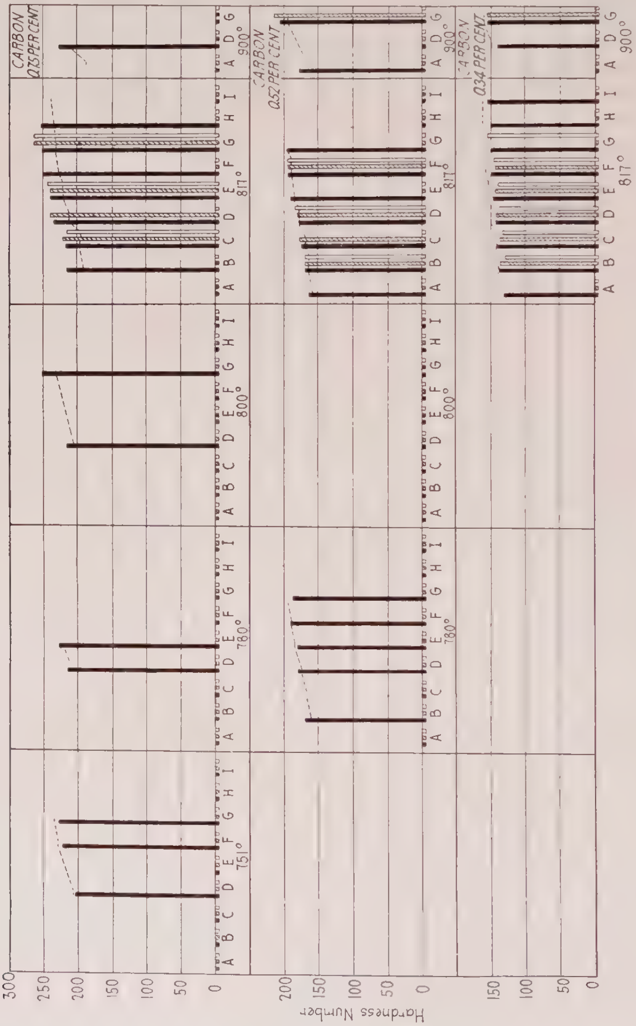

Fig. 20.—Brinell Hardness.

Contraction of Area.—The per cent. of contraction of area is increased with increase in cooling rate. The 0.75 per cent. C steel basal experiment (751° C., 10-min. holding time) does not show this increase in per cent. of contraction in the three specimens tested but it is apparent in all the other series.

Increasing T max. above Ac_3 causes a loss in the per cent. of contraction of area. For the 0.75 per cent. C steel, this loss is decreased by increasing the rate of cooling from 817° C.

Impact Resistance (*Charpy*).—The results from the 0.75 per cent. C steel are not considered enough to serve as a basis from which to draw any conclusions. There is good evidence, however, from the 0.52 and 0.34 per cent. C steels that there is an increase in impact resistance with increase in rate of cooling.

Such evidence as is offered by the results at hand indicate a loss in impact resistance with increase in T max. above Ac_3.

Increasing the time of holding at temperature unquestionably lowers the average impact value; but the effect is an insidious one. In Fig. 19, the length of the blocks indicates the range of the impact values. With the exception of the values for the *I* rate for 0.34 per cent. C steel heated to 817° C. the black columns are shorter than the others. One of the *I* rate bars has a value of 6.75 ft.-lb. and is undoubtedly a sport. Usually the 2-hr. holdings produce bars giving fair to quite poor values.

Hardness

The scleroscope test was found to be less reliable than the Brinell, which is here used as the main index to hardness.

Hardness is increased with increase in cooling rate.

Greater hardness resulted in the 0.75 per cent. C steel by increasing T max. above the Ac_3. This increase in hardness became less apparent in the 0.52 per cent. C steel, and there was no alteration in the hardness of 0.34 per cent C by increasing temperature to 900° C.

Lengthening the time of holding at temperature decreases the hardness of 0.34 per cent. C steel. Time at temperature has no definite hardening nor softening effect on 0.52 and 0.75 per cent. C steels.

Unaffected Results.—The foregoing analysis of the results reveals some unusual findings. It is probably the prevailing opinion among those engaged in the heat treatment of steel that increasing the rate of cooling of steel increases hardness at the expense of ductility. This investigation reveals a general increase, with increase in cooling rate, in all the mechanical properties, with the possible exception of per cent. extension, which property is not impaired.

Another phenomenon strongly suggested by these results is that whereas increasing T max. has a hardening effect on the higher carbon steels, increasing the time of holding at T max. causes a decrease in all properties. By hardness is meant resistance to permanent deformation as shown by tensile strength, elastic limit, and the Brinell test. The low-carbon steel shows a decrease in all physical properties with increase in both temperature and time of holding at temperature. As it is logical to infer that time and temperature have a like effect, it appears extraordinary, at first glance, to find opposite effects in the higher carbon steels.

The most consistent influence of time and temperature is observed in the measurements of the per cent. extension and contraction of area, and in the impact values, properties associated with ductility. These values invariably decrease with increase in both temperature and time. The cause of these apparent anomalies may be found in a consideration of the mechanism of diffusion and of coalescence.

Diffusion and Grain Growth.—There are two factors that must be associated with both time and temperature—diffusion and grain growth. Diffusion has a beneficial effect on the physical properties of metals and grain growth is detrimental.

The rate of diffusion is dependent on what may be termed the diffuso-motive-force, which is proportional to the difference in concentration of the various portions of the mass. It involves migration. It is facilitated by temperature and takes time.

Grain growth is probably to be associated with change in orientation of the crystal elements that go to make up the grain. It does not involve migration of the particles, merely a swinging about *in situ* of the crystal elements. As grain boundaries are the border where masses of crystal elements of different orientation meet, the coalescence of two crystals to make one involves the rotation of the crystal elements of either or both to a position common to both. Like diffusion, coalescence is facilitated by temperature and takes time.

In steel, diffusion involves the dissemination of the impurities, notably of the carbon. The higher the carbon content, therefore, the more pronounced will be the effect of diffusion. Grain growth is equally effective in all steel.

Austenite.—When carbon steel is heated through its critical range, it has recrystallized and become a solid solution of carbon in gamma iron, called austenite. Once the transformation is completed, further heating causes grain growth through coalescence of the austenitic grains and diffusion of the carbon. If such metal were cooled rapidly enough to preserve the austenite, its physical properties would depend solely on its grain size and degree of homogeneity. In this investigation, such a rapid rate was not used; the steel was cooled at such rates that the austenite was completely transformed.

When the austenite, in cooling by the methods here used, has reached a temperature where it is supersaturated with iron, such excess iron goes out of solution as crystals of α iron. Such crystals either migrate or are rejected to the grain boundaries of the austenite so that the grain remains austenitic, but surrounds itself with a shell of α iron, which becomes thicker as the temperature falls. When the austenite grain has become impoverished of iron so that it is of eutectoid composition, the carbon is liberated from solution, forming carbide of iron, and no further rejection of iron to the grain boundaries of the austenite takes place. The previously transformed α iron, therefore, occupies what were the grain boundaries of the pre-existing austenite.

In heating steel, following the transformation to γ iron, a process of grain growth and diffusion ensued; therefore, we may reasonably suppose that following the formation of α iron from the γ iron and the formation of carbide, there is a similar process of grain growth by coalescence. However; if there is coalescence of the ferrite and of the carbide particles we can hardly conceive of a diffusion of the carbide also. The carbide would, through coalescence, tend to become massed and coarse. We are now dealing with a mechanical mixture, not with a solid solution such as austenite; we can readily, therefore, conceive of grain growth but not of diffusion. This grain growth is not an increase in the size of the so-called pearlite grains, which are commonly found oriented by ferrite in hypoeutectoid steels, it is a growth of the crystal of ferrite in the network about such grains and in the ferrite and carbide composing the pearlite. If time is permitted at the higher temperatures following and during the period of phasial change, coarse ferrite and pearlite result

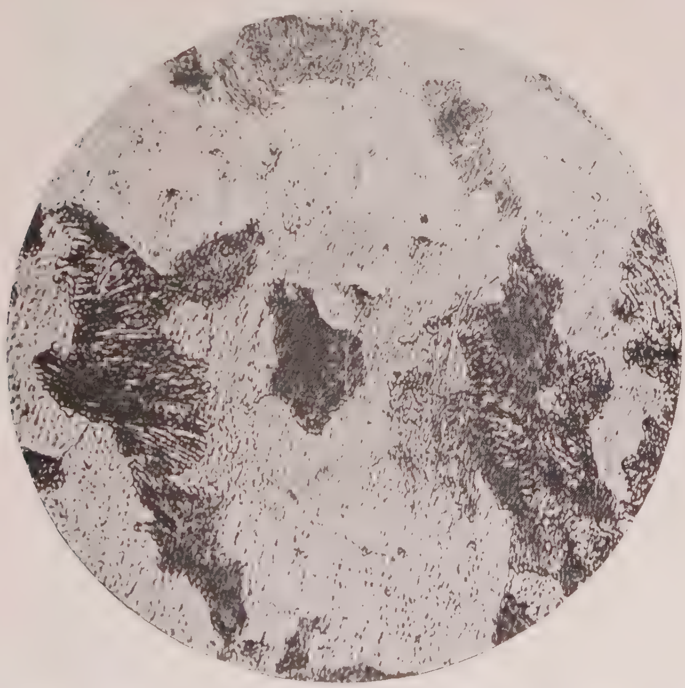

FIG. 21.—STEEL CONTAINING 0.34 PER CENT. C, T MAX. 900° C., HELD 10 MIN., RETARDED FURNACE COOLING, HEAT F 99. ×500.

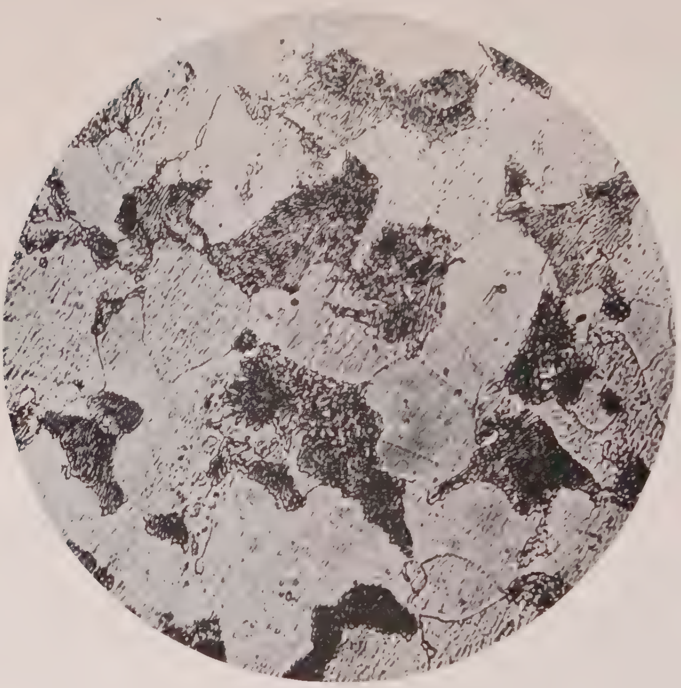

Fig. 22.—Steel containing 0.34 per cent. C, T max. 817° C., held 10 min., retarded furnace cooling, heat F. ×500.

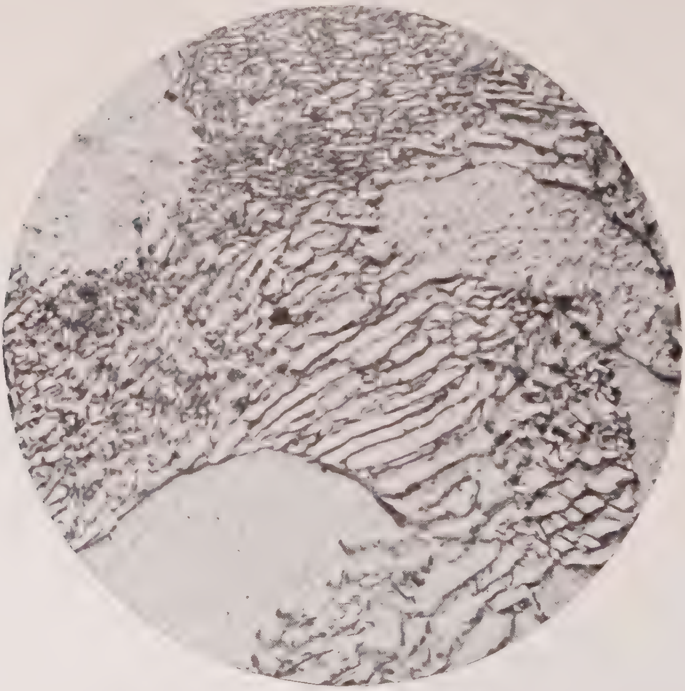

Fig. 23.—Steel containing 0.34 per cent. C, T max. 817° C., held 10 min., retarded furnace cooling, heat F 66. ×2000.

(Figs. 21, 22, and 23). If the time is short, fine ferrite grains and fine pearlite result, Figs. 24 to 28. Rapid coolings lower the temperature at which the *Ar* transformations take place; such lower temperatures retard coalescence and the migration of the α iron to the grain boundaries (Fig. 29). We may conceive of a rate of cooling that would produce ferrite and carbide of, perhaps, colloidal dimensions at such a temperature that neither migration of ferrite nor coalescence of ferrite nor carbide could take place.

In examining annealed steel, one is inclined to regard the ferrite network as an outline of the grain of the metal and declare the steel "fine grained" or "coarse grained." As a matter of fact, a steel showing a fine network of ferrite may be coarse-grained and one showing a large ferrite network may be fine-grained. The large-ferrite network means that the austenite from which it was rejected was coarse-grained. But the substance we are examining is no longer austenite and its grain size does not depend on the austenitic grain size but on the rapidity of its formation and its opportunities for grain growth, in other words on the rapidity of the cooling through the *Ar* transformation zone and the temperature zone immediately below.

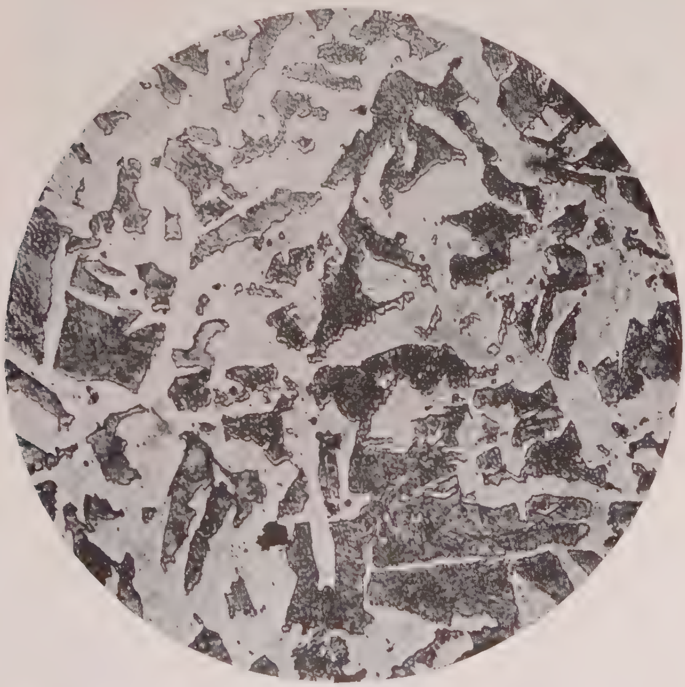

FIG. 24.—STEEL CONTAINING 0.34 PER CENT. C, T MAX. 900° C., HELD 10 MIN., ACCELERATED AIR COOLING, HEAT F 68. ×500.

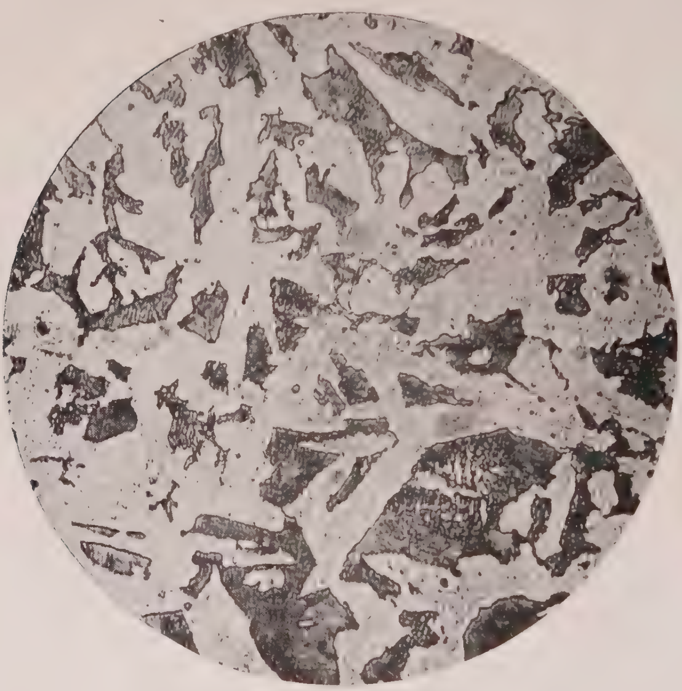

FIG. 25.—STEEL CONTAINING 0.34 PER CENT. C, T MAX. 817° C., HELD 10 MIN., ACCELERATED AIR COOLING, HEAT F 16. ×500.

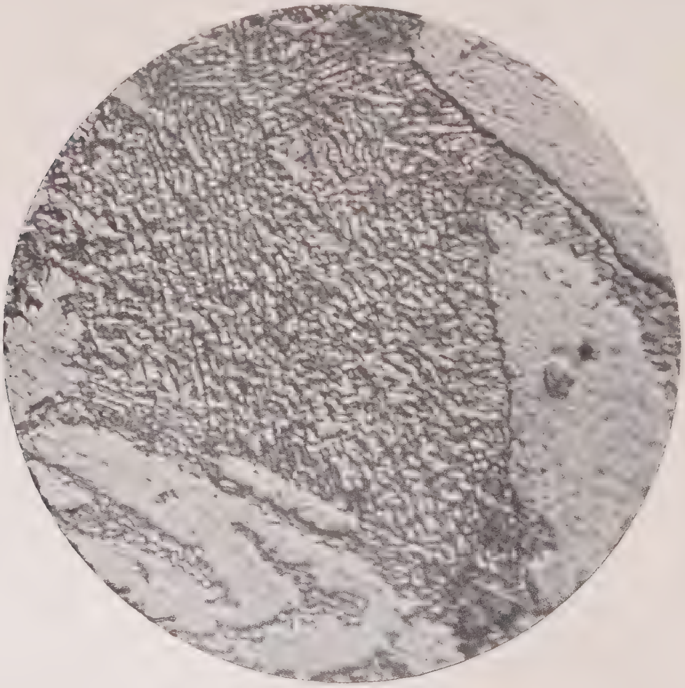

FIG. 26.—STEEL CONTAINING 0.34 PER CENT. C, T MAX. 817° C., HELD 10 MIN., ACCELERATED AIR COOLING, HEAT F 16. ×2000.

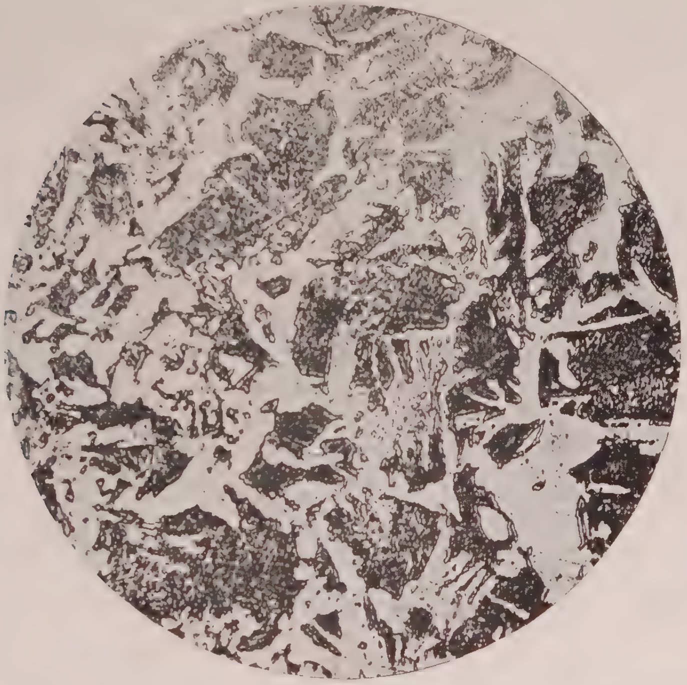

FIG. 27.—STEEL CONTAINING 0.34 PER CENT. C, T MAX. 817° C., HELD 10 MIN., COOLED 2 SEC. IN WATER THEN IN AIR, HEAT F 81. ×500.

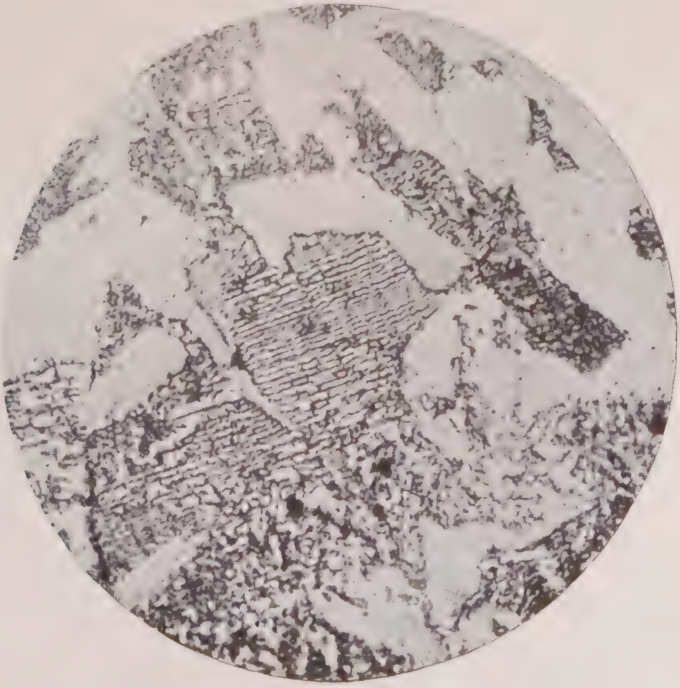

FIG. 28.—STEEL CONTAINING 0.34 PER CENT. C, T MAX. 817° C., HELD 10 MIN., COOLED 2 SEC. IN WATER THEN IN AIR, HEAT F 81. ×2000.

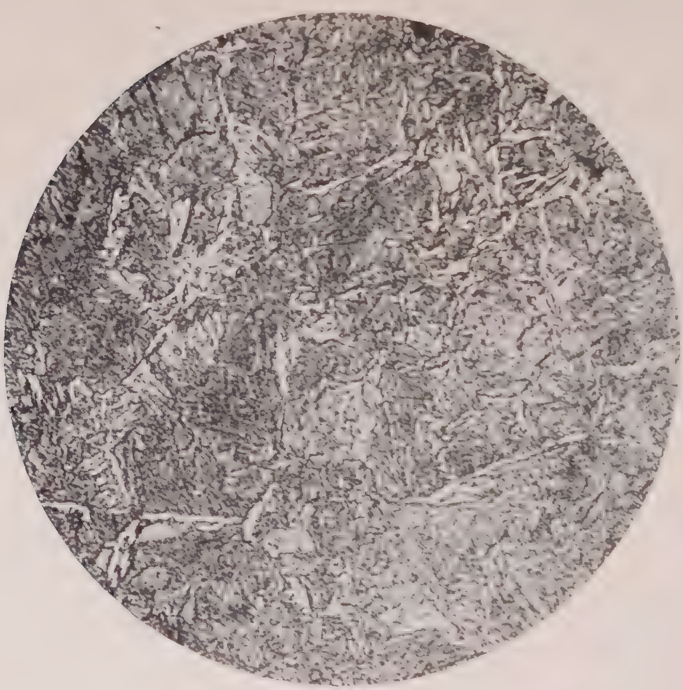

FIG. 29.—STEEL CONTAINING 0.34 PER CENT. C, T MAX. 817° C., HELD 10 MIN., COOLED 3 SEC. IN WATER THEN IN AIR, HEAT F 80. ×500.

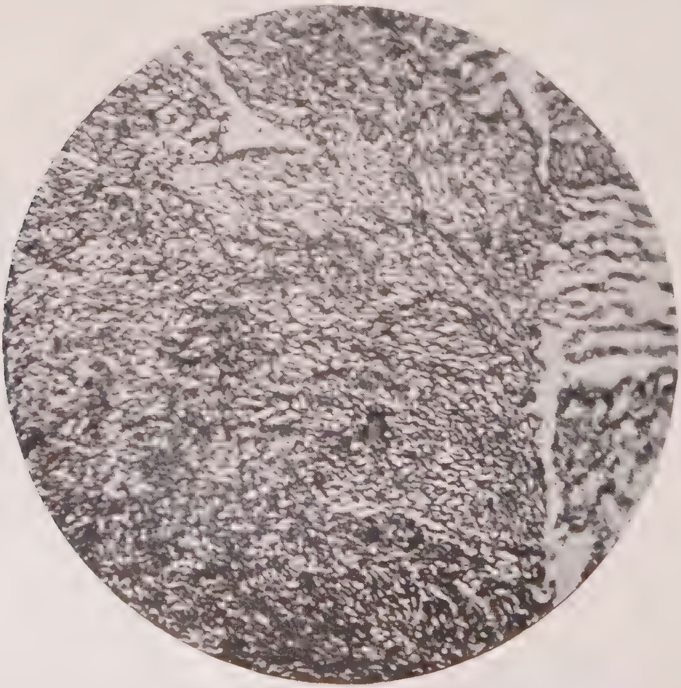

FIG. 30.—STEEL CONTAINING 0.34 PER CENT. C, T MAX. 817° C., HELD 10 MIN., COOLED 3 SEC. IN WATER THEN IN AIR, HEAT F 80. ×2000.

We may have, therefore, as conventionally illustrated in Fig. 31: A fine-grained ferrite-carbide structure in a small network *a*, produced by a fairly rapid cooling of a fine-grained austenite; a coarse-grained ferrite-carbide structure in a small network *b*, which results from a very slow cooling of a fine-grained austenite; a fine-grained ferrite-carbide structure in a large network *c*, which results from a fairly rapid cooling of a large-grained austenite; a coarse-grained ferrite-carbide structure with a large network *d*, which results from a very slow cooling of a large-grained austenite.

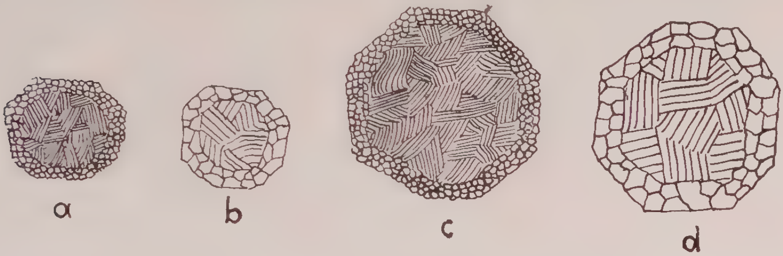

FIG. 31.—NETWORK AND GRAIN SIZE OF FERRITE AND RELATIVE COARSENESS OF PEARLITE LAMELLA.

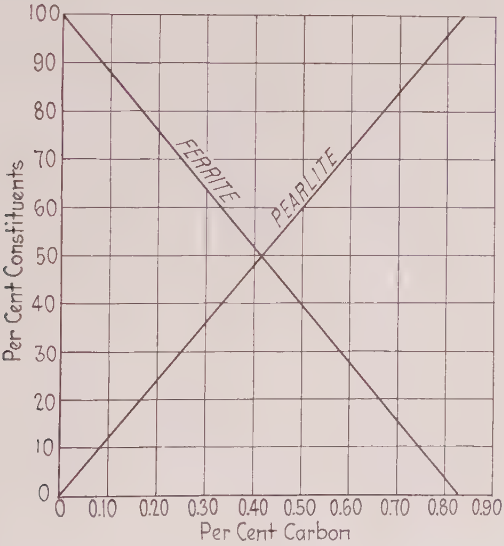

FIG. 32.—STRUCTURAL COMPOSITION OF HYPOEUTECTOID STEEL (AFTER SAUVEUR).

The ferrite network of annealed hypoeutectoid steels must be considered a segregate. If this segregate is made up of very fine crystals it is less harmful than one made up of large crystals. Likewise, the meshes of carbide and ferrite may consist of coarse or fine crystals.

In the basal experiments of this investigation, we have invoked the conditions that produce results ranging from the inferior properties, which we associate with *b*, to the superior properties which we associate with condition *a*. In the overheated steels, we have showed, to some degree, the effect of conditions *c* and *d* on the physical properties.

It is at once apparent, from Fig. 32, derived from tabulations by Sauveur,[4] that, in steels to the left of steels containing 0.415 per cent. C ferrite is the predominant constituent, and in hypoeutectoid steels of higher carbon, pearlite predominates. Our results indicate that the hardness of our steels (0.52 and 0.75 per cent. C), which fall to the right of the intersection of the pearlite and ferrite lines in Fig. 32, is increased on annealing with increase in T max. and the increase becomes more marked the farther we depart to the right of the intersection, being greater in the 0.75 per cent. steel than in the 0.52 per cent. steel. The hardness of the 0.34 per cent. C steel, which falls to the left of the intersection, does not increase. Ferrite being the predominant constituent in the 0.34 per cent. steel, we suppose that improvements in physical properties will be looked for in an improvement in the state of the ferrite but we cannot conceive of any improvement of ferrite resulting from increasing T max. above Ac_3.

As pearlite is the predominant constituent of steels containing 0.52 and 0.75 per cent. C, we expect improvement in the physical properties of these steels to result from improvement in the state of the pearlite.

The results obtained with the higher carbon steels in which there is little proeutectoid ferrite show an increase in the hardness properties with increase in T max. The ductility in the overheated specimens is increased as the condition represented by *c*, Fig. 31, is approached. The evil effects of the segregation of the proeutectoid ferrite are only slightly in evidence, the physical properties being more profoundly affected by the state of the pearlite; and as the pearlite forms, in cooling, from the carbon in the austenite, its distribution and size must be affected by the degree of dissemination of the carbon in the gamma iron. Carbon is probably more finely and uniformly dispersed in austenite at the higher temperatures because the conditions are more propitious for migration and therefore diffusion takes place more rapidly. To this we attribute the increase in hardness of the higher carbon steels subjected to a higher T max. Apparently the beneficial effect of this diffusing is balanced in the low-carbon steel by the harmful influence of the predominant large ferrite network segregate and there is no beneficial influence resulting from increasing T max. above Ac_3.

Diffusion probably reaches its maximum for a given temperature after holding the steel at that temperature between 10 and 30 min.; holding it at a constant temperature for a longer period does not produce an improvement in physical properties. Grain growth, persisting with time, causes a departure from the conditions shown by *a*, Fig. 31, toward that shown by *c*, thus a slight deterioration of properties results from a long period of holding.

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[4] "Metallography and Heat Treatment of Iron and Steel," 135, 2d ed., 1916

There is some indication that the remains of the austenitic grain boundaries may have a detrimental effect on the ductility of pearlite steel, but the greatest influence results from the fact that the austenitic grain boundaries fix the position of the proeutectoid ferrite and thus produce either a coarse or a fine ferrite segregate according as the austenite was fine or large grained.

It will be observed that the contraction of area of steel containing 0.75 per cent C overheated to 817° (Fig. 18) decreases very rapidly as the cooling rate is decreased. The space between the dotted line (calculated values) and the observed values gradually increases toward the slower cooling rates. This may well be attributed to the extra period during which grain growth may proceed in cooling through the temperature range between T max. and Ac_3.

Summary

1. Increasing T max. (above Ac_3) produces austenite, with progressively finer and more uniformly distributed carbon, which, when cooled at such a rate as to produce pearlite, causes an increase in the hardness of high-carbon hypoeutectoid steel. The uniformity of distribution of the carbide in the ferrite-carbide aggregate is a function of T max., and we should expect finer pearlite lamellae to result, with a given cooling rate, from austenite in which the carbon was most uniformly and finely disseminated than from austenite in which the carbon was not uniformly distributed.

2. Increasing T max. (above Ac_3) produces a large-grained austenite, to the grain boundaries of which the proeutectoid ferrite of hypoeutectoid steels is rejected during cooling; and the segregation thus formed is progressively more detrimental to the physical properties of the slowly cooled steel as the carbon content of the steel decreases.

3. Increasing the time of holding at temperatures above Ac_3, beyond a certain undetermined limit (probably between 10 and 30 min. for the steel used) has a detrimental effect on the physical properties of the steel when cooled slowly enough to produce pearlite. This is attributed to a slackening in diffusion and a continuance of grain growth with time at a given temperature.

4. All physical properties of hypoeutectoid steels are increased with increase in cooling rate from above Ac_3, provided the resultant steel is pearlitic. This is attributed to the fact that the rapidly cooled steels are finer grained, regardless of the temperature above Ac_3 from which they cooi

QUENCHED AND DRAWN STEEL CONTAINING 0.52 PER CENT. CARBON

Mainly for the purpose of comparison of physical properties, some of the same 0.52 per cent. C steel used in the preceding experiments was quenched in water and drawn at various temperatures below Ac_3. The equipment and general procedure used in the previous experiments were maintained. The treatments and results are shown in Table 4 and the impact values are plotted in Figs. 33 and 34.

The curves of physical properties, Fig. 33, show that the tensile strength of the steel drawn at lower temperatures, and therefore more closely related to the original austenite, increases with increase in quenching temperature and that the ductility decreases. Undoubtedly there is a more complete suppression of the Ac transformations in water-quenched steels, as T max. increases. As suggested by Portevin and Garvin[5] and by Howe and Levy,[6] the higher temperatures have the effect of destroying crystal starting points and thus increase the lag in the transformation of the solid iron-carbon solution.

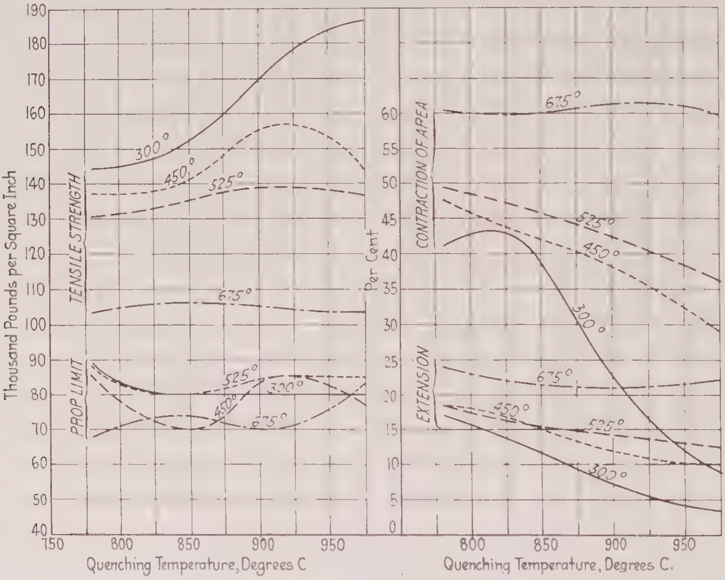

FIG. 33.—PHYSICAL PROPERTIES OF 0.52 PER CENT. C STEEL, WATER QUENCHED AND DRAWN IN ⅞-IN. ROUND BARS.

Microexamination of the specimens drawn at 300° C. disclosed the presence of some ferrite patches in the specimens quenched from 780°, apparently the remains of a segregate that failed to dissolve in the austenite. The predominant constituent was troostite inclined to be rather acicular. The 845° quench showed a little free ferrite and the troostite was more acicular. The 910° and 975° quenches produced martensite and the free ferrite had completely disappeared.

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[5] Investigations on Influence of the Rate of Cooling on Hardening of Carbon Steels. *Jnl.* Iron and Steel Inst. (1919) **99,** 551.

[6] Notes on Pearlite. *Jnl.* Iron and Steel Inst. (1916) **94,** 218.

The specimens drawn at 675° C. showed distinct separation of carbide into very small globules discernible at a magnification of 500 diameters.

The presence of undissolved particles of alpha iron in the specimens heated to the lower temperatures appears to have promoted a greater degree of transformation during the cooling. These undissolved ferrite particles undoubtedly owe their presence to the coarseness of the ferrite segregation in the steel before heating.

Between drawing temperatures of 525° and 675° C., the effect of the austenitic state is apparently lost and a drawing at 675° C. shows practically no influence of the quenching temperature in the ordinary tensile properties. The impact-value versus quenching-temperature curve (Fig. 34) for the 675° C. drawing shows a marked upward trend at the high quenching temperatures.

Evidently the fine, uniform, alpha ferrite-cementite mixture of the highly drawn steel has not suffered from any heritage of grain size from the coarse austenite from which it was born, but has been greatly benefited in ductility, probably by the fineness and greater uniformity of distribution of the constituents.

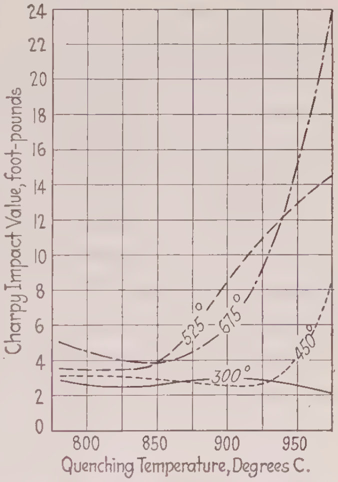


Fig. 34.—Impact resistance of 0.52 per cent. C steel, water quenched and drawn in ⅞-in. round bars.

We have somewhat belittled the importance under certain conditions of the grain size of the austenite from which our ultimate product is derived. The conditions were those under which the austenitic grains were superseded by a very fine pearlite-ferrite grain structure. The greatest influence that the austenite grain size can have under these conditions is in fixing the position of the proeutectoid ferrite in relatively slowly cooled hypoeutectoid steels. When the carbon content is such that little proeutectoid ferrite separates out of the solid solution during annealing, the effect of the size of the austenite grains on the physical properties of the material becomes less important. Uniformity of the material and the grain size of the constituents is much more influential. True, when the austenite is retained by extremely rapid cooling, its grain size is of great importance; but if we completely change the austenite, by drawing, the ghosts of the old boundaries (for apparently the austenite grain boundaries are not completely obliterated and can be revealed by proper etching) have only a minor effect, particularly when the transition has been carried to a state of equilibrium.

The deduction may be made from all of these experiments that ductility and toughness are products of free ferrite, fine crystal structure, and homogeneity.

Repeated Heatings of Steel Containing 0.52 Per Cent. Carbon

The effect of double refinings by repeated heatings was mentioned in a series of heats run under the conditions maintained in the foregoing experiments. The details of temperature, time, and cooling method, with the resultant physical properties, are given in Table 5. The rolled bars were given the two preliminary annealings noted at the top of the table.

Heats *A* and *B* consisted in heating the steel to 780° C. and cooling in air (in heat *A* to 597° C. and in heat *B* to 95° C.) and then reheating to 780° C. and air cooling. Heat *A* has practically duplicated the results obtained by plain air cooling of steel containing 0.52 per cent. C from 780° C. in the investigation first described. The impact value and contraction of area from heat *B* are slightly lower.

In heat *N*, a more rapid cooling, in lead, replaces the air cooling in *A* and *B* and after reheating to 795° C. and air cooling the results show an increase in ductility and a lowering of the tensile strength, the proportional limit and yield point being only slightly impaired. The rapid cooling following the first heating has been beneficial, probably because it has produced a fine carbide which more readily dissolved in the gamma iron.

Heats *I* and *J* are similar to the quenched and drawn heats of the steel containing 0.52 per cent. C. The differences are in the preliminary refinement, by quenching in water from 780° C., and in the water cooling of heat *I* from the 525° C. drawing. The preliminary refinement has worked an improvement, exactly as it did in heat *N*. The quench and draw experiments of the 0.52 per cent. carbon steels indicate, for a 795° quench and 525° draw, an extension of 17.5 per cent., a contraction of area of about 49 per cent., proportional limit of 83,000 lb., tensile strength of 131,000 lb., and an impact value of about 3.5 ft.-lb., from which it will be observed that the ductility of the preliminary refined steel has been materially increased at the expense of a small amount of hardness.

In heats *P*, *Q*, *M*, and *O*, the first refining quench is followed by a quench from 750° C. within the transformation range and a drawing at 525° C. Heat *Q* appears to be a sport. The second heating in these four runs is not a complete refining, which fact is reflected in the results.

The foregoing experiments showed that the impact value was higher in steels cooled slowly than in those quenched in water from the draw, but the difference is not of the magnitude of that found by others in favor of water quenching some alloy steels from the drawing temperature.

General Conclusions

It seems obvious that a fine-grain ferrite binder is the chief requisite for toughness. Apparently pearlite, however fine, is not relatively tough; otherwise we should have an increase in ductility and impact value with increase in carbon content.

Hardness is dependent on the amount, fineness, and uniformity of dissemination of the carbide in the ferrite-carbide aggregate; and apparently the hardest state in which any constituent of a given steel is produced is that in which the constituent exists in its smallest grain size. The "slip interference" hardness theory of Jeffries and Archer is supported by the results throughout this work.

If the transformations of austenite are permitted to become just complete, either by cooling through the transformation zone at a proper rate or by reheating steel in which the transformations have been suppressed by quenching austenite, steel giving the highest resistance to impact is obtained.

TABLE 1.—*Physical Properties of Steels* (*Composition 0.34 Per Cent. C, 0.55 Per Cent. Mn,*

	Treatment			Tensile											
				Proportional Limit			Yield Point			Tensile Strength			Stress at Rupture		
Heat No.	Cooling Method	Rate: ° C per Sec. 815°—585°		Pounds per Sq. In.	Difference Observed—Calculated		Pounds per Sq. In.	Difference Observed—Calculated		Pounds per Sq. In.	Difference Observed—Calculated		Pounds per Sq. In.	Difference Observed—Calculated	
					Diff.	Per Cent. of Observed		Diff.	Per Cent. of Observed		Diff.	Per Cent. of Observed		Diff.	Per Cent. of Observed
As rolled				39000			43000			74750					
Annealed, 900° C. (1652° F.), ½ hr., furnace cooled				37000			38000			68500			113000		
Heated to 900° C.															
F 99	*A*	0.0192	Observed	33000			35000			79000					
			Calculated	31905	−1095	−3.3	36100	+1100	+3.1	69600	−9400	−11.9	122598		
F 98	*A*-1	0.0180	Observed	33000			34000			69200			112000		
			Calculated	31800	−1200	−3.6	36000	+2000	+5.9	69500	+300	+0.4	122500	+10500	+9.4
F 65	*D*	0.247	Observed	30000			37350			71200			128000		
			Calculated	37100	+7100	+23.6	40300	+2950	+7.9	72600	+1400	+2.0	134500	+6500	+5.1
F 68	*G*	1.4838	Observed	38500			44500			78250			150000		
			Calculated	44900	+6400	+16.6	46700	+2200	+4.9	77300	−950	−1.2	158400	+8400	+5.6
F 67	*G*	?	Observed	40000			45000			79000			148000		
F 97	*G*-1	1.55	Observed	36000			43500			78900			147000		
			Calculated	45100	+9100	+25.3	46900	+3400	+7.8	77400	−1500	−1.9	159400	+12400	+8.4
Heated to 817° C.															
F 66	*A*	0.0095	Observed	29500			35000			70375			121900		
			Calculated	30700	+1200	+4.1	35100	+100	+0.3	68900	−1475	−2.1	121200	−700	−0.6
F 7	*B*	0.0264	Observed	36800			36940			71290			124000		
			Calculated	32000	−4800	−13.0	36200	−740	−2.0	69700	−1590	−2.2	123400	−600	−0.5
F 8	*B*-1	0.0269	Observed	32600			35700			70890			119000		
			Calculated	32100	−500	−1.5	36200	+500	+1.4	69700	−1190	−1.7	123400	+4400	+3.7
F 9	*B*-2	0.0287	Observed	32500			36500			70500			119000		
			Calculated	32200	−300	−0.9	36300	−200	−0.5	69800	−700	−1.0	123600	+4600	+3.9
F 4	*C*	0.0535	Observed	28500			37940			72520			125500		
			Calculated	33200	+4700	+16.5	37200	−740	−2.0	70400	−2120	−2.9	125700	+200	+0.2
F 5	*C*-1	0.0534	Observed	35400			35820			71890			124000		
			Calculated	33200	−2200	−6.2	37200	+2620	+7.3	70400	−1490	−2.1	125700	+1700	+1.4
F 6	*C*-2	0.0514	Observed	31500			36000			71000			120000		
			Calculated	33200	+1700	+5.4	37100	+1100	+3.1	70300	−700	−1.0	125500	+5500	+4.6
F 1	*D*	0.2190	Observed	40500			40940			75260			134800		
			Calculated	36700	−3800	−9.4	40000	−940	−2.3	72430	−2830	−3.8	135500	+700	+0.5
F 2	*D*-1	0.2188	Observed	36200			38940			75040			136000		
			Calculated	36700	+500	+1.4	40050	+1110	+2.9	72430	−2610	−3.5	133500	−2500	−1.8
F 3	*D*-2	0.2141	Observed	39500			40500			74000			131900		
			Calculated	36600	−2900	−7.3	39950	−550	−1.4	72380	−1620	−2.2	133400	+1500	+1.1
F 13	*E*	0.5707	Observed	42000			42810			76280			142000		
			Calculated	40100	−1900	−4.5	42830	+20	+0.0	74470	−1810	−2.4	143080	+1080	+0.8
F 14	*E*-1	0.5542	Observed	42500			43560			75140			142000		
			Calculated	40000	−2500	−5.9	42730	−830	−1.9	74390	−750	−1.0	142700	+700	+0.5
F 15	*E*-2	0.5623	Observed	38500			42000			75150			139700		
			Calculated	40100	+1600	+4.2	42780	+780	+1.9	74430	−720	−1.0	142900	+3200	+2.3
F 10	*F*	0.9278	Observed	42000			43030			77510			148000		
			Calculated	42400	+400	+1.0	44640	+1610	+3.7	75780	−1730	−2.2	150000	+2000	+1.4
F 11	*F*-1	0.9350	Observed	41400			43180			76880			146000		
			Calculated	42400	+1000	+2.4	44670	+1490	+3.5	75800	−1080	−1.4	150100	+4100	+2.8
F 12	*F*-2	0.9091	Observed	41500			43500			76500			144700		
			Calculated	42250	+750	+1.8	44560	+1060	+2.4	75720	−780	−1.0	149660	+4960	+3.4
F 16	*G*	1.2637	Observed	47430			47430			77760			151000		
			Calculated	43970	−3460	−7.3	45960	−1470	−3.1	76730	−1030	−1.3	155300	+4300	+2.8
F21W	*G*-2	1.3295	Observed	39000			44500			77400			147000		
			Calculated	44260	+5260	+13.5	46200	+1700	+3.8	76900	−500	−0.6	156280	+9280	+6.3
F100	*H*	1.6312	Observed	39000			47200			78200			154000		
			Calculated	45440	+6440	+16.5	47150	−50	−0.1	77590	−610	−0.8	160390	+6390	+4.1
L 1	*H*-10		Observed	49500			58400			92200			174500		
L 2	*H*-20		Observed	53600			56800			91200			170000		
F 81	*I*	2.2782	Observed	44000			49500			81600			163000		
			Calculated	47550	+3550	+8.1	48880	−620	−1.3	78840	−2760	−3.4	168100	+5100	+3.1
F 80	*I*-3		Observed	52000			58750			91250			178000		
F 79	*I*-4		Observed	65000						108800			193000		
Heated to 800° C.															
F101	*D*		Observed	36000			37250			70400			125500		
F 17	*G*		Observed	36000			43500			77250			146000		
Heated to 780° C.															
F 22	*C*		Observed	28500			36500			69850			122000		
F 28	*F*		Observed	33500			37600			72400			130500		
F 34	*G*		Observed	34000			40000			76870			141000		
F 72	*I*		Observed	52000						99900			160000		
Heated to 751° C.															
F 41	*D*		Observed	32800			33950			71500			126000		

Containing 0.34 *Per Cent. Carbon*
0.030 *Per Cent. P*, 0.032 *Per Cent. S*, 0.14 *Per Cent. Si*)

Properties						Impact (Charpy)							Hardness			
Extension			Contraction of Area			Resilience, Foot-pounds							Brinell			
	Difference Observed—Calculated			Difference Observed—Calculated		Foot-pounds					Difference Observed—Calculated			Difference Observed—Calculated		
Per Cent.	Diff.	Per Cent. of Observed	Per Cent.	Diff.	Per Cent. of Observed	Min.			Max.	Average	Diff.	Per Cent. of Observed	No.	Diff.	Per Cent. of Observed	Scleroscope
30.0			55.6			15.21	15.47	15.60	17.28	15.89						
29.0			46.8			5.16	5.48	5.48	6.00	5.53						
(1652° F.) = Ac_3 + 93° C.																
29.0			48.5			6.00			7.19	6.60						
33.8	+4.8	+16.6	47.6	−0.9	−1.9					6.60	0.0	0.0	139			
29.5			46.5			4.55	5.69	5.90	5.90	5.51						
33.8	+4.3	+14.6	47.5	+1.0	+2.1					6.48	+ 0.97	+ 17.6	139			
31.0			53.0			7.18	8.07	8.53	16.11	9.97			140			24
33.8	+2.8	+ 9.0	54.1	+1.1	+2.1					13.85	+ 3.88	+ 38.9	150	+10	+7.1	
32.0			57.6			17.67	17.80	18.07	19.41	18.24			156			22
33.8	+1.8	+ 5.6	59.2	+1.6	+2.8					23.28	+ 5.04	+ 27.6	161	+ 5	+3.2	
31.0			56.3			8.87	13.59	17.54	17.67	14.42			157			25
31.0			56.5			4.24	5.90	13.71	19.14	10.75			156			24
33.8	+2.8	+ 9.0	59.2	+2.7	+4.8					23.59	+12.84	+119.4	161	+ 5	+3.2	
(1503° F.) = Ac_3 + 10° C.																
29.5			50.8			5.06	5.27	5.90	6.11	5.57			131			
33.8	+4.3	+14.6	46.2	−4.6	−9.1					5.38	− 0.19	− 3.4	137	+ 6	+4.6	23
33.0			51.6			5.06	5.58	5.79	6.65	5.77			139			
33.8	+0.8	+ 2.4	48.6	−3.0	−5.8					7.24	+ 1.47	+ 25.4	140	+ 1	+0.7	22
32.0			48.9			5.58	6.11	6.32	6.65	6.17			137			
33.8	+1.8	+ 5.6	48.7	−0.2	−0.4					7.28	+ 1.11	+ 18.0	140	+ 3	+2.2	20
32.0			49.5			5.06	5.06	5.58	7.52	5.80			130			
33.8	+1.8	+ 5.6	48.8	−0.7	−1.4					7.42	+ 1.62	+ 27.9	141	+11	+8.4	22
32.5			51.4			6.65	7.08	11.30	11.78	9.20			142			
33.8	+1.3	+ 4.0	50.4	−1.0	−1.9					8.88	− 0.32	− 3.5	143	+ 1	+0.7	19
33.0			50.8			6.11	6.65	6.65	10.02	7.36			138			
33.8	+0.8	+ 2.4	50.4	−0.4	−0.8					8.90	+ 1.54	+ 20.9	143	+ 5	+3.6	23
29.5			49.8			5.90	6.11	10.48	11.78	8.57			134			
33.8	+4.3	+14.6	50.3	+0.5	+1.0					8.78	+ 0.21	+ 2.5	143	+11	+8.2	22
34.5			54.4			17.02	18.33	18.74	19.41	18.37			143			
33.8	−0.7	− 2.0	54.0	−0.4	−0.7					13.35	− 5.02	− 27.3	149	+ 6	+4.2	22
34.0			54.1			14.84	14.84	14.84	15.47	15.00			143			
33.8	−0.2	− 0.6	54.0	−0.1	−0.2					13.35	− 1.65	− 11.0	149	+ 6	+4.2	23
33.5			53.8			6.54	6.65	8.30	16.50	9.50			138			
33.8	+0.3	+ 0.9	54.0	+0.2	+0.4					13.35	+ 6.19	+ 65.2	149	+11	+8.0	22
34.0			56.5			19.41	20.77	20.77	20.77	20.43			147			
33.8	−0.2	− 0.6	56.7	+0.2	+0.4					17.65	− 2.82	− 13.8	154	+ 7	+4.8	21
32.5			56.0			16.11	16.76	17.41	18.07	17.09			145			
33.8	+1.3	+ 4.0	56.6	+0.6	+1.1					17.54	+ 0.45	+ 2.6	154	+ 9	+6.2	24
36 0			56.0			8.30	8.53	14.97	16.76	12.14			141			
33 8	−2.2	− 6.1	56 6	+0.6	+1.1					17.58	+ 5.44	+ 44.8	154	+13	+9.2	23
34.5			58 0			19.97	21.33	22.16	22.16	21.40			149			
33.8	−0.7	− 2.0	58.1	+0.1	+0.2					20.32	− 1.08	− 5.0	158	+ 9	+6.0	22
34.0			57.6			17.41	17.41	20.77	20.77	19.09			144			
33.8	−0.2	− 0.6	58.1	+0.5	+0.9					20.37	+ 1.28	+ 6.7	158	+14	+9.7	26
35 0			57.6			10.48	17.28	17.41	17.41	15.65			147			
33.8	− 1.2	− 3.4	58.0	+0.4	+0.7					20.20	+ 4.55	+ 29.1	157	+10	+6.8	24
34.0			58.8			20.10	21.47	22.16	22.16	21.47			149			
33.8	−0.2	− 0.6	59.0	+0.2	+0.3					22.22	+ 0.75	+ 3.5	160	+11	+7.4	23
32.5			58.4			10.83			15.60	13.22			155			
33.8	+1.3	+ 4.0	59.1	+0.7	+1.2					22.56	+ 9.34	+ 70.7	160	+ 5	+3.2	24
32.5			60.2			19.97	20.36	21.05	22.02	20.85			150			
33.8	+1.3	+ 4.0	60.5	+0.3	+0.5					23.93	+ 2.08	+ 10.0	162	+12	+8.0	29
23.0			59.4			18.87	19.55	20.10	20.56	19 75			187			25
26.0			58.5			10.95	12.26	13.71	17.41	13.58			185			23
29.0			60.3			6.75	22.44	24.45	26.65	20.07			156			
33.8	+4.8	+14.2	60.7	+0.4	+0.7					26.38	+ 6.31	+ 31.4	163	+ 7	+4.5	22
24.0			61.3			5.27	7.08	9.55	22.16	11.02			255			48
17.5			59.6			4.95	5.06	6.65	9.90	6.64						
(1472° F.) = Ac_3 − 7° C.																
32.0			53.0			7.41			15.72	11.56			138			22
32.0			57.4			8.76	11.90		12.38	11.01			149			23
(1436° F.) = Ac_3 − 27° C.																
34.0			52.2			5.37	5.37	5.69	6.00	5.61			136			21
32.0			55.0			10 13	13.71	17.80	19.55	15.30			146			24
32.5			54.7			6.75	7.41	14.58	14.97	10.93			150			24
16.0			46.8			2.66	3.95	3.95	4.85	3.85			217			36
(1384° F.) = Ac_3 − 56° C.																
33			52.5			5 .90	6.00	6 11	7.63	6 41			143			23

TABLE 2.—*Physical Properties of Steel (Composition 0.52 Per Cent. C, 0.55 Per Cent. Mn,*

Heat No.	Treatment			Tensile											
	Cooling Method	Rate: ° C. per Sec. 770°–585°		Proportional Limit			Yield Point			Tensile Strength			Stress at Rupture		
				Pounds per Sq. In.	Difference Observed—Calculated		Pounds per Sq. In.	Difference Observed—Calculated		Pounds per Sq. In.	Difference Observed—Calculated		Pounds per Sq. In.	Difference Observed—Calculated	
					Diff.	Per Cent. of Observed		Diff.	Per Cent. of Observed		Diff.	Per Cent. of Observed		Diff.	Per Cent. of Observed
As rolled				39000						95250			143000		
Annealed, 900° C. (1652° F.), ½ hr., furnace cooled				41000			44000			86750			123000		
															Heated to 900° C.
*F*99	*A*	0.0169	Observed	43000			43800			88000			122000		
			Calculated	34150	−8850	−20.6	41410	−2390	−5.5	79725	− 8275	− 9.4	122320	+ 320	+0.2
*F*68	*G*	1.321	Observed	46000			55000			100600			152000		
			Calculated	51340	+5340	+11.6	54230	− 770	−1.4	92240	− 8360	− 8.3	156150	+ 4150	+2.7
*F*67	*G*	?	Observed	43500			54400			100900			150000		
*F*97	*G*-1	1.35	Observed	50000			57000			106000			153000		
			Calculated	51500	+1500	+ 3.0	54350	−2650	−4.6	92350	−13650	−12.9	156580	+ 3580	+2 3
															Heated to 817° C.
*F*66	*A*	0.0082	Observed	37000			42500			86000			131300		
			Calculated	33140	−3860	−10.4	40580	−1920	−4.5	78990	− 7010	− 8.1	121000	−10300	−7.9
F 7	*B*	0.0237	Observed	41900			42300			88370			130500		
			Calculated	34880	−7020	−16.8	41880	− 420	−1.0	80250	− 8120	− 9.2	123110	− 7390	−5 7
F 8	*B*-1	0.0242	Observed	44000			44330			88170			131000		
			Calculated	34900	−9100	−20.7	41910	−2420	−5.5	80280	− 7890	− 8.9	123170	− 7830	−6.0
F 9	*B*-2	0.0266	Observed	40500			41950			84400			123000		
			Calculated	35110	−5390	−13.3	42050	+ 100	+0.3	80420	−3980	− 4.7	123400	+ 400	+0.2
F 4	*C*	0.0501	Observed	38500			45180			89370			135200		
			Calculated	36540	−1960	− 5.1	43120	−2060	−4.6	81460	− 7910	− 8.9	125430	− 9770	−7.2
F 5	*C*-1	0.0502	Observed	41000			44810			88870			131000		
			Calculated	36540	−4460	−10.9	43130	−1680	−3.7	81460	− 7410	− 8.3	125440	− 5560	−4.2
F 1	*D*	0.2347	Observed	50000			50420			93110			142500		
			Calculated	41580	−8420	−16.8	46910	−3510	−7.0	85130	− 7980	− 8.6	134090	− 8410	−5.9
F 2	*D*-1	0.2424	Observed	45300			47430			92410			139000		
			Calculated	41720	−3580	− 7.9	47010	− 420	−0.9	85230	− 7180	− 7.8	134340	− 4660	−3.4
F 3	*D*-2	0.2292	Observed	44700			49000			92500			138900		
			Calculated	41480	−3220	− 7.2	46830	−2170	−4.4	85060	− 7440	− 8.0	133890	− 5010	−3.6
*F*13	*E*	0.5506	Observed	54000			54920			95460			152000		
			Calculated	45700	−8300	−15.4	50000	−4920	−9.0	88130	− 7330	− 7.7	142630	− 9370	−6.2
*F*10	*F*	0.8371	Observed	49500			52170			97000			156000		
			Calculated	48200	−1300	− 2.6	51870	− 300	−0.6	89950	− 7050	− 7.3	148380	− 7620	−4.9
*F*11	*F*-1	0.8525	Observed	55500			55920			97900			153000		
			Calculated	48320	−7180	−12.9	51960	−3960	−7.1	90030	− 7870	− 8.0	148650	− 4350	−2.8
*F*12	*F*-2	0.8258	Observed	54000			55000			100000			151400		
			Calculated	48110	−5890	−10.9	51810	−3190	−5.8	89890	−10110	−10.1	148170	− 3230	−2.1
*F*36	*G*	1.423	Observed	47500			53500			96380			150000		
			Calculated	51910	+4410	+ 9.3	54660	+1160	+2.2	92650	−3730	− 3.9	157610	+ 7610	+5.1
L 1	*H*-10		Observed	74000						127400			196500		
*F*80	*I*-3		Observed												
															Heated to 780° C.
*F*25	*B*	0.0132	Observed	34000			41000			83000			120000		
			Calculated	33840	− 160	− 0.5	41100	+ 100	+0.2	79500	− 3500	− 4.2	121820	+ 1820	+1.5
*F*22	*C*	0.0491	Observed	43000			44000			84150					
			Calculated	36490	−6510	−15.1	43090	− 910	−2.1	81420	− 2730	− 3.2			
*F*19	*D*	0.1380	Observed	42500			47000			87870			143000		
			Calculated	39550	−2950	− 6.9	45380	−1620	−3.4	83650	− 4220	− 4.8	130330	−12670	−8.9
*F*31	*E*	0.4743	Observed	43500			49500			90500			146000		
			Calculated	44900	+1400	+ 3.2	49390	− 110	−0.2	87540	− 2960	− 3.3	140860	− 5140	−3.5
*F*28	*F*	0.7940	Observed	42500			52000			94000			155000		
			Calculated	47870	+5370	+12.6	51620	− 380	−0.7	89710	− 4290	− 4.6	147590	− 7410	−4.8
*F*34	*G*	1.4015	Observed	51200			51700			95100			158500		
			Calculated	51790	+ 590	+ 1.2	54570	+2870	+5.6	92560	− 2540	− 2.7	154990	− 3510	−2.2
*F*72	*I*		Observed	85000						135400			207000		
*F*73	*I*-3		Observed	77000						145500					
															Heated to 751° C.
*F*50	*F*		Observed	44500			45900			91700			147500		
*F*27	*F*		Observed	40000			45000			89400			139000		
*F*56	*G*		Observed	45500			49900			95600			154500		
*F*26	*G*		Observed	35500			47000			91200			140000		
*F*29	*G*-1		Observed	50500			51250			94700			151000		
*F*32	*G*-2		Observed	44000			50300			96300			151000		

Containing 0.52 *Per Cent. Carbon*
0.030 *Per Cent. P*, 0.029 *Per Cent. S*, 0.22 *Per Cent. Si*)

Properties						Impact (Charpy)							Hardness			
Extension			Contraction of Area			Resilience, Foot-pounds							Brinell			
	Difference Observed—Calculated			Difference Observed—Calculated		Foot-pounds					Difference Observed—Calculated			Difference Observed—Calculated		
Per Cent.	Diff.	Per Cent. of Observed	Per Cent.	Diff.	Per Cent. of Observed	Min.			Max.	Average Foot-pounds	Diff.	Per Cent. of Observed	No.	Diff.	Per Cent. of Observed	Scleroscope
22.0			41.8			3.15	4.04	4.24	4.55	4.00						
21.5			32.7			2.76	2.76	2.95	3.15	2.90						
(1652° F.) = Ac_3 + 130° C.																
22.0			33.1			10.72			12.14	11.43			178			27
26.2	+4.2	+19.1	38.6	+5.5	+16.6					3.26	−8.17	− 71.5	162	−16	−9.0	
20.5			40.7			3.05	3.05	3.05	3.85	3.25			207			33.5
26.2	+5.7	+27.8	48.0	+7.3	+17.9					11.54	+8.29	+255.1	194	−13	−6.3	
21.0			39.5			3.15	3.15	3.15	3.15	3.15			207			34.2
19.5			38.3			2.18	2.47	2.56	2.76	2.49			215			30
26.2	+6.7	+34.4	48.0	+9.7	+25.3					11.61	+9.12	+366.3	194	−21	−9.8	
(1503° F.) = Ac_3 + 47° C.																
27.5			40.3			3.05	3.05	3.25	3.55	3.23			164			26
26.2	−1.3	−4.7	37.2	−3.1	− 7.7					2.64	−0.59	− 18.3	159	− 5	−3.0	
26.0			37.2										169			25
26.2	+0.2	+0.8	39.2	+2.0	+ 5.4					3.60			164	− 5	−3.0	
26.0			37.9			3.05	3.05	3.15	3.25	3.13			170			24
26.2	+0.2	+0.8	39.3	+1.4	+ 3.7					3.62	+0.49	+ 15.7	164	− 6	−3.5	
25.0			37.2			2.85	3.05	3.05	4.85	3.45			170			30
26.2	+1.2	+4.8	39.5	+2.3	+ 6.2					3.72	+0.27	+ 7.8	164	− 6	−3.5	
26.0			39.4										175			25
26.2	+0.2	+0.8	40.7	+1.3	+ 3.3					4.47			168	− 7	−4.0	
26.0			37.9										178			25
26.2	+0.2	+0.8	40.7	+2.8	+ 7.4					4.47			168	−10	−5.6	
26.5			42.2										179			26
26.2	−0.3	− 1.1	44.0	+1.8	+ 4.3					6.99			179	0	0	
26.0			40.3										181			25
26.2	+0.2	+ 0.8	44.1	+3.8	+ 9.4					7.06			178	− 3	−1.7	
26.0			40.3			2.95	3.05	3.05	3.55	3.13			184			28
26.2	+0.2	+ 0.8	43.9	+3.6	+ 8.9					6.95	+3.82	+122.0	178	− 6	−3.3	
26.5			44.3										190			29
26.2	−0.3	− 1.1	45.9	+1.6	+ 3.6					8.96			185	− 5	−2.6	
26.5			44.6										195			28
26.2	−0.3	− 1.1	46.9	+2.3	+ 5.2					10.03			189	− 6	−3.1	
25.0			42.8			3.25	3.55	4 04	5.06	3 97			195			29
26.2	+1.2	+ 4.8	46.9	+4.1	+ 9.6					10.17	+6.20	+156.2	189	− 6	−3.1	
25.0			41.3			3.05	3.25	3.45	3.55	3.33			196			32
26.2	+1.2	+ 4.8	46.9	+5.6	+13.6					10.07	+6.74	+202.4	189	− 7	−3.6	
26.0			44.0			8.30			8.98	8.64			196			27
26.2	+0.2	+ 0.8	48.2	+4.2	+ 9.5					11.80	+3.16	+ 36.6	195	− 1	−0.5	
17.5			46.6			2.18	2.76	3.75	4.55	3.31			255			39
						3.05			4.04	3.55						
(1436° F.) = Ac_3 + 10° C.																
26.0			36.7			2.85	3.05	3.15	3.15	3.05			169			24
26.2	+0.2	+ 0.8	38.1	+1.4	+ 3.8					3.04	−0.01	− 0.3	161	− 8	−4.7	
26.5			41.2			3.15	3.55	3.65	9.32	4.92						
26.2	−0.3	− 1.1	40.7	−0.5	− 1.2					4.44	−0.48	− 9.8	167			
26.5			45.5			4.04	4.24	5.90	9.90	6.02			179			26
26.2	−0.3	− 1.1	42.8	−2.7	− 5.9					6.00	−0.02	− 0.4	174	− 5	−2.8	
27.0			46.0			4.14	7.96	8.18	10.13	7.60			181			27
26.2	−0.8	− 3.0	45.6	−0.4	− 0.9					8.58	+0.98	+ 12.9	184	+ 3	+1.7	
26.0			46.8			8.87			10.60	9.73			191			28
26.2	+0.2	+ 0.8	46.8	0	0					9.96	+0.23	+ 2.4	189	− 2	−1.0	
26.5			48.0										187			29
26.2	−0.3	− 1.1	48.1	+0.1	+ 0.2					11.75			195	+ 8	+4.3	
17.5			51.1			3.05	3.05	4.04	5.58	3.93						
15.0			51.9			5.27			7.19	6.23						
(1384° F.) = Ac_3 − 19° C.																
24.5			45.1			2.66	3.55	5.16	5.27	4.16			181			28
24.0			42.1			3.25			3.45	3.36			182			28
27.0			45.4			2.95	3.05	3.95	4.95	3.72			188			28
24.0			43.4			3.85	3.85	4.65	5.06	4.35			187			28
25.0			44.3			2.95			3.15	3.05			192			29
21 0			43 5			3.45			6 32	4.89			194			31

TABLE. 3.—*Physical Properties of Steel* (*Composition, 0.75 Per Cent. C, 0.53 Per Cent. Mn,*

	Treatment			Proportional Limit			Yield Point			Tensile Strength			Tensile Stress at Rupture		
					Difference Observed—Calculated			Difference Observed—Calculated			Difference Observed—Calculated			Difference Observed—Calculated	
Heat No.	Cooling Method	Rate: ° C. per Sec. 740°–585°		Pounds per Sq. In.	Diff.	Per Cent. of Observed	Pounds per Sq. In.	Diff.	Per Cent. of Observed	Pounds per Sq. In.	Diff.	Per Cent. of Observed	Pounds per Sq. In.	Diff.	Per Cent. of Observed
As rolled				42000						118500					
Annealed, 900° C. (1652° F.), ½ hr., furnace cooled				33000			43500			100000					
Heated to 900° C															
F 98	A-1	0.0149	Observed	41000			43500			103150			119000		
			Calculated	35190	− 5810	−14.2	46130	+ 2630	+ 6.0	94140	− 9010	− 8.7	122050	+ 3050	+ 2.6
F 65	D	0.2140	Observed	42000						112050					
			Calculated	41290	− 710	− 1.7	49900			103650	− 8.400	− 7.5	133360		
Heated to 817° C															
F 7	B	0.0219	Observed	36300						107890					
			Calculated	35780	− 520	− 1.4	46490			95060	−12830	−11.9	122910		
F 4	C	0.0472	Observed	39100						109340					
			Calculated	37190	− 1910	− 4.9	47370			97250	−12090	−11.1	125210		
F 5	C-1	0.0479	Observed	41800						109330			126000		
			Calculated	37220	− 4580	−11.0	47390			97310	−12020	−11.0	125270	− 730	− 0.6
F 6	C-2	0.0459	Observed	37000			47250			104900					
			Calculated	47130	+ 130	+ 0.4	47330	+ 80	+ 0.2	97170	− 7730	− 7.4	125110		
F 1	D	0.2225	Observed	44500						117820			139600		
			Calculated	31420	− 3080	− 6.9	49980			103860	−13960	−11.8	133660	− 5940	− 4.3
F 2	D-1	0.2322	Observed	44200						117320					
			Calculated	41570	− 2630	− 6.0	50080			104090	−13230	−11.3	134000		
F 13	E	0.5000	Observed	46500						121320					
			Calculated	44670	− 1830	− 3.9	51990			108930	−12410	−10.2	141480		
F 14	E-1	0.4740	Observed	52000						122810					
			Calculated	44430	− 7570	−14.6	51840			108550	−14260	−11.6	140860		
F 15	E-2	0.4952	Observed	46500						124000					
			Calculated	44630	− 1870	− 4.0	51970			108860	−15140	−12.2	141360		
F 10	F	0.7570	Observed	55000						124060					
			Calculated	46700	− 8300	−15.1	53250			112110	−11950	− 9.6	146890		
F 16	G	1.091	Observed	60500						128560					
			Calculated	48740	−11760	−19.4	54510			115280	−13280	−10.3	152680		
F 20	G-1	0.9810	Observed	52500						127500					
			Calculated	48130	− 4370	− 8 3	54130			114330	−13170	−10.3	150890		
F 21	G-2	1.131	Observed	47000						125500					
			Calculated	48960	+ 1960	+ 4.2	54590			115470	−10030	− 8.0	153290		
F100	H	1.281	Observed	55000						125500					
L 2			Calculated	49740	− 5260	− 9.6	54990			116500	− 9000	− 7.2	155580		
	H-20		Observed	78500						161500					
F 81	I	1.703	Observed	65000						137750			185000		
			Calculated	51130	−13870	−21.3	56280			119740	−18010	−13.1	161330	−23670	−12.8
F 79	I-4		Observed	80000						165400					
Heated to 800° C.															
F101	D	0.1275	Observed	41000			44000			105050					
			Calculated	39640	− 1360	− 3.3	48890	+ 4890	+11.1	101090	− 3960	− 3.8	129850		
F 17	G	0.9758	Observed	46000						121500			153000		
			Calculated	48090	+ 2090	+ 4.5	54110			114270	− 6230	− 5.1	150780	− 2220	− 1.5
Heated to 780° C.															
F 25	B	0.0119	Observed	32000			38000								
			Calculated	35540	+ 3540	+11.1	46350	+11650	+30.7	94690			122560		
F 19	D	0.202	Observed	40000						103900					
			Calculated	41090	+ 1090	+ 2.8	49780			103340	− 560	− 0.5	132920		
F 31	E	0.4446	Observed	45000						110500					
			Calculated	44140	− 860	− 1.9	51670			108110	− 2390	− 2.2	140140		
F 73	I-3		Observed	72000						181000			240000		
Heated to 751° C.															
F 41	D	0.1442	Observed	40000						99900			144500		
			Calculated	40010	+ 10	0.0	49110			101660	+ 1760	+ 1.8	130610	−13890	− 9.6
F 50	F	0.8245	Observed	52000			53000			109400			153000		
			Calculated	47160	− 4840	− 9.3	53530	+ 530	+ 1.0	112820	+ 3420	+ 3.1	148850	− 4150	− 2 7
F 27	F	0.6485	Observed	42000						105600					
			Calculated	45910	+ 3910	+ 9.3	52760			110870	+ 5270	+ 5.0			
F 56	G	1.24	Observed	49500			56400			115000			158000		
			Calculated	49520	+ 20	0 0	54990	− 1410	− 2.5	116500	+ 1500	+ 1.3	154990	− 3010	− 1.9
F 26	G	0.9936	Observed	31000						109600					
			Calculated	48200	+17200	+55.5	54180			114440	+ 4840	+ 4.4			
F 29	G-1	1 076	Observed	48000			56000			115500			150000		
			Calculated	48670	+ 670	+ 1.4	54470	− 1530	− 2.7	115180	− 320	− 0.3	152430	+ 2430	+ 1.6
F 32	G-2	1.0472	Observed	50000			58500			118000					
			Calculated	48500	− 1500	− 3 0	54360	− 4140	− 7.1	114910	− 3090	− 2.6	151960		

Containing 0.75 *Per Cent. Carbon*
0.031 *Per Cent. P,* 0.027 *Per Cent. S,* 0.11 *Per Cent. Si)*

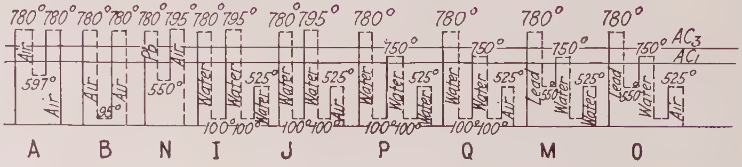
Properties						Impact (Charpy)							Hardness			
Extension			Contraction of Area			Resilience, Foot-pounds							Brinell			
	Difference Observed—Calculated			Difference Observed—Calculated		Foot-pounds				Average Foot-pounds	Difference Observed—Calculated			Difference Observed—Calculated		
Per Cent.	Diff.	Per Cent. of Observed	Per Cent.	Diff.	Per Cent. of Observed	Min.			Max.		Diff.	Per Cent. of Observed	No.	Diff.	Per Cent. of Observed	Scleroscope
11.5			22.6			1.99	2.09	2.09	2.27	2.11						
10.0			12.9			1.99	2.09	2.09	2.09	2.06						
(1652° F.) = Ac_3 + 159° C.																
12.0			12.9			1.89	1.99	2.09	2.09	2.02						
18.6	+6.6	+55.0	26.9	+14.0	+108.5					1.12	−0.90	− 44.6	189			
12.5			15.0			1.80	1.99	1.99	1.99	1.94			228			37
18.6	+6.1	+48.8	30.7	+15.7	+104 6					2.42	+0.48	+ 24.7	212	−16	− 7.0	
(1503° F.) = Ac_3 + 76° C.																
13.0			13.0										216			3
18.6	+5.6	+43.1	27.4	+13.6	+104.6								192	−24	−11.1	
14.5			16.3										218			32
18.6	+4.1	+28.3	28.5	+12.2	+ 74.8								198	−20	− 9.2	
13.0			14.8										222			32
18.6	+5.6	+43.1	28.5	+13.7	+ 92.6								198	−24	−10.8	
12.5			14.4			2.09	2.09	2.18	2.56	2.23			217			35
18.6	+6.1	+48.8	28.5	+14.1	+ 97.9					1.55	−0.68	− 30.5	198	−19	− 8.8	
14.0			18.5										236			36
18.6	+4.6	+32.9	30.8	+12.3	+ 66.5								213	−23	− 9.7	
14.0			17.8										240			34
18.6	+4.6	+32.9	30.9	+13.1	+ 73.6								213	−27	−11.3	
15.0			22.4										241			35
18.6	+3.6	+24.0	32.1	+ 9.7	+ 43.3								222	−19	− 7.9	
14.0			20.2			1.34	1 61	1.61	2.09	1.66			241			36
18.6	+4.6	+32.9	32.0	+11.8	+ 58.4					3.04	+1.38	+ 83.1	222	−19	− 7.9	
14.5			20.5			1 61	1 61	2.09	2.09	1.85			244			40
18.6	+4.1	+28.3	32.1	+11.6	+ 56 6					3.08	+1.23	+ 66.5	222	−19	− 7.9	
15.5			23.1										249			35
18.6	+3.1	+20.0	32.7	+ 9.6	+ 41.6								228	−21	− 8.4	
15.5			25.8			1.61	1.80	2 09	5.06	2.64			255			36
18.6	+3.1	+20.0	33.3	+ 7.5	+ 29.1					3.88	+1.24	+ 47.0	233	−22	− 8.6	
14.0			26.2			1.34	1.52	1.70	1.99	1.64			265			36
18.6	+4.6	+32.9	33.2	+ 7.0	+ 26.7					3.76	+2.12	+129.3	232	−33	−12.5	
14.0			24.8			1.52			1.61	1.57			265			37
18.6	+4.6	+32.9	33 4	+ 8.6	+ 34.7					3.92	+2.35	+149.7	234	−31	−11.7	
13.0			25.8			1.16	1 43	1.61	1.80	1.50			255			38
18.6	+5.6	+43.1	33.6	+ 7.8	+ 30.2					4.06	+2.56	+170.7	236	−19	− 7.6	
14.5			35.0			1.99	2.09	2.56	2.66	2.32			321			49
12.5			32.0			1.25	1.34	1.34	1.61	1.39						
18.6	+6.1	+48.8	34.1	+ 2.1	+ 6.6					4.51	+3.12	+224.5	240			
12.0			38.8			2.76	3.15	3.15	3.45	3.13			340			55
(1472° F.) = Ac_3 + 59° C.																
14.0			20.0			1.43			1.61	1 52			217			33
18.6	+4.6	+32.8	29.9	+ 9.9	+ 49.5					2 08	+0 56	+ 36.8	207	−10	− 4.6	
15.0			25.0			1.16			1.43	1.35			252			37
18.6	+3.6	+24.0	33.2	+ 8.2	+ 32.8					3.75	+2.40	+177.8	232	−20	− 7.9	
(1436° F.) = Ac_3 + 37° C.																
						1.25	1.25	1.80	1.80	1.53						
18.6			27.2							1.05	−0.48	− 31.4	188			
16.0			24.0			1.61	1.99	2.09	2.09	1.95			215			33
18.6	+2.6	+16 3	30.6	+ 5.4	+ 22.5					2.38	+0.43	+ 22.1	211	− 4	− 1.9	
15.5			24.0			1.99	2.09	2.18	2.47	2.18			228			35
18.6	+3.1	+20.0	31.9	+ 7.9	+ 32.9					2.99	+0.81	+ 37.2	221	− 7	− 3.1	
11.0			36.5			4.04	4.04	6.00	6.65	5.18						
(1384° F.) = Ac_3+ 10° C.																
21.0			35.0			1.99	1.99	2.09	2.27	2.13			203			30
18.6	−2.4	−11.4	30.1	− 4.9	− 14.0					2.16	+0.03	+ 1.4	207	+ 4	+ 2.0	
19.0			32.8			7.19			8.30	7.75			223			34
18.6	−0.4	− 2.1	32.9	+ 0.1	+ 0.3					3.57	−4.18	− 53.9	229	+ 6	+ 2 7	
17.0			26.2			2.09			2.47	2.28			218			31
18.6	+1.6	+ 9.4	32.5	+ 6.3	+ 24.0					3.33	+1.05	+ 46.1	226	+ 8	+3.7	
19.0			31.1			1.70	1.80	1.80	1.89	1.79			228			35
18.6	−0.4	− 2.1	33.6	+ 2.5	+ 8.0					4.02	+2.23	+124.6	235	+ 7	+ 3.1	
15.0			22.5			1.70			2.56	2.13			228			31
18.6	+3.6	+24.0	33.2	+10.7	+ 47.5					3.77	+1.64	+ 77.0	232	+ 4	+ 1.8	
17.0			27.8			1.99			2.09	2.04						
18.6	+1.6	+ 9.4	33.3	+ 5.5	+ 19.8					3.86	+1.82	+ 89.2				
15.5			27.0			1.52			1.70	1.61						
18.6	+3.1	+20.0	33.3	+ 6.3	+ 23.3					3 83	+2.22	+137.9				

TABLE 4.—*Physical Properties of Quenched and Drawn Steel Containing* 0.52 *Per Cent. Carbon*
(*Composition,* 0.52 *Per Cent. C,* 0.55 *Per Cent. Mn,* 0.030 *Per Cent. P,* 0.029 *Per Cent. S,* 0.22 *Per Cent. Si*)

Quenched in Water from ° C.	Time at Quenching Temp. Min.	Tensile Properties						Impact (Charpy)				
		Pounds per Square Inch				Per Cent.		Foot-pounds				
		Prop. Limit	Yield Point	Tensile Str.	Stress at Rup.	Ext.	Con.	Min.			Max.	Aver.
As rolled		39,000	.....	95,250	143,000	22.0	41.8	3.15	4.04	4.24	4.55	3.99
Annealed, 820° C. (1508° F.), ½ hr., cooled in air		50,000	54,000	91,500	142,000	23.5	41.8	4.24	4.24	4.44	8.30	5.30
Annealed, 900° C. (1652° F.), ½ hr., furnace cooled		42,500	44,000	78,000	147,000	26.5	57.7	2.95	3.05	3.05	3.05	3.02
Quenched as shown and drawn at 300° C. (572° F.), 2 hr., cooled in air												
780	10	89,000	.....	144,200	......	17.0	40.9	2.76	2.76	2.85	2.95	2.83
845	10	80,000	.....	151,000	......	12.0	40.2	2.09	2.18	2.95	3.05	2.57
910	10	80,000	.....	173,500	......	6.5	18.7	2.18	2.66	3.05	3.85	2.93
975	10	80,000	.....	186,500	202,000	3.5	8.0	1.80	1.99	2.09	2.47	2.09
Quenched as shown and drawn at 450° C. (842° F.), 30 min., cooled in air												
780	10	85,000	98,750	137,250	204,000	18.5	47.7	2.66	3.05	3.15	3.55	3.10
845	10	70,000	.....	140,000	191,000	15.5	42.4	2.09	3.15	3.25	3.25	2.93
910	10	85,000	.....	156,750	212,000	11.5	37.1	1.99	3.05	3.55	5.58	2.54
975	10	77,000	.....	143,500	183,000	10.0	29.0	6.11	6.22	10.48	11.07	8.47
Quenched as shown and drawn at 525° C. (977° F.), 15 min., cooled in air												
780	10	88,000	92,500	130,750	200,000	18.5	49.4	3.05	3.45	3.55	4.04	3.52
845	10	80,000	.....	134,750	199,000	15.5	46.0	2.66	3.05	4.14	4.65	3.62
910	10	85,000	.....	138,750	197,000	14.0	41.5	7.19	8.53	9.32	11.78	9.21
975	10	85,000	.....	136,500	187,000	12.5	36.2	13.95	14.21	14.33	15.21	14.42
Quenched as shown and drawn at 675° C. (1247° F.), 10 min., cooled in air												
780	10	68,000	74,500	103,250	......	24.0	60.3	3.15	4.14	6.11	6.97	5.09
845	10	74,000	.....	106,000	......	21.5	59.8	2.56	3.25	3.85	5.69	3.84
910	10	70,000	.....	104,600	195,000	21.0	61.5	3.95	4.44	8.07	12.98	7.36
975	10	83,000	87,500	103,500	......	22.0	59.8	22.16	23.30	24.74	25.03	23.80

TABLE 5.—*Physical Properties of Steel Containing 0.52 Per Cent. Carbon after Double Refinings (Composition, 0.52 Per Cent. C, 0.55 Per Cent. Mn, 0.030 Per Cent. P, 0.029 Per Cent. S, 0.22 Per Cent. Si)*

	Treatment												Tensile Properties						Impact (Charpy)					Hardness	
	First Heating				Second Heating				Third Heating				Pounds per Square Inch				Per Cent.		Foot-pounds						
	Heat-ed to °C	Held Min.	Cooled		Heat-ed to ° C.	Held Min.	Cooled		Heat-ed. to °C	Held Min.	Cooled		Propor-tional Limit	Yield Point	Tensile Strength	Stress at Rupture	Ext.	Con.	Min.			Max.	Aver.	Brinell	Sclero-scope
			In	To ° C			In	To ° C			In	To ° C													
As rolled													39,000	......	95,250	143,000	22.0	41.8	3.15	4.04	4.24	4.55	3.99		
Annealed, 820° C. (1508° F.), ½ hr., cooled in air													50,000	54,000	91,500	142,000	23.5	41.8	4.24	4.24	4.44	8.30	5.30		
Annealed, 900° C. (1652° F.), ½ hr., furnace cooled													42,500	44,000	78,000	147,000	26.5	57.7	2.95	3.05	3.05	3.05	3.02		
A	780	10	Air	597	780	10	Air						50,000	52,500	92,000	145,600	26.5	42.8	8.18	8.98	9.10	9.44	9.55	187	30
													52,500	55,000	91,500	151,000	25.5	46.8	9.55	10.25	10.36	10.60		191	25
B	780	10	Air	95	780	10	Air						52,500	54,500	95,750	150,000	24.0	43.9						190	25
													50,000	53,100	94,200	......	....	....	6.65	7.19	7.85	8.30	7.50	193	28
N	780	10	Pb	550	795	3	Air						50,000	52,500	87,650	......	27.5	53.0							
													50,000	53,500	88,000	......	27.0	53.0	9.44	9.66	10.60	10.72	10.10		
I	780	10	Water	100	795	3	Water	100	525	10	Water		81,000	87,400	122,400	201,000	19.5	55.2							
													70,000	91,000	125,000	207,000	18.0	56.4	7.85	11.19	11.30	11.78	10.55	269	45
J	780	10	Water	100	795	3	Water	100	525	10	Air		77,000	87,000	117,500	200,000	21.5	57.5							
													80,000	92,000	120,000	199,000	20.0	56.0	12.98	14.33	14.97	17.41	14.92		
P	780	10	Water	100	750	10	Water	100	525		Water		69,000	73,500	98,700	178,000	22.5	59.3							
													57,000	72,000	98,200	191,000	23.5	60.8	4.65	4.85	4.85	6.32	5.17		
Q	780	10	Water	100	750	10	Water	100	525	10	Air		75,000	101,000	136,250	218,000	17.5	54.0							
													78,000	103,000	139,000	......	16.0	51.6	5.27	5.69	5.90	11.78	7.16		
M	780	10	Pb	550	750	10	Water	100	525	15	Water		63,000	68,500	104,500	181,000	21.5	54.0	2.56	2.66	2.85	2.85	2.73	231	38
O	780	10	Pb	551	750	10	Water	100	525	15	Air		61,000	64,000	100,375	181,000	22.5	56.7	2.66	3.15	3.75	4.55	3.53	241	42



Application of Colloid Chemistry to Production of Clean Steel*

By H. W. Gillett,† Ithaca, N. Y.

(New York Meeting, February, 1923)

Many of the parts of motor cars, aircraft, etc., that require strong light construction, hence must be made of high-quality steel, are stressed to the maximum limit only in a very small volume. In parts subject to bending stress a line or even a point may take the maximum stress.

A series of endurance tests under repeated bending in progress in the U. S. Bureau of Mines laboratories, like similar tests by other investigators, has shown the great difference in life of sister bars of the same steel, handled in the same way and tested at the same nominal stress. It is known that a surface notch or scratch greatly increases the local stress over the nominal calculated stress; internal notches, such as those formed by inclusions or sonims, probably have a similar effect.

From the tetss made by the Bureau of Mines, the cleanliness of the steel at the dangerous section, or point of maximum stress, seems to be one of the most important factors in determining the life of the piece. The more localized the stress, the more does chance, rather than the average properties, determine the life of the steel. The usual test, a tension test of a bar taken longitudinally, is not sensitive to the local drop in strength (or tendency to increase the true local stress over the nominal) because of inclusions, which probably have a more injurious effect in hard heat-treated steels, such as must be used when great strength is sought, than in softer steels.

Users of steel who make transverse tests find surprisingly low ductility on transverse bars of dirty steel, though the longitudinal tests may be good. Bending tests, reverse bending tests, and notch-bar impact tests, all of which as usually made test a definite limited volume rather than a large volume, are sensitive to the effect of inclusions and tend to disclose dirty steel. These tests, coupled with microscopic examination of unetched specimens, will show the general condition of a steel, but as the cleanliness may vary widely in very small distances, none of the tests will show the condition at the danger point of a piece in service; hence the factor of safety, with its attendant increase of weight, must be used.

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† Chief Alloy Chemist, U. S. Bureau Mines, Ithaca Field Office, Sibley College, Cornell University.

A large proportion of failures in service are due to fatigue breaks, *i.e.*, those resulting from progressive failure that starts from a definite nucleus. Such a failure is often directly traceable to an inclusion. In many other cases, the connection may be missed because, in polishing the specimen for microscopic examination, the material at the actual nucleus must be removed, thus destroying the evidence.

Other sources of nuclei for fatigue failure are poor surface finish; irregularities in structure, because of poor annealing or poor heat-treatment; unrelieved quenching strains; etc., but these causes are more or less avoidable by the user of the steel. Inclusions cannot be eliminated from the steel; they may only be prevented. Once dirty steel is frozen nothing can be done to remedy it.

If steel were transparent, so that the dirt would be obvious, more methods for its elimination would have been sought and greater care would be taken in its prevention. The need for clean steel is becoming more and more apparent, and more and more pressure is going to be brought to bear on the maker to produce it.

Doctor Mathews,[1] calling attention to it in connection with the electric furnace, says "when users acquire a full appreciation of what clean sound steel means in terms of national efficiency, safety, and economy, we shall see more rapid growth." Almost every page of the recent book by Giolitti[2] emphasizes the bad effect of dirt in steel. A large list of authorities can be cited to show that dirty steel is dangerous, but the situation is so obvious that a piling up of proof is scarcely needed All that is said about dirty steel is equally true of non-ferrous alloys, duralumin in dirigible construction for example; but this paper will be confined to steel.

What constitutes dirty steel is a matter of definition and comparison and the definition will vary with the point of view. We shall define clean steel as steel with nothing that is not in solid solution in the austenite field. Everything else is dirt. No such steel has been made, and

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[1] J. A. Mathews: Electric Furnace in Refining Iron and Steel. *Iron Age* (1922) **110,** 1127; *Chem. & Met. Eng.* (1922) **27,** 872.

[2] "Heat Treatment of Soft and Medium Steels." Trans. by E. E. Thum and D. G. Vernaci, 141, 283, 360. 1921.

FIG. 1.—PRIESTLEY'S ELECTRIC STEEL FROM WHICH THE OXIDES AND NON-METALLIC ENCLOSURES HAVE BEEN REMOVED. UNETCHED. × 25.

FIG. 2.—ELECTRIC STEEL FROM A LARGE STEEL COMPANY. UNETCHED. × 100.

FIG. 3.—TYPICAL FIELD, ELECTRIC STEEL FROM ANOTHER LARGE STEEL COMPANY. UNETCHED. × 100.

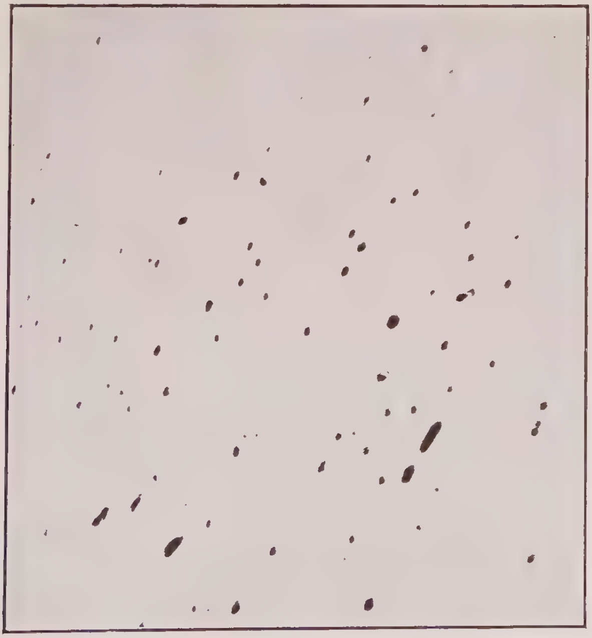

FIG. 4.—VERY DIRTY FIELD, ON SAME CROSS-SECTION OF $\frac{3}{8}$-IN. BAR AS FIG. 3. UNETCHED. × 100.

perhaps never will be. Comparatively clean steel is made, for example the electric furnace steel described by Priestley.[3] Mathews says that Priestley's results are due to clean steel; but Fig. 1 shows that this is only relatively clean steel. Fig. 2 shows a dirty portion of an electric furnace steel from a large steel company and Figs. 3 and 4 show a typical field and a local group of inclusions on the same cross-section of a 3/8-in. rod of electric-furnace steel. These photomicrographs are from endurance test bars of low life compared with cleaner specimens of the same steels.

While the photomicrographs show that even electric-furnace steel from careful and experienced makers is not necessarily truly clean, generally speaking, crucible and electric steels are cleaner than acid open-hearth, which is cleaner than basic open-hearth, and this, in turn, is cleaner than converter steel.

As Giolitti points out, the source of the dirt (dirt is a shorter and more expressive word than inclusions or sonims) lies in the fact that molten steel is an emulsion of liquid steel and liquid or solid dirt, sulfides, oxides, silicates, slag, etc. The amount of emulsified dirt may be reduced by using raw materials relatively free from dirt, by deoxidation, and by combining the sulfur as manganese sulfide and washing this out with a slag that will convert it to calcium sulfide, which is soluble in the slag but insoluble in the steel. It is essential to the breaking up of the emulsion that time be given for the steel to lie quiet and allow the dirt to rise, just as cream rises in milk. Complex deoxidizers, such as Mg-Mn-Si, are sometimes used with the object of forming an oxide mixture of low melting point and viscosity, which will more readily coalesce and rise.

Agitation of the bath previous to the quiet stage probably makes for coalescence and agglomeration of the dirt into larger particles, which will rise more readily though there is the attendant danger of emulsifying with the steel some slag that was not previously in suspension.

Some authorities advocate "boiling" the bath, others prefer to hold the agitation to the very minimum; all agree that a final quiet stage under non-oxidizing conditions is essential. Crucible melting allows this to a high degree, electric-furnace melting to almost as high a degree, while open-hearth melting is less favorable to those conditions and converter melting is decidedly unfavorable.

The methods of melting best calculated to allow the breaking of the emulsion give the cleanest and highest quality steel. While gas absorption also affects the quality, there is a definite relation between de-emulsified and high-quality steel. It is therefore pertinent to inquire whether there may not be methods of breaking up the emulsion other than just letting it set. While many of the inclusions in steel are so large that the emulsified or suspended particles of dirt are well above the critical dimensions of those in an aqueous colloidal emulsion or suspension, the mechanism of the retention of the dirt in suspension seems closely analogous to that of aqueous colloidal solutions. Hence the colloid chemist should be able to suggest other lines of attacking the problem. Prof. W. D. Bancroft, of Cornell University, says:

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[3] Effect of Sulfur and Oxides in Ordnance Steels. *Trans.* (1922) **67**, 331.

> The problem of eliminating sonims is, probably, essentially one in colloid chemistry, the cast metal containing sonims constituting a frozen colloidal solution. We are, probably, dealing with a case of emulsification of the slag and sulfides by the metal. The reverse case is well known. In the electrolysis of fused caustic soda to metallic sodium, at a slightly too high temperature, we get an emulsion of sodium in fused caustic as a metal fog.
>
> It might be possible to clean the metal from sonims by introducing more sonims of larger size, because of the greater tendency to agglomeration. One can take a dilute suspension of kaolin in water, from which only the coarser particles will settle in a reasonable time; a more concentrated suspension will settle clear because there are more coarse particles, which hit and carry with them the finer ones. If the particles are less dense than the medium they rise rather than sink, but the principle is the same. Cream rises faster and more completely in a deep narrow vessel than in a wide shallow one.
>
> There is also the possibility of adsorbing the sonims into other materials. A suspension of carbon black can be filtered through filter paper even though the pores of the filter are larger than the particles of black, because the black is adsorbed by or sticks to the cellulose of the filter paper. Wool will adsorb alumina completely from a dilute suspension in water. We are not able to predict what substances would strongly adsorb sulfides or slag particles and also have no deleterious effect on steel; but it is quite possible that there are such substances.
>
> The cleansing action of titanium and vanadium in steel is thought to have a specific coagulating effect on the sonims, enlarging them above the critical size so they will rise. Others say their oxidation products form inclusions above the critical size so they will rise. Increasing coalescence by making the sonims more fusible, *i.e.* fluxing them, is a possibility. Stirring, as by the pinch effect of an induction furnace, might aid coalescence.
>
> The question of after treatment with special slags should be considered. The slags in use were chosen, primarily, on chemical considerations and it does not follow that they are the best for removing sonims. If acid open-hearth steel is cleaner than basic, the difference in the slag gives a lead worth following.
>
> We have a fairly satisfactory theory for aqueous colloidal solutions but know little, as yet, about non-aqueous colloidal solutions, and it will be necessary to develop the theory as we go along. This does not mean that the problem of removing sonims is insoluble; there are definite and promising lines of attack. It is not a vague problem where we have to try things blindly, but a legitimate scientific and technical problem in which there is every reason to expect that real progress can be made. It is not, however, an overnight job; it is a long difficult task that must not be handled in a desultory or intermittent fashion."

Professor Bancroft thus encourages us to believe that a systematic attack on dirty steels along the line of studying the fundamentals of colloid chemistry applied to molten metals might bring results in time. Much information is needed on which small-scale laboratory tests would be of value, for while any modified or new methods must ultimately be tested on a large scale, the basic facts can be best studied on a small scale.

The evils of dirty steel are greatly appreciated by aircraft and automobile engineers, who have the problem of excess weight to contend with. The demand for clean dependable steel, however, will spread beyond these fields. A quality product always extends its field. Mathews calls the electric steel rail "still a desired possibility." Even the relatively clean steel of the electric furnace is too expensive for large tonnage uses. While the electric furnace would probably prove a desirable adjunct in the production of clean steel, even were other methods developed for removing the dirt, other methods might be found that would apply to open-hearth melting; in that case open-hearth steel might have the quality of present electric steel and the quality of electric steel might be greatly improved.

There are doubtless various paths leading to clean steel. Only one of these, a final quiet period in electric or crucible furnaces, is so far in use; but the method is not infallible. Until a systematic study is made of the methods that the colloid chemist can suggest, we have only scratched the surface of the possibilities. No stone should be left unturned in the effort to produce clean dependable steel for use where safety to life or limb is involved and where requirements of lightness make it desirable to use a low factor of safety.

The Bureau of Mines deems this a problem of great importance which should receive attention. With Professor Bancroft's aid, a fairly definite program of what to try first and how to go ahead has been planned, and as a problem in process metallurgy it falls in the legitimate field of the Bureau. But, inasmuch as the Bureau's funds are alloted for definite purposes, it is at present impossible for the Bureau to attack the problem experimentally except in a desultory and intermittent fashion, which would be a waste of time. The next best thing seems to be to discuss the problem, as has been done here, so as to convince the users and producers of steel, and non-ferrous alloys as well, that dirty metal is unreliable, often dangerous. Dirty metal is an emulsion; relatively clean metal is produced by one method of breaking up an emulsion; other methods are possible, which might be cheaper or more efficacious. To find such methods we should build on the present knowledge of aqueous colloids and develop fundamental knowledge of colloids in molten metals.

DISCUSSION

Albert Sauveur, Cambridge, Mass.—The author calls to our attention the well-known fact that when a piece of forged steel is tested in the direction in which it has been worked, and also at right angles to that direction, different results are obtained. He is inclined to attribute this to the presence of inclusions that have been elongated in the direction of the work, such, for instance, as manganese sulfide. Certainly inclusions may have something to do with it, but the difference in physical properties imparted by work is caused chiefly by dendritic segregation. That difference would still exist in the complete absence of inclusions.

HAAKON STYRI, Philadelphia, Pa.—As to inclusions, apparently the author means that when steel is melted, and is free from all foreign matter, it will also be free from inclusions in the solid state. But the difference in solubility in the solid state and the liquid state must be considered. For instance, if there is a silicate or an oxide included in the solid state, it will dissolve partly, at least, when the steel is melted. In other words, it might not be possible to get a steel perfectly free from foreign inclusions at any time because of the solubility of such material when the steel is melted. The steel maker is therefore not able to get out all of them; he does not put them in, they are there to begin with.

H. W. GILLETT.—I realize that the solubility of oxides and other inclusions in liquid steel is disputed; we have little conclusive data on that point. But the indications are that the bulk of the ordinary inclusions are insoluble to almost the same degree in both the liquid and the solid state.

As to the question of longitudinal-transverse results, it is not impossible that dendritic structure may be tied up with the inclusions or non-metallic impurities. Giolitti emphasizes that point when he says: "Longitudinal and transverse properties do not differ greatly in pure steels but do differ widely in dirty steels." Also Aitchison, who says: "Although it is recognized that the toughness of a steel varies with the direction, it is not generally recognized that the difference is largely due to the presence of slag and non-metallic impurities and their distribution within the ingot."

Priestley brought out the same point at the meeting of the Institute last year. There is much evidence that inclusions do play an important part in the difference between longitudinal and transverse properties.

HAAKON STYRI.—I would like to contradict the statement that the solubility of impurities in the solid and liquid state are the same. I have a specimen that contains about 0.10 per cent. oxygen, practically free from carbon, manganese, and silicon, that was kept molten in the induction furnace for a considerable time; it is the dirtiest steel I have seen under the microscope. Iron oxide is very soluble in the liquid state; it is not so soluble in the solid state. The solubility of iron oxide or oxygen will depend on the amount of other elements present, such as carbon, manganese, and silicon.

The extent to which small foreign particles which are undissolved or are precipitated from solution are expelled from the molten steel depends on the methods used, the skill and care of the operator, and the time allowed. Good success is obtained by conducting the reducing period so that the boiling gradually diminishes to quiescence, and by controlling the temperature carefully. These are old and well known methods; but too few know how to use them.

Albert Sauveur, Cambridge, Mass.—It is obvious that if the steel is very pure, the difference of properties in the transverse and longitudinal directions will not be so great. Pure steel, however, does not mean simply absence of inclusions, it also means absence of impurities; therefore, also, absence or nearly complete absence of dendritic segregation.

H. W. Gillett.—While the evidence is somewhat contradictory, it is probable that Doctor Styri is correct in stating that iron oxide is soluble in liquid steel and thrown out on freezing. The aim in deoxidation is to convert the iron oxide into silicon, manganese, or other oxides which are practically insoluble in liquid steel. With this step accomplished the next is the removal of the insoluble oxides and sulfides. It is no argument against the use or the importance of the second step on properly deoxidized steel that it is unavailing on improperly made steel, if indeed, material with 0.10 per cent. oxygen can be called steel. I should call it an abortion.

I do not grasp Professor Sauveur's terminology. I should class included "dirt" as one sort of impurity. Of course, a dendritic structure is undesirable. Let us avoid that by all means available, as we avoid an oxidized steel, but let us distribute the blame for poor steel among all the causes, of which inclusions are a cause important enough to avoid by means understood at present and by better means if they can be found.

George F. Comstock, Niagara Falls, N. Y. (written discussion).—I would suggest that the statement that "inclusions cannot be eliminated from the steel" be corrected by inserting the word "solid" before "steel."

The distinction between sulfide inclusions and slag inclusions in steel seems fundamental to the writer; yet this distinction is seldom made. Photomicrographs showing dark spots on a brighter field do not mean much unless we know what the spots represent. Sulfides occur in steel in proportion to its content of sulfur; the higher the sulfur content, the more sulfide inclusions there are in the metal. As we cannot make steel without sulfur, is it fair to call a steel dirty merely because a photomicrograph of it shows the presence of sulfide inclusions?

Aside from all question of the sulfur content, however, a piece of steel may be dirty from slag inclusions, or clean because slag inclusions are practically absent. One of the best expositions of the various sources of such slag inclusions in steel was given by L. B. Lindemuth in discussing a paper "Present Knowledge Concerning Non-metallic Impurities of Steel" by H. D. Hibbard, at the May meeting of the American Iron & Steel Institute in 1919.

Although the author mentions Mg-Mn-Si as a complex deoxidizer that has been "used with the object of forming an oxide mixture of low melting point and viscosity," titanium has been used successfully with manganese and silicon, to a far greater extent than magnesium.

The greater cleanness of acid open-hearth steel as compared to basic has not been noticed in our experience, which however has been largely confined to ordinary, low-grade steels. A distinction between effervescing and killed open-hearth steel would seem more logical than between acid and basic, for the ordinary grades made in the greatest quantity. Our experience with such steels would indicate that, in general, the effervescing steel is cleaner than the killed, although sulfide inclusions are apt to be larger and more segregated in the former.

In this paper, most attention seems to be paid to cleaning the steel in the furnace. It is important to remember that before the metal is used, it is poured out of the furnace through the air, into a ladle, where it is violently mixed with at least some slag. Furthermore, before solidification it is poured through a clay nozzle, which is more or less eroded, into a mold, where there is more splashing in the air. It seems questionable whether important results can be expected from applying the principles of colloid chemistry toward cleaning the steel in the furnace, unless much improved practice can be developed for getting it into the ladle and molds without contamination.

In order to make clean steel according to the author's definition, the first requisite is to make it free from sulfur. As manganese sulfide and iron sulfide are undoubtedly both soluble in liquid steel, this is distinctly a problem of metallurgical rather than colloid chemistry. If we could remove all the sulfur from steel, probably 75 per cent. or more of the non-metallic inclusions would disappear.

The next most important requisite, in the writer's opinion, is to find a material for the ladle lining, the stopper, and the nozzle that would be absolutely infusible and not eroded by the liquid steel. If these two conditions could be met, the inclusions arising from other sources could probably be kept down to negligible amounts by good metallurgical practice without involving the aid of colloid chemistry to a very great extent. For instance, with sulfur absent, manganese would not be required to prevent hot-shortness, and silicon could be removed in the basic furnace. The steel as tapped need then contain only iron, carbon, oxygen, and a little phosphorus, which is not known to form inclusions. A small addition of a scavenger, that leaves no oxidation product in the steel, would be sufficient in the ladle to flux off the slag particles mixed with the steel in tapping. Assuming no erosion of refractories, the steel could be poured into absolutely clean molds, and allowed to effervesce until solid. This effervescence amounts to deoxidation by carbon, and if enough carbon were present, all the oxide coming out of solution would be used up in this way, the oxidation product being a gas and the steel being left clean. This procedure would be applicable only to fairly low-carbon steels, which are made at present in large quantities by a process substantially as outlined. Of course such steels are more or less contaminated by manganese sulfide and silicate, as sulfur cannot be removed and refractories are not perfect.

To make clean killed steel, which is required when the carbon is high, or when segregation must be prevented, would be more of a problem even under the ideal conditions of absence of sulfur and perfect ladle lining, etc. In this case, probably the best practice would be to deoxidize in the acid furnace with carbon, manganese, and silicon, killing at that point, and tapping into the ladle with the least possible agitation. A cleaning in the ladle with a scavenger like ferrocarbon-titanium should then be effective in separating the slag from the metal if sufficient time were given. In the absence of fused refractories, which is assumed together with a zero sulfur content, the contamination caused by teeming into the molds might with care be kept inappreciable, so that the finished steel would be really clean.

H. W. Gillett (author's reply to discussion).—There is some evidence that at very high temperatures manganese sulfide may be, to a certain extent, soluble in liquid steel. It may be in true solution, or it may be in colloidal suspension and amenable to coagulation by some method. Even though it may be in true solution at some temperature, it may fall out of solution at a lower temperature while the steel is still liquid.

The experimental difficulties of quantitative work on such a problem are so great that conclusions from what little has been done are speculative. Even when sulfur is lowered to 0.006 per cent., as in Priestley's Fig. 2,[4] plenty of inclusions are left, even though the steel has been vastly improved.

The shape and distribution of the inclusions are of great importance. Manganese sulfide[5] draws out on forging and does not present marked internal notches across the path of force applied in the direction of forging, while it does present notches to a transverse test.

The writer did not care to bring up the controversial subject of the effect of titanium. It should be pointed out, however, that of all types of inclusions, the sharp-cornered ones shown in Bureau of Standards Circular 113, Fig. 17, p. 36, as characteristic of inclusions in steel treated with titanium, and Fig. 53, p. 80, as characteristic of those in steel treated with zirconium, are of a peculiarly dangerous type because they do not draw out, but form very sharp internal notches. Hence, if a steel is improperly treated with titanium or zirconium so that these sharp inclusions do not float up to the slag, the steel may be injured instead of benefited.

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[4] *Op. cit.*, 324.
[5] *Op. cit.*, 319.

The writer agrees that desulfurization by chemical means is desirable, and that cleanliness in pouring the steel is as essential as obtaining a clean steel in the furnace. But the fact that three or more main factors are essential in the production of clean steel is no reason why possible improvement in one of the factors should not be considered. This paper stressed but one factor, though the writer well knows that the other factors are important.

Low-temperature Brittleness in Silicon Steels

By Norman B. Pilling,* East Pittsburgh, Pa.

(New York Meeting, February, 1923)

Practical limitations to the usefulness of silicon steels are the hardness and brittleness silicon imparts to iron, making iron-silicon alloys of more than 8 per cent. silicon content unusable except where castings can be employed. In the use of commercial silicon steel, in thin rolled sheets, as in transformer construction, when the silicon content exceeds 4.2 per cent. the sheet is too brittle for satisfactory shaping by punching or similar operations. When such a brittle steel is sheared, it frequently breaks along a haphazard path in advance of the cutting tool.

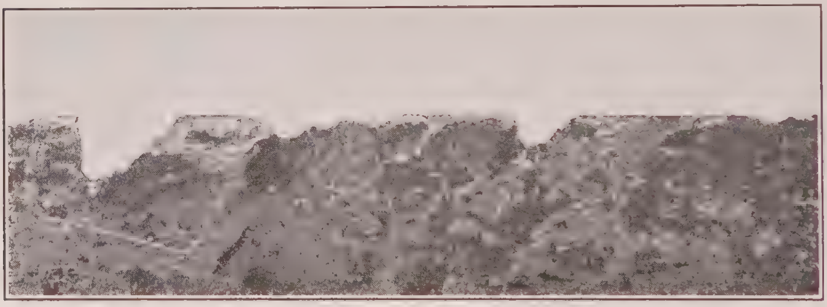

Fig. 1.—Sheared edge of a brittle silicon steel sheet. ×1.2.

In Fig. 1 is shown a slightly enlarged view of the sheared edge of a 14-mil enameled sheet of nominally 4 per cent. silicon steel. In 5 in. along the sheared edge, only 44 per cent. was actually cut; the rest was torn, with the path of fracture varying widely from the path of travel between the shears. This is probably an extreme case of such brittleness, but a shearing or punching test is not necessary to reveal brittleness of this order; narrow strips will break when bent between the fingers. In Fig. 2, the proportion of actual cut edge is greater, but secondary fractures branch off at about 30° from the line of shearing in advance of the cut. Deprived of the support of adjacent parts of the sheet, the tips of the angles are bent out of the plane of the sheet by the shears, leaving a peculiar flared effect, like the set of the teeth on a saw. An end-on view *A*, Fig. 3, shows this well; *B* is a similar end view of a sheet in which the quality of the sheared edge was rated as usable; the slight burr indicates the condition of the dies rather than of the steel.

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* Research Department, Westinghouse Elec. & Mfg. Co.

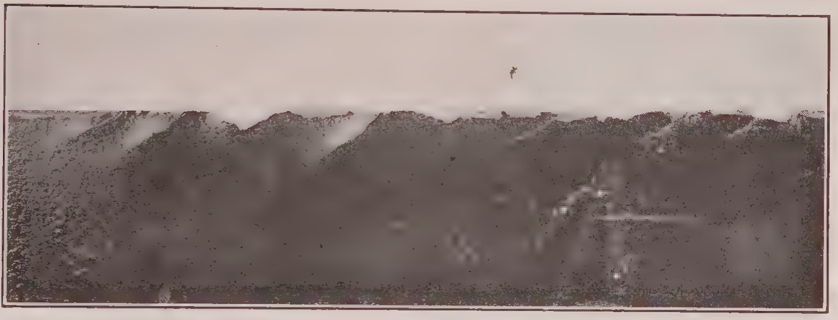

FIG. 2.—SHEARED EDGE OF A BRITTLE SILICON STEEL SHEET. × 1.2.

Several large sheets were taken from a large lot of steel that had been rejected as unusable after thorough trial by the metal-stamping department. These sheets vary somewhat in composition, as shown in Table 1, and differ from the average of this grade of material by the high silicon and carbon contents (silicon will average 4.0 per cent., carbon 0.03). All are so brittle that when an attempt was made to bend strips of them in the fingers, a crisp fracture resulted. Sheet number 1 was selected for an experimental investigation, the report of which follows; unless otherwise indicated reference is to this sheet.

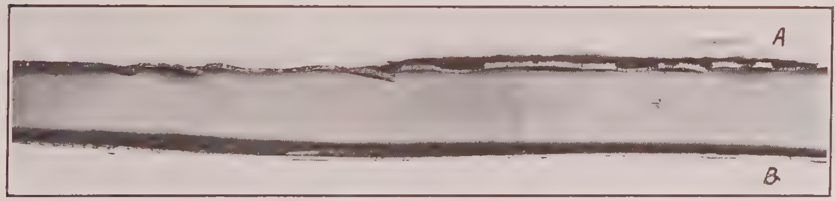

FIG. 3.—END VIEW, SHEARED EDGE OF SILICON STEEL SHEETS. *A*, BRITTLE; *B*, USABLE. × 1.2.

TABLE 1.—*Composition of Brittle Silicon-steel Sheets*

Laboratory Stock Number	Sheet	Composition		
		Silicon, Per Cent.	Manganese, Per Cent.	Carbon, Per Cent.
0217	1	4.69	0.22	0.075
0218	2	4.51	0.19	0.075
0219	3	4.71	0.22	0.087

Type of Fracture

In getting to the root of the cause of this brittleness, information as to the actual path that the fracture takes is of primary interest. A fractured edge is brightly crystalline, and in coarse-grained sheets seems to be composed of plane facets, which usually extend through the entire thickness of a 14-mil sheet. Portions of a sheet annealed at a high temperature in vacuum showed the crystalline pattern on the surface without acid etching. One of these "vacuum etched" sheets was bent with the fingers and promptly broke. The actual path of the fracture in relation to the crystalline structure could easily be followed with a hand magnifying glass; it is shown in Fig. 4 enlarged to twelve diameters. The crystalline structure is somewhat concealed by the surface roughness of the sheet, together with the fact that the surface at the fracture is curved. An outline sketch is given of the path of fracture, based both on this photomicrograph and visual examination under the microscope. The fracture is transcrystalline; that is, the grains do not tear apart but break in two. Although the fracture as a whole is irregular, it follows only one direction within individual grains.

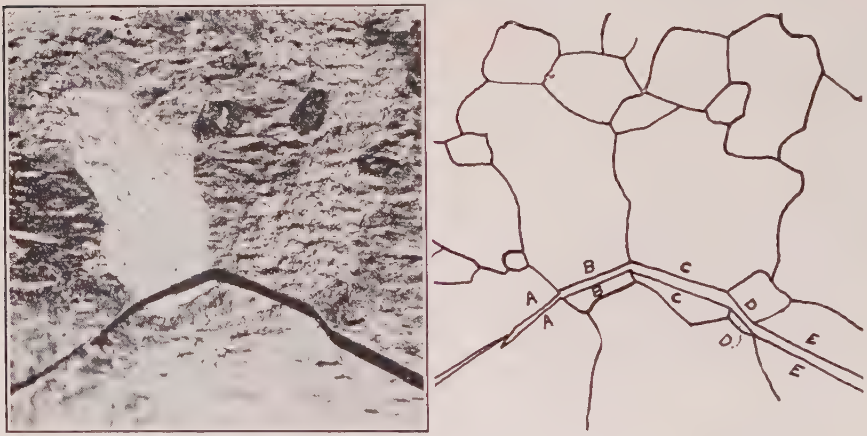

Fig. 4.—Path of fracture in 4.7-per cent. silicon steel. ×12.

These facts shed considerable light on the mechanism of failure. The actual fractured surface in one single grain is a cleavage plane of the metallic crystal, whence it is evident that it is the properties of the principal crystalline phase which are concerned. Commercial silicon steel contains silicon, manganese, sulfur, phosphorus, carbon, and iron; of these, the sulfur and carbon are present as segregated particles (sulfide and carbide respectively), while the silicon, manganese, and phosphorus dissolve in the iron forming a single crystalline substance (solid solution) which constitutes the principal metallic phase. The problem, then, concerns the behavior of this crystalline metallic solution when subjected to plastic deformation.

EFFECT OF RATE OF DEFORMATION

The operations of punching and shearing involve deformation at a high rate of speed, compared with bending by hand. Although poor shearing properties tie up with poor bending properties, it was desirable to determine whether or not the mode of failure depends on the speed of fracture. Strips were clamped in a vise and bent around a small radius with a steel plate by hand at different rates of angular motion and one strip was struck with a 0.22 calibre bullet from a target revolver.

TABLE 2.—*Effect of Rate of Bending*

Time to Bend 90° or Break, Seconds	Broken?	Type of Break	Angle of Bend Before Fracture
185	No		90°+
1	Yes	Transcrystalline	39°
0.0001	Yes	Transcrystalline	Very small

Two facts are evident from these trials at enormously different rates: (1) The type of fracture does not change when the rate of bending is greatly increased; it remains transcrystalline. (2) The ability to withstand bending depends on the rate of bending. The glasslike fragility of the sheet bent by a rapidly moving bullet and the successful right-angle bend when 3 min. were taken to complete it are extremes in ductility.

EFFECT OF TEMPERATURE

There are two ways in which temperature may modify the properties of a metal: (1) Changes brought about by thermal treatment (such as annealing) and persisting as permanent changes at other temperatures; (2) strictly reversible changes, characteristic simply of the temperature at which the property is measured. The former concerns permanent, the latter temporary, changes of the property concerned.

It does not seem that the brittleness of this 4.7-per cent. silicon steel is especially susceptible to permanent improvement by heat treatment. In order to obtain the proper magnetic and electric characteristics, this grade of steel is given a long annealing at about 800° C., followed by very slow cooling, and is the treatment this brittle metal has received. Modifying the rate of cooling to an extent sufficient to change the carbon present from carbide to temper carbon, or increasing the annealing temperature has no noticeable effect, other than that quenching seems to increase the brittleness.

TABLE 3.—*Effect of Heat Treatment*

Annealing Temperature, Degrees C.	Atmosphere	Cooling Rate	Bent at 25° C.
1200	Vacuum	Slow	Failed
800	Neutral	Very slow	Failed
800	Air	Fast	Failed
800	Air	Quench	Failed

Reannealing sheets from this same lot at seven temperatures between 625 and 815° C., according to the regular temperature cycle, had practically no effect.

The ductility at high temperatures must be far different from its cold ductility, in order to permit the fabrication of the metal to thin sheets. When bent at 750° C. in a Meker burner, a strip could easily be bent and the bend flattened with a hammer without the slightest indication of failure; therefore, somewhere between 25° and 750° this steel is no longer brittle, but becomes workably ductile. A rough estimate of this temperature was made by bending strips, hot, somewhat more than a right angle into such shape that the bend could be quickly crushed flat, cooling, bringing to the temperature range desired and then flattening. The surprising result is the lowness of the temperature to which the steel remains ductile. Raised to the temperature of boiling water, the brittle steel is workably ductile; cooled to the temperature of boiling liquid air (−190° C.) it is glasslike in fragility.

TABLE 4.—*Effect of Temperature*

Temperature, Degrees C.	Bending Ability
750	Very soft and ductile.
400–500	Ductile.
200–300	Ductile.
100	Stiff, but deformed without breaking.
20	Fractured after 50 per cent. bend.
−190	Very brittle; broke without bending.

The ability of silicon steel to deform plastically and exhibit ductility is profoundly affected by the temperature at which it is deformed; for this particular steel a slight rise in temperature yields sufficient temporary ductility to permit drastic deformation without failure.

EFFECT OF SILICON

It has been noted that brittleness is associated with an unusually high silicon content; it is therefore probable that a definite connection exists between composition and the temperature-ductility relation. A series of commercial silicon steels of graded silicon content was obtained in 14-mil sheets and the temperature of incipient ductility estimated from bending tests made at different temperatures.

TABLE 5.—*Composition of Steel Sheets*

Laboratory Stock Number	Grade	Composition		
		Silicon, Per Cent.	Manganese, Per Cent.	Phosphorus, Per Cent.
020	Pure, vacuum fused electrolytic iron.	0.011	0.01	Less than 0.005
0221	MA silicon steel.	0.86	0.24	
0222	3 AW silicon steel.	2.02	0.23	
0223	1 AW silicon steel.	3.84	0.15	0.043
0217	1 AW silicon steel.	4.69	0.22	0.030

A scale of temperatures was fixed by the melting and boiling points of a number of substances, small bits of which were placed in the angle of the perviously bent strips. When the desired temperature was indicated, the strip was quickly placed on a wooden block and the bend flattened with one blow of a hammer. Partial or complete cracking was considered failure by this test.

TABLE 6.—*Scale of Temperatures*

DEGREES C.		
−190	Liquid air	Boiling point
−130	Ethyl alcohol	Melting point
− 95	Acetone	Melting point
− 40	Mercury	Melting point
+ 25	Room temperature	
+ 65	Wood's metal	Freezing point
+100	Water	Boiling point
+150	Bismuth-cadmium eutectic	Freezing point

The results of this experiment are plotted in Fig. 5; examples are shown in Figs. 6 and 7. Even iron of the highest purity develops this brittleness when the temperature of deformation is sufficiently low (−130° C.)[1] and it appears that the brittleness of silicon steel is this property of iron made manifest at ordinary room temperatures by the effect silicon has of raising the temperature of incipient brittleness. The effect of the first 2 per cent. of silicon is slight; thereafter the raising of the critical temperature proceeds rapidly and reaches the ordinary indoor atmospheric temperature range (10–30° C.) at 3.9 per cent. silicon. At higher silicon contents, the temperature must be raised an amount proportioned to the silicon content to reach the range of workable ductility.

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[1] A recent French report places the temperature at which electrolytic iron shows a sudden increase in Brinell hardness at −110° C. L. Guillet and J. Cournot, *Rev. Met.* (1922) **9**, 215.

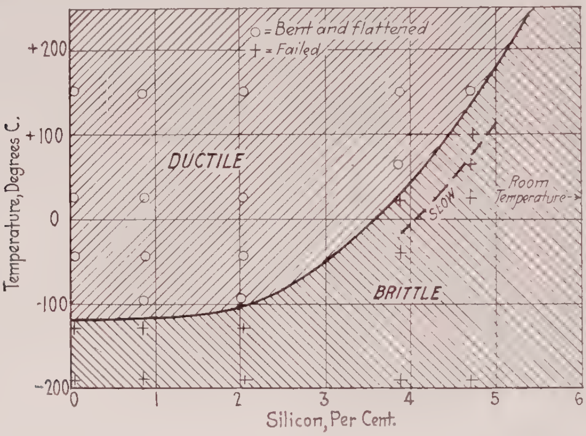
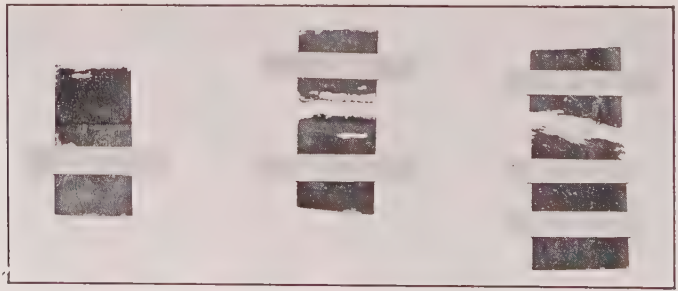


Fig. 5.—Effect of silicon on temperature of incipient ductility of silicon steel.

The degree of ductility necessary for successfully withstanding the rapid deformation of the flattened bend test is rather high and a steel of given composition that fails at a certain temperature may successfully pass a less severe test. The dotted curve gives an approximation of the temperature-composition limits for the appearance of brittleness at the other extreme; viz., bending slowly around a comparatively large radius, as in a rough finger bend. Thus, in the neighborhood of 4.0 per cent. silicon, a temperature range of nearly 50° C. and a silicon range of about



Bent at −95° C. −130° C. −190° C.

Fig. 6.—Ductility of 2 per cent. silicon steel at different temperatures.

0.4 per cent. form a zone in which brittleness may or may not be apparent, depending on the conditions of stressing.

For a given manner of testing, the actual temperature difference separating a workable from an unworkable degree of ductility is apparently quite small. The steel sheet shown in Fig. 2 was decidedly unsatisfactory, as shown by both bending and shearing experience. A strip from it was held in a running stream of water, the temperature of which could be regulated with considerable nicety, and bent with the fingers around a ⅛-in. radius.

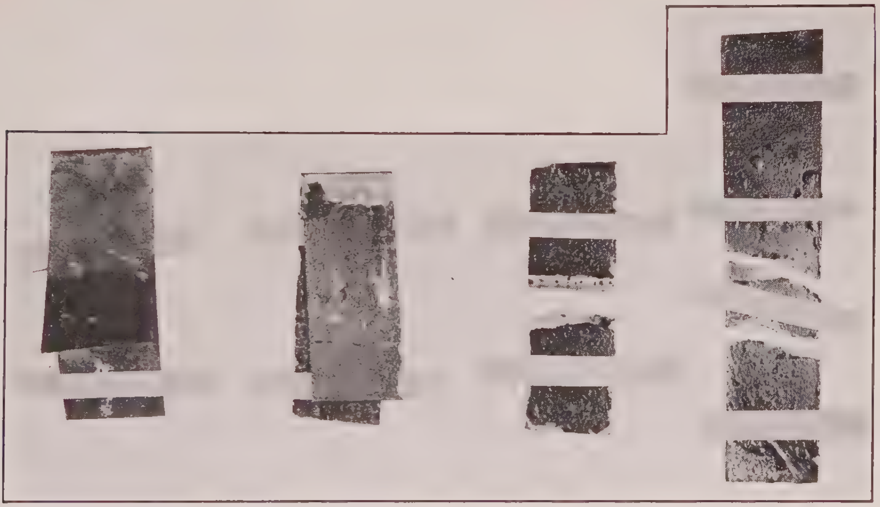

Bent at 750° C. 100° C. 20° C. −190° C.

FIG. 7.—DUCTILITY OF 4.7 PER CENT. SILICON STEEL AT DIFFERENT TEMPERATURES.

Temperature, Degrees C.	Attempt to Bend
37	Failed
43	Bent without cracking
37	Failed

A 6° difference in temperature was enough for definite differentiation between success and failure. The composition of this sheet, by analysis, was silicon 4.57 per cent., manganese, 0.20 per cent. This has its corollary in the conduction of ductility tests, such as the Erichsen, of silicon steel, especially around the critical silicon content 4.0 per cent. or above, in which the actual temperature of testing should receive a consideration not given with customary usage.

EFFECT OF MANGANESE

The pure iron sheet (manganese 0.01 per cent.) was distinctly more brittle at −130° C. than the 0.86-per cent. silicon steel with 0.24 per cent. manganese; this together with the tendency of the 3.84-per cent. silicon alloy, containing but 0.15 per cent. manganese, toward a higher critical ductility temperature than the curve value would indicate, points to manganese as having an effect opposite to that of silicon; that is, lowering the critical ductility temperature, or producing a greater workability at a given temperature, other conditions being equal. This is in agreement with the well-known effect of manganese on iron at much higher temperatures.

Effect of Phosphorus

A sufficient range of phosphorus content was not available to decide its effect. An iron-phosphorus alloy containing 0.14 per cent. phosphorus was ductile enough at room temperature to be peened with a hammer.

Effect of Aluminum

Although aluminum is not present in silicon steels in any quantity, metallurgically it is similar to silicon in that it forms solid solutions with iron up to high concentrations (32 per cent. aluminum). At 5 per cent. aluminum, the ductility begins to decrease markedly and the alloys become brittle. A 7 per cent. iron-aluminum alloy broke into fragments when hammered at room temperature; at 100° C. it fractured; at 150° C. it deformed successfully. It appears, then, that aluminum has an effect on the critical-ductility temperature of iron similar to that of silicon.

Effect of Carbon

Knowing that carbon, when present as precipitated carbide in steels of this type, tends to segregate as an intercrystalline layer, it would seem unlikely for it to have a direct effect on brittleness as the path of fracture does not follow, but crosses, the seat of carbon segregation. Replacing carbide by temper carbon appears to have no effect. Positive information as to the influence of carbon has not been obtained.

Effect of Grain Size

The fracture in silicon steel follows cleavage planes of the crystal; anything that serves to break the continuity of the path of rupture will aid in checking it. Hence, the frequent change in direction that the fracture is compelled to make in passing from one grain to another is an effective agent in limiting an incipient fracture. An example of this was noted when an attempt was made to cut, with a rather dull pair of tin snips, a strip of 4.0 per cent. silicon steel in which the grain size was non-uniform. The fine-grained end cut well, but it was impossible to cut across one large crystal about 1½ in. in diameter, as shearing persisted along a set of cleavage planes of the crystal obliquely to the intended direction of cutting. When sheet and shears were held under a running stream of warm water the cut was easily made.

SUMMARY

The brittleness of silicon steel is an inherent property of iron, modified by the alloyed silicon.

Commercial silicon steel becomes brittle when the silicon content exceeds 4.2 per cent.

Temporary ductility may be obtained by carrying on cutting or deformational operations at temperatures slightly above atmospheric, the temperature depending on the steel composition.

Brittleness is only slightly modified by heat treatment.

The purest iron shows a similar brittleness at about −130° C. The effect of several alloying elements is: Silicon raises critical ductility temperature; aluminium raises critical ductility temperature; and manganese lowers critical ductility temperature.

DISCUSSION

H. S. RAWDON, Washington, D. C.—I would not say that the effects are somewhat dependent on the grain size, but rather that they are vitally affected by it. At the Bureau of Standards, we were interested in the properties of iron in connection with some of the refrigeration investigations, and found that ordinary stovepipe iron wire, which is soft and very ductile at ordinary temperature and has relatively large crystals, when dipped into a jar of liquid air (the temperature of which is about −180° C.) will break like glass when it is taken out. The same wire heated to its critical point, when dipped in liquid air will stand almost as many bends as an untreated coarse wire would at ordinary temperature. Therefore, we concluded as a result of repeated tests of this kind that grain size is a vital factor in this transcrystalline brittleness that exists in iron at low temperatures.

This transcrystalline brittleness is a characteristic property of metals in general. There is some temperature below which a metal is normally brittle and above which it is ductile; that temperature varies with different metals. In metals such as iron, it is considerably below zero centigrade; in the high melting-point metals, such as tungsten, brittleness occurs about at room temperature.

S. L. HOYT, Cleveland, Ohio.—Having worked recently with tungsten, a metal that apparently behaves quite differently from ordinary metals, I could not but wonder what would be the effect of deformation on these high-silicon steels, that is the steels that are normally brittle at room temperature, if they were deformed at a slightly elevated temperature and then tested at room temperature.

If this silicon steel is brittle at room temperature, might it not be made ductile if it could be put into a fibrous condition; but putting it into a fibrous condition would necessitate deformation at a temperature below its normal annealing temperature.

Norman B. Pilling.—To my knowledge nothing has been done in that direction, perhaps as ductility is not the only requirement for steel of this type; it is a magnetic material and as such is quite sensitive to deformational treatment. Perhaps a treatment such as Doctor Hoyt suggests would make it ductile; the treatment would certainly ruin the steel for the purpose for which it is intended.

Effect of Silicon on Equilibrium Diagram of System Carbon-iron near Eutectoid Points

By H. A. Schwartz,* M. E., H. R. Payne,† A. F. Gorton,‡ Ph. D., Cleveland, Ohio

(Canadian Meeting, August, 1923)

In a previous paper[1] we published what we believed to be a correct stable equilibrium diagram for an iron-carbon alloy containing ±1.20 per cent. silicon. The purpose of the present paper is to record the results of studies concerning the effect of silicon on the location of the eutectoid points of both systems, including also certain incidental observations of general interest.

We prepared white cast irons containing roughly 2½ per cent. carbon, 0.05 per cent. manganese, and 0.03 per cent. each of phosphorus and sulfur, but of accurately known silicon contents which varied from 0.40 to 3.32 per cent. These were completely graphitized, and both A_1 points determined, as in our previous work, the specimens being converted from the stable to the metastable condition and vice versa by appropriate heat treatments.

Each stable alloy was heated to a few degrees above Ar_1 for 3 to 7 hr., quenched and analyzed for total and graphitic carbon, the difference being the "combined" or, better, agraphitic[2] carbon content of the saturated solid solution just above the eutectoid transformation. One specimen was lost through accident, and the 0.40-per cent. silicon material gave an anomalous result ascribed to difficulty in maintaining the stable system in the presence of but little silicon.

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* Manager of Research, The National Malleable Castings Co.

† Chief Chemist and Metallographer, Research Dept., The National Malleable Castings Co.

‡ Lately physicist, The National Malleable Castings Co.

[1] H. A. Schwartz, H. R. Payne, A. F. Gorton, M. M. Austin: Conditions of Stable Equilibrium in Iron-carbon Alloys. *Trans.* (1923) **68**, 916.

[2] *Op. cit.*, 918.

The data are recorded in Fig. 1. The shape of our A_1 curves is very similar to Scott's.[3] The curves are always somewhat higher, possibly because of differences in manganese content. The points at 1.20 per cent. silicon are taken from our previous publication and tie together the two investigations.

Gontermann,[4] Yensen,[5] Charpy and Cornu-Thenard[6] seem to agree that the effect of silicon on A_3 (metastable) is to diminish its intensity and raise its temperature. A_3 disappears at a silicon content variously estimated between 1.30 and 2.50 per cent. The latter noted an increase in temperature between 20° to 240° C. for 1 per cent. in silicon. It is assumed that the metastable eutectoid shifts to the left (*i.e.*, lower carbon) with increasing silicon content. Accepting as a fact that A_3 is always raised by silicon, the minimum possible metastable eutectoid carbon content consistent with raising A_3 for increasing silicon can be determined by noting the intersection of A_1 for a given silicon with A_3 of the silicon-free system. These values have been plotted in Fig. 1.

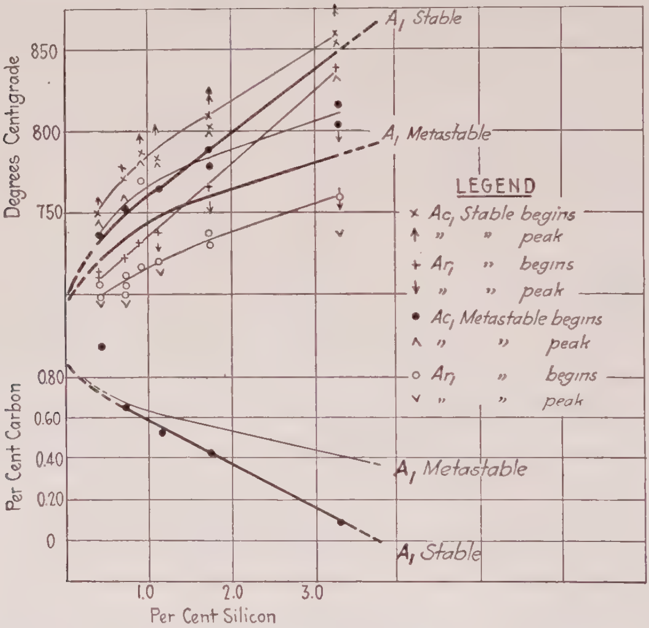

FIG. 1.—EFFECT OF SILICON ON STABLE EUTECTOID IN THE IRON-CARBON-SILICON ALLOYS.

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[3] Graphitization of White Cast Iron. *Trans.* (1922) **67,** 453.

[4] *Jnl.* Iron and Steel Institute (1911–I) 431.

[5] Univ. of Illinois Engineering Exper. Station *Bull.* 83, 16.

[6] *Jnl.* Iron and Steel Institute (1915–I) 303.

The A_3 transformation, being characteristic of hypoeutectoid alloys, cannot be determined in the stable system, which is not known to exist in such alloys. We can, however, draw the A_3 line in that series of ternary alloys in which the carbon and silicon content are related, as shown in the lowest curve of Fig. 1. These alloys are of eutectoid composition and the eutectoid point is the lower terminus of A_3. A_3 stable with varying silicon is thus plotted in Fig. 2 for these particular alloys. For comparison, the A_3 line of the silicon-free system is shown. As a check, we determined A_3 in a sample of Armco iron; the mean of the peaks representing Ar_3 and Ac_3 was 904° C., which agrees closely with the accepted 900° C. value. An approximation of the effect of the silicon present, using Charpy's lowest rate, is also plotted.

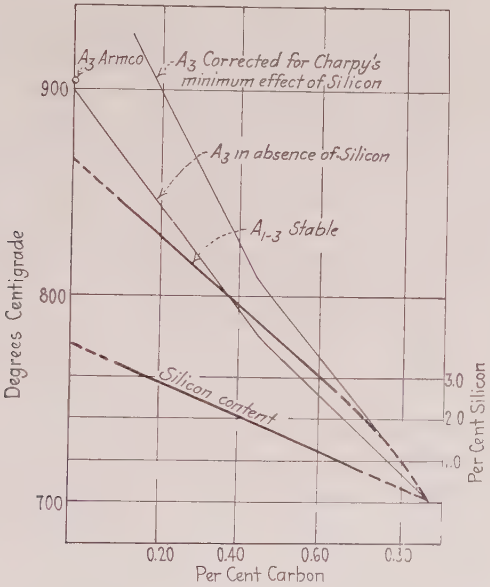

FIG. 2.—EQUILIBRIUM IN IRON-CARBON-SILICON ALLOYS OF STABLE EUTECTOID COMPOSITION.

Some authors, Archer,[7] Hayes, Diederichs and Dunlap,[8] postulating the identity of the solid solutions in the two systems, have drawn conclusions as to the limitations placed thereby on the relation between eutectoid composition and temperature. Our data do not support the postulated identity of the A_3 lines in the two systems. Fig. 1 strongly suggests that the two eutectoid points are identical in the absence of silicon.

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[7] Graphitization of White Cast Iron. *Trans.* (1922) **67**, 464.

[8] Am. Soc. for Steel Treating, *Trans.* (1923) 624.

An alloy containing 0.05 per cent. silicon, about 5.5 per cent. carbon, and a few hundredths of a per cent. of manganese, phosphorus, and sulfur, could not be graphitized below 0.90 to 1.00 per cent. agraphitic carbon, even by two malleable anneals followed by a 400-hr. heat treatment near 700° C., but remained pearlitic throughout.

The graphitization of pure cementite is now being studied. The sequence of metallographic changes in the immediate neighborhood of A_1 stable was studied in a white cast iron, containing 0.98 per cent. silicon, subjected to the heat treatment shown in Fig. 3. Specimens quenched at the intervals marked by arrows were analyzed and examined metallographically. Ferrite was determined by planimeter measurements.

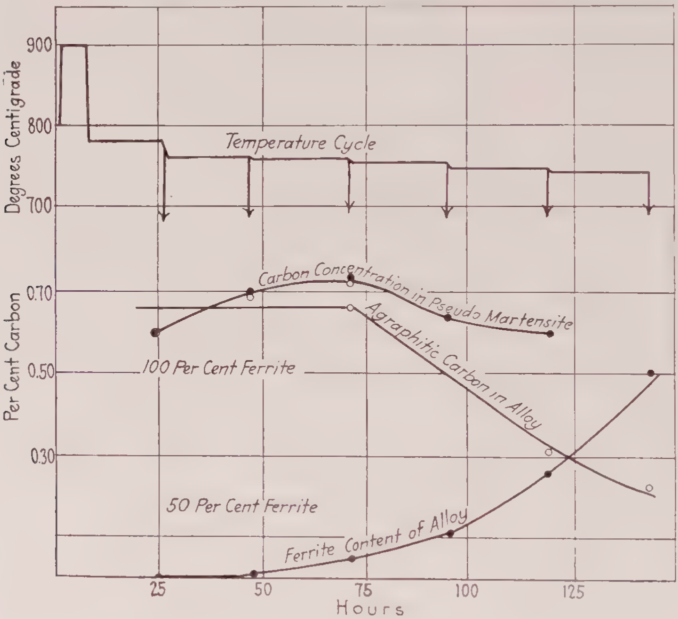

Fig. 3.—Chemical and metallographic changes while slowly traversing the A_1 range.

A specimen cooled in 5 min. from 770° C. to below A_1 consisted of fine pearlite and an excess of ferrite so divided that its amount could be only approximated at 10 to 20 per cent. Its agraphitic carbon content was 0.64 per cent., pointing to a metastable eutectoid of 0.70 to 0.80 per cent. carbon. Three points must be remembered: The record is of furnace temperature only; the time to the removal of the first specimen was kept down so that equilibrium was only approximated; in quenching the specimens, 1 cm. in diameter, there is at least a brief slower cooling before the metal, especially within, experiences the quenching arrest.

Martensitic structure was always present, though only in thin, widely distributed veins in the last specimen.

The appearance of ferrite at 765° C. seems to mark the crossing of A_3 (metastable), the abrupt decrease in agraphitic carbon at 760°, the passing of A_1 stable. At the intersection of A_3 and Agr, the alloy has a choice of equilibria down along Agr with separation of carbon to the stable eutectoid and down along A_3 with separation of iron to the metastable eutectoid. The increase in agraphitic carbon in the second and third specimens may be an expression of the tendency to revert to the metastable system and not merely an accidental error.

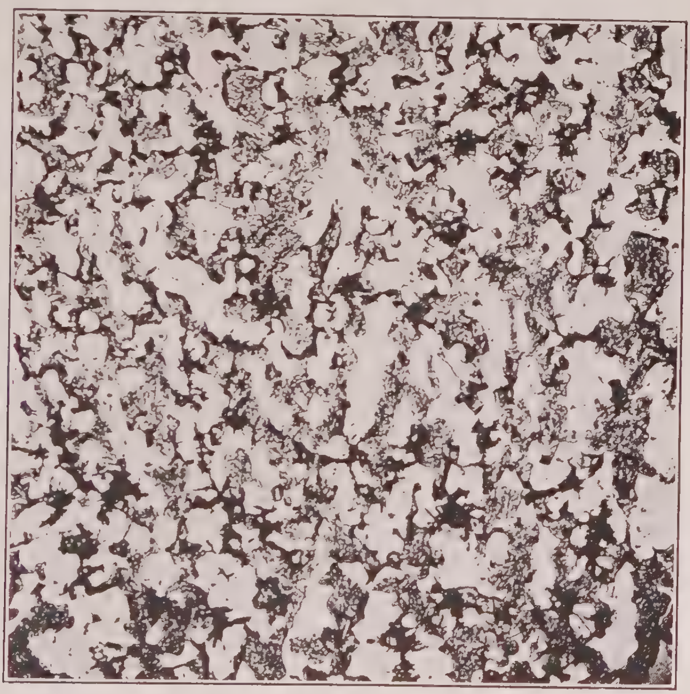

Fig. 4. × 200.

The ferrite distribution encountered is similar to that shown in Fig. 4, and does not resemble the frequently observed "bull's eye" structure.[9] The latter, explained by Hayes *et al.* (*loc. cit.*) by assuming a connecting neck of solid solution joining the graphite to the surrounding solid solution, is frequently, if not always, the result of graphitization below A_1, where no material solubility exists in either system.

The explanation would seem to be either that referred to by Archer,[10] or graphitization through the intervention of carbon monoxide, as suggested by Honda,[11] which, according to Matsubara's data,[12] can go on below 695° C. only, and is independent of pressure. Proof is lacking as to the solubility of carbon monoxide and carbon dioxide, as such, in ferrite.

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[9] Malleableizing of White Cast Iron. *Trans.* (1922) **67**, Fig. 10.

[10] Graphitization of White Cast Iron. *Trans.* (1922) **67**, 456.

In addition to such value as the preceding data may have in fixing the loci of certain transformations in the carbon-iron-silicon system, they point also to a further distinction between the solid solutions of the two systems, in that the alpha-gamma transformation of the stable solid solution of given composition takes place at lower temperatures than in the metastable system. Conversely, at a given temperature and silicon concentration, ferrite is more soluble in boydenite than in austenite. We have previously suggested "boydenite" as the name of the solid solution in the stable system, corresponding to austenite in the metastable. The present data point to the presence of silicon as a requirement in boydenite, silicon-free austenite and boydenite being perhaps identical.

More direct evidence was sought by x-ray spectrometer investigations of a manganiferous (15 per cent.) and nickel-bearing (20 per cent.) cast iron as free as practicable from all elements save iron, carbon (± 3 per cent.), and the alloying metal. The former metal consisted of cementite and solid solution; the latter of graphite and solid solution containing 0.60 per cent. agraphitic carbon.

E. C. Bain[13] reports that the metals, in addition to the space lattices of cementite and graphite, respectively, showed in the one case the pattern of gamma iron; in the other that of nickel, which imposes its lattice on alloys of iron and nickel in the ratio here used. The lattice of the solvent metal, gamma iron, has thus not been measurably affected by the considerable amounts of dissolved carbon. The spectrometer is blind to the presence of this carbon and the results are negative.

Austin[14] and Hird,[15] with the senior author, have observed certain sharp discontinuities in the rate of deposition of graphite at constant temperature, which suggest a distinct change in the graphitizing medium as the *Acm* concentration is passed. The details are reserved for publication at another time.

DISCUSSION

Ancel St. John, Long Island City, N. Y.—With respect to the possibility of two solid solutions of only two constituents, that is, the simultaneous existence of two solid solution phases, I am becoming more and more persuaded that that is not impossible even though it may seem highly improbable. There may be certain compositions that might almost be looked upon as compounds although they still are solid-solution types having two preferred equilibrium conditions. In some work I have been doing recently the only way in which I have been able to account for structures disclosed by *x*-ray analysis is to assume that there are at least two carbides present in what we believe to be simply a metal and carbon combination.

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[11] *Jnl.* Iron and Steel Institute (1920–II).

[12] Chemical Equilibrium between Iron, Carbon, and Oxygen. *Trans.* (1922) **67,** Fig. 5.

[13] Research Metallurgist, The Atlas Steel Corpn., Dunkirk, N. Y.

[14] Chemist, lately of The National Malleable Castings Co., Research Dept.

[15] Chemist, The National Malleable Castings Co., Research Dept.

ALEXANDER L. FEILD, Long Island City, N. Y.—The existence of two solid solutions in stable equilibrium with each other and containing a common constituent is not contrary to the phase rule any more than the existence of two immiscible liquid solutions is contrary to that rule. It might be possible to show whether there are really two different solid solutions by determining whether they act as separate phases.

ANCEL ST. JOHN.—When we are dealing with the behavior of atoms with respect to each other, it is well to divest our minds entirely of time-worn restrictions. Atoms do not behave the way we used to think of them. E. C. Bain, in his work on the iron-nickel series, found that from zero iron to 25 per cent. iron nothing is apparent except a solid solution of nickel in iron; from 35 per cent. to 100 per cent. nickel, there is nothing apparent but the solid solution of iron in nickel; in the range between 25 per cent. and 35 per cent. nickel, there is a mixture of both solid solution of nickel in iron and solid solution of iron in nickel. This is a transition range of just the character under consideration.

In the brasses, there are seven transitions of a similar character, but within much more restricted ranges.

Nitrogen in Steel

By C. Baldwin Sawyer,* Ph.D., Cleveland, Ohio

(Canadian Meeting, August, 1923)

During the last half century, much time has been devoted to investigations of the effect of nitrogen on the physical properties of steel, but in all discussions of results there is considerable doubt as to the accuracy and interpretation of analyses. No conclusions concerning the effects of nitrogen on the physical properties of steel can be more reliable than the methods used for determining the quantities of nitrogen present. The work described in this paper endeavors to establish some additional conditions required by the customary distillation method for reliable and accurate determinations. Results by a combustion method of analysis for total nitrogen content are presented; these indicate that all of the nitrogen in steel, exclusive of that contained in blowholes, may be determined by the distillation method.

Based on these more sound and more reliable analyses, results have been obtained that bear on the following divisions of the general subject, nitrogen in steel: Nitrification of steel by melting in nitrogen, nitrification of steel by heating in ammonia, decomposition of nitrified steel by heating, and thermal analysis of nitrified steel.

CONSIDERATION OF TWO METHODS OF ANALYSIS AND SOME RESULTS

The details of the distillation and combustion methods are chemical in character and of little interest to most readers. They are therefore described in an appendix. The distillation method is essentially that described by Hurum and Fay[1] and was generally used throughout this work. It is capable of the greatest accuracy and all results given are significant to the last figure. Where analyses do not check, the samples

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* The Brush Laboratories Co.

[1] *Chem. and Met. Eng.* (1922) **26**, 218.

probably differed. Hurum, Fowler,[2] Beilby and Henderson,[3] Braune,[4] and Tschischewski[5] have found that the distillation method gives results on samples of iron, nitrified by ammonia, corresponding to the observed gain in weight.

Professor Fay[6] has described an experiment designed to test the distillation method for recovery of nitrogen contained in cyanide treated steel; 97.6 per cent. of the gain in weight accompanying the cyanide treatment was accounted for by results of analyses before and after treatment. This seems to establish the fact that all nitrogen introduced into steel by treatment in ammonia or cyanide, generally called nitride nitrogen, is accounted for by the distillation method. But it does not preclude the possibility that nitrogen introduced in some other way may exist in a form that escapes determination. It therefore seemed desirable to start the development of a method capable of determining total nitrogen, combined or uncombined.

The ordinary combustion method for total carbon is adaptable to total nitrogen determinations. Electrolytic oxygen may enter the combustion tube and, combining with the iron, liberate its nitrogen in molecular form, which can then be swept out and estimated by gas analysis. Gaseous products of the combustion never contained nitrogen oxides, and all nitrogen in the steel must be liberated uncombined.

Although the electrolytic oxygen was generated in the laboratory, it contained about 0.20 per cent. of a residual gas, probably nitrogen, which necessitated a correction proportional to the total volume of oxygen used. This correction amounts to about 2 cc. per analysis, and is computed as explained in the appendix. As the volume of nitrogen contained in a 2-gm. sample of steel of 0.03-per cent. nitrogen is only 0.48 cc., it is not surprising that results obtained on such steels vary greatly from the average. For greater accuracy, the residual impurities contained in the oxygen must be eliminated or greatly reduced. Other improvements were desirable, but the necessary apparatus was not available, and therefore only the few analyses given in Table 1 were performed. In the first group of results, the analyses were made in the usual way, but with no sample in the boat. Results are calculated in per cent. as though for a 2-gm. sample and show the variations to be expected from causes not associated with the sample.

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[2] *Jnl.* Am. Chem. Soc. (1901) **79,** 291.

[3] *Jnl.* Am. Chem. Soc. (1901) **79,** 1245.

[4] *Bull. de L'Industrie Minerale* (1907) **7,** 489.

[5] *Jnl.* Iron and Steel Inst. (1915) **92,** II, 47.

[6] *Chem. and Met. Eng.* (1921) **24,** 289.

TABLE 1.—*Analyses for Nitrogen by Combustion Method*

Description of Sample	Weight of Sample Grams	Volume of Gas Obtained Cubic Centimeters	Correction for Blank, Cubic Centimeters		Per Cent. Nitrogen by by Combustion		Per Cent. Nitrogen by Distillation
			10 Min.	Total		Average	
Blank—alundum only in boat.		1.89	0.45	1.80	0.006	−0.002	
		1.14	0.36	1.08	0.004		
		1.25	0.37	1.48	−0.015		
Armco iron melted in vacuum.	2.05	2.10	0.38	2.50	−0.024	−0.011	0.00010
	2.02	2.10	0.30	1.95	0.009		
	2.03	2.12	0.37	2.40	−0.017		
Armco iron as received.	1.98	1.80	0.27	1.80	0.000	0.000	0.0048
Armco iron melted in 3 atm. absolute nitrogen.	2.00	3.00	0.30	2.10	0.056	0.039	0.0326
	2.01	2.39	0.33	2.14	0.016		
	2.05	2.31	0.28	1.82	0.030		
	2.04	2.39	0.23	1.50	0.055		
Thin sheet iron treated in cyanide.	1.069	6.58	0.15	0.75	0.682	0.738	0.726
	0.983	7.00	0.11	0.66	0.805		
	1.053	7.53	0.17	1.42	0.727		

The second group contains results of analyses on pure-iron ingots freed from nitrogen by melting in a vacuum. Variations in results of analyses on 2-gm. samples of steel containing no nitrogen may equal numerically 0.025 per cent.

Material for samples of the third group was prepared by melting pure iron in nitrogen at a pressure of three atmospheres absolute. Such iron, nitrified with molecular nitrogen, would be more likely to contain nitrogen molecularly occluded and in a form not determined by the distillation method, than would iron nitrified by means of any nitrogen compound. Results by combustion do not vary more than would be expected from their average, which agrees well with results by distillation.

The fourth group of analyses was performed on the same cyanide-treated sheet iron that had been prepared for the gain-in-weight experiment. Results by combustion agree well with those by distillation, and variations are not excessive for a 1-gm. sample. Everything in the gain-in-weight experiments and in the total nitrogen analyses indicated that the distillation method was reliable, so this method was adopted for the remainder of the work.

Unfortunately the work by H. E. Wheeler[7] did not appear before the combustion apparatus had been torn down. It would have been extremely interesting to have analyzed some of his steels which were so nitrified that no nitrogen was detected by the distillation method.

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[7] *Trans.* (1922) **67**, 257.

Ingots Melted under Nitrogen

For the work on total nitrogen analysis, the General Electric Co. supplied two small ingots of iron melted under pressure of nitrogen, as requested by Professor Fay. These ingots contained about 0.03 per cent. of nitrogen, and the effect seemed well worth further investigation. A number of ingots were therefore prepared by melting under nitrogen gas in an Arsem furnace. Pressures varied from a vacuum to a maximum of three atmospheres. The nitrogen was obtained from a tank and contained about 1 per cent. of oxygen. Alundum extraction thimbles, 13 mm. in diameter and 68 mm. in length, served for crucibles; for additional strength at higher temperatures, they were backed by a thin, closely fitting shell of graphite. Very little carbon entered the steel. All ingots were made from Armco iron supplied by the Page Steel and Wire Co. except one ingot of tool steel, containing about 1 per cent. carbon.

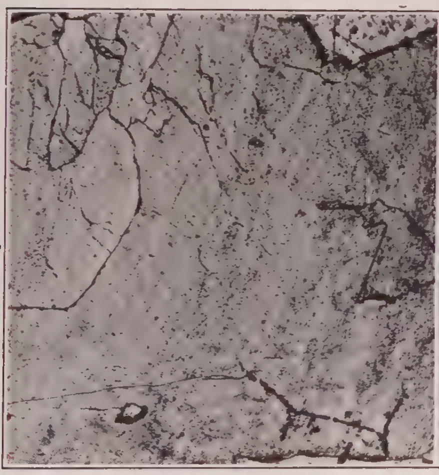

Fig. 1.—Ingot No. 11, melted in vacuum; nitrogen, 0.0001 per cent., no nitride needles. × 100.

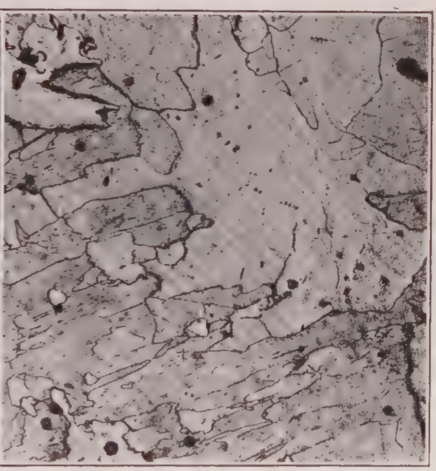

Fig. 2.—Ingot No. 2, melted by General Electric Co. under one atmosphere of nitrogen; nitrogen, 0.0332 per cent.; no needles. × 100.

The ingots obtained may be classified as those melted in vacuum and those melted under pressure. Those melted in vacuum were dull and had slight depressions in their tops. But with the single exception of ingot No. 2, all ingots melted under pressure were bright and silvery, with bulged tops covering deep pipes. The pressure of nitrogen, rejected from the liquid metal during its solidification, must be responsible for the pipes and bulges. Steel melted in vacuum always spattered at the moment of liquefaction and remained quiet during solidification, when all nitrogen had been removed. The reverse was true with steel melted under pressure of nitrogen, and its spattering during solidification must be due to the lesser solubility of nitrogen in solid than in liquid steel. In fact, the formation of bulges was visible through the mica window in the top of the furnace. A moment after the appearance of a crust, the top of the ingot became unsteady and wavering, as rejection of nitrogen and slow formation of pipe and bulge proceeded.

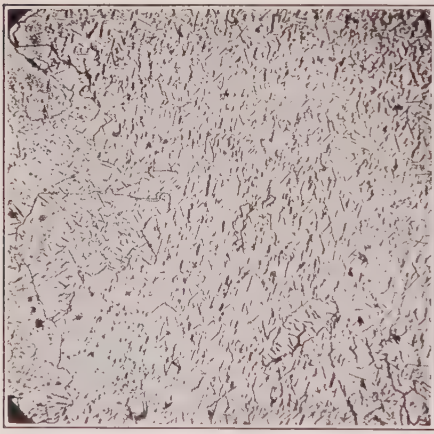

FIG. 3.—INGOT NO. 1, MELTED BY GENERAL ELECTRIC CO. UNDER TWO ATMOSPHERES OF N_2, NITROGEN, 0.0326 PER CENT.; MANY NITRIDE NEEDLES. × 100.

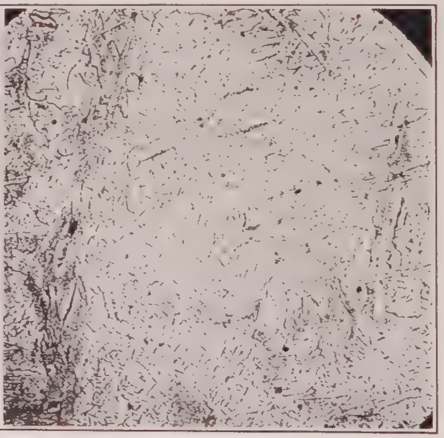

FIG. 4.—INGOT NO. 7, MELTED UNDER THREE ATMOSPHERES OF N_2, NITROGEN, 0.0327 PER CENT.; MANY NEEDLES. × 100.

Results of analyses are shown in Table 2. Included with them are results on two ingots prepared elsewhere and designated HU. Samples for analyses were turnings from the whole cross-section. The table shows that nitrogen contents increase with increase in pressure. Ingots melted in a vacuum have a nitrogen content so low that it may be taken as zero; those melted under one atmosphere contain from 0.013–0.017 per cent. nitrogen, though ingot No. 2, prepared by the General Electric Co., is an exception. Ingots melted under two atmospheres have a still higher content, suggesting a gain of 0.015 per cent. for each atmosphere; but ingots melted under three atmospheres show a gain of only about 0.005 per cent. Further information concerning the behavior of steel melted under nitrogen may be had from the work of J. H. Andrew,[8] who found that a low-carbon steel, melted and cooled under a pressure of 200 atmospheres of nitrogen, contained 0.3 per cent. of nitrogen.

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[8] Carnegie Scholarship *Memoirs* (1911) **3**, 236.

Table 2.—*Nitrogen Contents of Ingots Melted under Pressure of Nitrogen*

Ingot Number	Treatment	Nitrogen Content, Per Cent.	Remarks
HU	Vacuum	0.00103	
		0.00089	
10	Melted under 7 cm. mercury	0.00011	No needles
11	Melted seven times under 1 cm. mercury	0.00011	No needles, Fig. 1
		0.00008	
HU	1 atmosphere	0.0138	
2	General Electric 1 atmosphere	0.0352	No needles, Fig. 2
		0.0276	
		0.0348	
		0.0342	
5	Melted under 1 atmosphere	0.0158	No needles
1	General Electric 2 atmospheres	0.0322	Many needles, Fig. 3
		0.0330	
3_1	Melted under 2 atmospheres	0.0249	Few needles outside but many inside
		0.0305	
3_2	Part of ingot 3_1 held at 1200° C. for 2 hr. under 2 atmospheres.	0.0310	
		0.0296	
4	Tool steel melted under 2 atmospheres	0.0356	Very fine pearlite
		0.0356	
6	Melted under 3 atmospheres	0.0362	Many needles
		0.0347	
7	Melted under 3 atmospheres	0.0318	Many needles
		0.0337	
8	Melted under 3 atmospheres with 10 g. $Hg(CN)_2$ in crucible.	0.0362 (0.06 % C)	Many needles

The concentration of nitrogen in liquid iron should bear a relation to the partial pressure of nitrogen above the liquid, and the concentration of nitrogen dissolved in solid iron should bear a similar relation to the concentration of nitrogen in the mother melt, provided cooling from the temperature of solidification does not alter the concentration of nitrogen in the ingot. That it does not, is indicated by part of ingot No. 3 held at 1200° C. for 2 hr. under two atmospheres of nitrogen without significant change in concentration. Then, the two relations may be combined into one, with the assumption that the temperature of solidification is the same for all ingots. If nitrogen is monatomic in solid steel and diatomic as gas, this relation should be:

$$\%N = k\sqrt{P_N} \quad \text{or} \quad \frac{\%N}{\sqrt{P}} = k$$

where P_N = pressure of nitrogen in atmospheres;

$\%N$ = per cent. nitrogen in solid steel;

k = solubility coefficient.

Results in Table 2 are most nearly uniform for ingots melted under three atmospheres, which have an average nitrogen content of 0.034 per cent. Substitution of these data, and also those of Andrew, in the equation, gives two independently derived values for k which should be equal, if the equation holds.

$$\frac{0.3}{\sqrt{200}} = 0.021 = k; \qquad \frac{0.034}{\sqrt{3}} = 0.020 = k.$$

Assuming, on the other hand, the relation: $\% N = kP_N$ as for nitrogen diatomically dissolved in solid iron, the equations are:

$$\frac{0.3}{200} = 0.0015 = k; \qquad \frac{0.034}{3} = 0.012 = k.$$

when the two values of k are far apart.

Substituting $k = 0.021$ in the first equation gives the results in Table 3. Because of its exceptional behavior, the results of ingot No. 2 are not averaged with those of other ingots melted under one atmosphere. The fair agreement of observed and calculated values in Table 3 indicates that the square-root formula is applicable, and, therefore, that nitrogen exists monatomically when dissolved in solid iron near its melting point.

TABLE 3.—*Concentrations of Nitrogen Observed and Calculated*

P_N	Per Cent. Nitrogen Calculated	Per Cent. Nitrogen Observed
0	0.0	0.0001
1	0.021	0.015
2	0.030	0.030
3	0.036	0.034
200	0.3	0.3

Liquid bessemer steel, at the finish of a blow, should have a nitrogen content corresponding to the partial pressure of nitrogen in the converter, which is not less than three-fourths atmosphere. The fact that finished bessemer steels are much lower in nitrogen may be associated with the formation of gaseous oxides during deoxidation. These oxides bubble through the liquid steel and should carry off nitrogen in a fashion similar to distillation by air; the presence of oxygen in liquid steel, therefore, may be an aid to denitrification. This would seem to be promoted by adding the deoxidizer in successive small quantities instead of all at once.

The value of k may not be the same for different alloys of iron. Tschischewski[9] found that pure iron nitride, produced from iron reduced by hydrogen, retained no nitrogen after melting. But iron nitrides containing manganese, silicon, or aluminum retained nitrogen in increasing amounts in the order named; such substances may increase the value of k.

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[9] *Op. cit.*

Certain factors tend to increase the nitrogen content over that expected from gas-pressure calculations and may account for variable results obtained both experimentally and commercially. First, and of importance in commercial steelmaking, a column of liquid iron 49 in. high should produce a ferrostatic pressure equal to one atmosphere; this would oppose the rejection of nitrogen during solidification and tend to raise the content. Second, the ability of nitrogen to exist in a supersaturated solution of liquid iron might produce the same effect. The violent spattering during solidification is an evidence of supersaturation, which, if it be due to surface tension, should be large in iron, as the surface tension of liquid iron is 980 dynes per centimeter, while that of water at 0° C. is only 76 dynes per centimeter. Third, the formation of a crust over and around the liquid iron should prevent escape of rejected nitrogen, raising its pressure and increasing the nitrogen content toward the inner parts of ingots. Analyses of turnings from separate concentric layers of bulged ingots confirmed this last, showing much greater nitrogen contents in the inner portions. Table 4 contains the results of these analyses. No simple way suggests itself of accounting for the lower nitrogen content of the outermost layer from ingot No. 7.

TABLE 4.—*Nitrogen Contents of Concentric Layers of Ingots*

Ingot Number	Treatment	Nitrogen Content, Per Cent.	Remarks
3	Melted under 2 atmospheres, turnings from	outside 0.0308	Few needles
		inside 0.0362	Many needles
3	Melted under 2 atmospheres, turnings from	outside 0.0308	Few needles
		middle 0.0338	
		inside 0.0389	Many needles
7	Melted under 3 atmospheres, turnings from	outside 0.0251–0.0284	Few needles
		middle 0.0356	
		inside 0.0404	Many needles

Microscopical examination of ingots melted under two or three atmospheres shows that the density of nitrogen needles increases with increasing nitrogen content toward the center. But though nitrogen needles may be quite thick at the center of ingots, they are sparsely distributed or not at all present in outer portions; ingot No. 3 is an example of this condition.

The outer rings of turnings from ingots Nos. 3 and 7 showed very few nitrogen needles, though they contain from 0.0251 to 0.0308 per cent. nitrogen. Strauss[10] mentions nitrogen dissolved in ferrite up to 0.028 per cent. and Tschischewski states that carbonless samples of steel with less than 0.02 per cent. of nitrogen appear under the microscope as pure ferrite. Ingot No. 2 showed no nitrogen needles though its nitrogen content was about 0.034 per cent. All ingots melted under less than two atmospheres of nitrogen appeared as pure ferrite. Nitrogen appears, then, to form a solid solution with ferrite up to approximately 0.030 per cent. of nitrogen.

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[10] *Stahl und Eisen* (1914) **34**, 1814.

In Fig. 5, a piece of boiler plate showing Stead's brittleness, there is a structure much resembling coarse nitride needles; but the low nitrogen content, 0.0038 per cent., and the failure of the needles to become blue on heat tinting (as later explained) indicated that this needle structure was not nitrogenous.

The opening of the furnace after a run was always accompanied by a smell of cyanide, though blowing the furnace gases through a sodium-hydroxide solution yielded insufficient cyanide to give a positive test. However, the outside layers of a stick of sodium hydroxide, placed in the bottom of the furnace so that the furnace gases might sweep over it during the run, became friable and a water solution containing these layers gave a decided test for cyanide. Therefore, to try the effect of cyanogen in the furnace gases, 10 gm. of mercuric cyanide were placed on top of a crucible charge and heated with the iron. Mercuric cyanide decomposes, on heating, into mercury and cyanogen, but its presence did not definitely increase the nitrogen content of the ingot (ingot No. 8 in Table 2). Strauss states that iron has the same nitrogen content whether melted in an atmosphere of nitrogen or in one of ammonia.

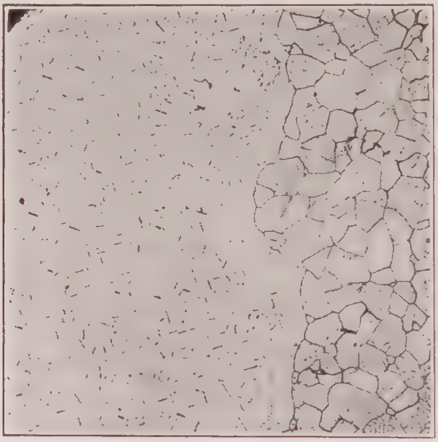

Fig. 5.—Boiler plate showing Stead's brittleness; nitrogen, 0.0038 per cent. needle structure not nitrogenous. × 100.

The nitrification of steel by melting under a pressure of nitrogen may be a means by which nitrified ingots can be prepared of sufficient size for fabrication into bars and test pieces. With this end in view, a large Arsem furnace, adapted for high pressures, is being constructed, and it is hoped soon to present results of tension tests.

Nitrification in a Stream of Ammonia

In all experimental work on nitrogen in steel, it has been difficult to obtain specimens uniformly nitrified. The pressures required for higher nitrogen contents make the method of melting under nitrogen impractical. The method of diffusion of nitrogen into solid steel by treatment at red heat in a stream of ammonia is tedious, but possible. Tschischewski[11] treated iron filings in this way at various temperatures; the temperature-concentration curve he obtained shows 11 per cent. nitrification at 400° C., and that nitrification decreases with increase of temperature, until at 900° C. the nitrifying action is very slight. It should be possible, therefore, to nitrify iron uniformly to any desired degree, by holding the iron at the proper temperature for sufficient time.

The apparatus used to investigate this possibility consisted of an ammonia tank connected through a bubbler to the combustion furnace previously described. For 3 in. at the center of the furnace, temperatures varied less than 10° C., as determined by exploration. Material used for nitrification was pure iron wire 0.226 mm. in diameter. It was cleaned with emery paper and made into loose bundles weighing about 1 gm. and about 2 in. long. These bundles, weighed and laid on granular alundum in a boat, could be easily put into the furnace. A platinum-platinum rhodium thermocouple gave temperature measurements.

At first, the bundles were put in a cold furnace, heated, and then cooled with ammonia flowing, under the assumption that both heating and cooling were quick enough to exclude noticeable nitrification at temperatures below that intended for treatment. All bundles so nitrified were colored, though varying in brilliance from one another and not always of so many shades; beginning with the ammonia end, the colors were: Gray, violet, yellow, green, blue, purple, brown. The bundles were brittle at the ammonia end, though often ductile at the exit end.

The results given in Table 5 for a bundle twice treated in the same way show that one treatment gave equilibrium values. Additional analyses gave 4.72 per cent. as the nitrogen content of the brittle end (toward the ammonia inlet) and 0.44 per cent. as that of the ductile end

Table 5.—*Wire Treated at Red Heat in Ammonia without Quenching*

NH_3 per Hour, Liters	Time of Treatment, Hour	Temperature, Degrees C.	Nitrogen by Gain in Weight, Per Cent.	Nitrogen, by Nitride Analysis, Per Cent.
11	1	900	2.33	
11	1	900	no change	2.10

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[11] *Op. cit.*

(toward the exit). Iron must catalyze the dissociation of ammonia, producing hydrogen with denitrifying action. To verify this, a bundle of wire, nitrified by similar treatment was reversed and a run made with the ductile end toward the incoming ammonia. On removal from the furnace, the previously highly nitrified portion had become ductile and the ductile part brittle.

Bundles nitrified by a mixture of hydrogen and ammonia had a decreased nitrogen content, as measured by gain in weight, but even with ammonia diluted to one part in eight the bundles were brittle at the ammonia end and ductile at the other. Further experiments showed that nitrification of the bundles during the heating and cooling in the furnace was by no means negligible; thereafter, in obtaining equilibrium values, all bundles were quenched from the temperature of treatment.

Uniform contact of ammonia with every part of the wire is not easy to obtain when the bundle is long and narrow. Changing the shape of the bundle to an open network, like a small cart wheel, and placing the cart wheel directly across the combustion tube, afforded better contact. Experiments made with wire 0.109 mm. in diameter gave variable results, probably because of oxidation during quenching. A series of experiments with cart-wheel bundles of wire 0.226 mm. in diameter gave the results in Table 6. Such bundles weighed about 0.32 g. and two of them were placed in the furnace together, in such manner that ammonia passed over them successively. All bundles were quenched and the results are grouped in pairs corresponding to bundles simultaneously treated.

Table 6.—*Wire Treated in Ammonia at Red Heat and Quenched*

Sample, Number	Liters NH_3 per Hour	Time of Exposure, Hour	Temperature, Degrees C.	Per Cent. Nitrogen by Gain in Weight	Per Cent. Nitrogen by Analysis
$BW_1 1$	12	1	900	1.16	0.965
$BW_1 2$				0.89	0.598
$BW_1 3$	12	1	900	0.92	0.643
$BW_1 4$				0.59	0.466
$BW_1 5$	24	1	900	0.847	0.591
$BW_1 6$				0.730	0.468
$BW_1 7$	12	1	850	1.04	0.85
$BW_1 8$				0.68	0.58
$BW_1 9$	12	½	850	1.24	1.06
$BW_1 10$				1.16	0.93
$BW_1 11$	12	1	800	1.76	1.54
$BW_1 12$				1.33	1.26

Throughout the table, the percentage of nitrogen by gain in weight exceeds that by analysis by about 0.2 units, irrespective of the actual nitrogen content. This difference is probably caused by the formation of surface oxide at the moment of quenching. It should be approximately constant throughout, for conditions affecting oxidation were not appreciably changed. It should be remembered that finer wire could not be used, presumably because of excessive oxidation.

In each pair, the second bundle has a lower nitrogen content than the first; this must be caused by decomposition of ammonia by the first. Such decomposition practically excludes the possibility of preparing uniformly nitrified steel by continually passing ammonia over it, no matter for how long. In agreement with Tschischewski, lower temperatures produce higher nitrogen contents.

Note.—Possibly nitrification carried on at lower temperatures, where decomposition of ammonia is not so rapid, and with a large proportion of hydrogen mixed with the ammonia would give uniformly nitrified steel of small nitrogen content.

To determine the rate of penetration of nitrogen into typical steel, 1.5 in. lengths of bessemer rod about ½ in. in diameter and of 0.10 per cent. carbon were cleaned with emery paper and nitrified, with furnace cooling. Cross sections, polished and examined under the microscope, gave the results of Table 7.

Table 7.—*Penetration of Nitrogen into Bessemer Rod*

Piece Number	Temperature, Degrees C.	Time, Hours	Penetration, Millimeters
$Ar_1$3 Number 2	900	3½	0.6
$Ar_1$4 Number 2	800	4	0.7
$Ar_1$5	700	3	0.3

Etching with alcoholic nitric acid produced a dark exterior zone, allowing easy measurement of the depth of penetration. Heat tinting, according to Stead's directions, colored this zone a beautiful blue, as described by Strauss. It shows vividly against the red of the unnitrified parts and is easily distinguished from pearlite, barely colored by heat tinting.

Heat tinting gave indication of a retarding effect of phosphorus on the penetration of nitrogen; phosphorus segregations in heat-tinted ferrite showed as a darker red. In some of the nitrified specimens, dendrites of red ferrite extended into the blue nitrified zone, like fingers, and were of a deeper red especially inside the dendrite, exactly as in phosphorus segregations. These dendrites seem to prevent the penetration of nitrogen and recall similar dendrites found in case-hardened steel, especially in that containing nickel. The specimen of bessemer rod nitrified below Ac_1 and at 700° C. showed the dark nitrified zone penetrating along the grain boundaries toward the interior, indicating that nitrification of ferrite begins at grain boundaries.

Decomposition of Nitrified Steel by Heating

If the quantity of nitrogen remaining after heating is a function of the temperature of heating, it should be possible to obtain uniformly nitrified steel by decomposition of non-uniformly nitrified steel, provided that the original nitrogen content exceeds in every part that finally intended. Welding a spiral of sheet iron containing nitrogen from 2.46 to 1.94 per cent. reduced the nitrogen content to about 0.15 per cent. and indicated that decomposition could be easily carried out, in the laboratory. The high temperature required for welding, resulting in low nitrogen content, excludes this method for higher nitrogen contents.

It seemed especially important to determine the effect of nitrogen on transformation points, and nitrified steel produced by decomposition had to be of such shape as to lend itself to these determinations. The Leeds & Northrup transformation-point apparatus is of the LeChatelier neutral-body type and uses a solid specimen ¾ in. long and ⅜ in. in diameter, drilled along its axis to receive the platinum thermocouples. The nickel neutral body entirely surrounds the specimen and, for the specimen to increase in temperature, heat must flow radially into it from the nickel holder.

It is apparent that a solid specimen of these dimensions would require a very long time for sufficient nitrification. This time can be reduced by making the specimen of nitrified thin sheet iron, built in cylindrical form, provided it is so built that the radial flow of heat into it from the nickel holder cannot set up inequalities of temperature leading to transformation of different parts at different times. This provision excludes the use of specimens having concentric cylinders, for poor thermal contact between cylinders would surely cause temperature inequalities. But a specimen constructed of disks piled one on top of the other should work very well, even with perfect thermal insulation between adjacent disks; this form of specimen was adopted.

A tool-steel punch and die easily produced disks 11⁄32 in. in diameter from sheet iron 0.228 mm. thick. A square-head, threaded, nickel plug, screwing into the sample hole of the nickel neutral, adapted it for use with disks. To drill them for transformation-point determinations, the disks were placed in the tapped hole of the neutral, which had been lined with sheet mica, forced tightly together with the screw plug. The neutral and disks were then placed in a bench lathe and a 5⁄32-in. hole for the thermocouple was drilled into the disks for ¼ in., and a 1⁄32-in. hole through the remainder. The disks were then strung on 1⁄32-in. iron wire and separated by winding loops of finer wire over and between the disks. Ammonia circulated easily between these disks and nitrified them in every part.

Several such sets of disks were nitrified with ammonia in the same furnace as before. A few disks from each end of a set were analyzed before and after decomposition. Heating for decomposition followed nitrification; and where the temperature was not too high, was performed in the Leeds & Northrup apparatus exactly as in a transformation-point determination, the two operations being combined. Where decomposition is expected, the disks must not be pressed tightly together, as by screwing up the nickel-head screw, lest a pressure effect opposing decomposition be introduced.

A carbon-tube furnace used with a Leeds & Northrup optical pyrometer gave the higher temperatures required. Wrapping the set of disks in thin sheet iron protected it from contamination by carbon, and quenching from the high temperature avoided oxidation.

Table 8 shows the decomposition effected by various temperatures. The nitrogen contents of opposite ends of a set of disks are grouped in pairs. Analyses of opposite ends of such sets check with each other sufficiently well to indicate uniform nitrification as a result of decomposition. The curve in Fig. 6 has the general shape to be expected for a volatile solute in a non-volatile solvent, and may be interpreted as showing that temperature to which it is necessary to heat iron of any nitrogen content so that the vapor pressure of nitrogen due to the solid solution

TABLE 8.—*Decomposition of Nitrified Disks*

Number of Set	Temperature to which Heated, Degrees C.	Nitrogen before Heating, Per Cent.	Nitrogen after Heating, Per Cent.	Average, Per Cent.
HD	1400	0.756	0.0131	0.0131
		0.594		
GD	1300	1.17	0.0193	0.0193
		1.07		
FD	1198		0.0291	
		2.19	0.0323	0.0308
ED	1096		0.0922	
			0.124	0.103
CD	978	2.52	0.245–0.258	0.269
		2.27	0.259–0.313	
DD	843	4.56	0.743	0.755
		3.51	0.767	
AD	883	1.38	0.852	0.838
		1.14	0.817–0.828	
ID	816		1.156	1.12
		2.41	1.077	
KD	803	4.27	1.68	
			1.86	1.77
JD	782	2.75	1.92	
			1.88	1.89

of nitrogen in iron may practically equal one atmosphere. The part of the curve applying to lower temperatures approaches a horizontal straight line at approximately 750° C., indicating that below this temperature no decomposition of the solution takes place.

The point, at 0.838 per cent. nitrogen and 883° C., lying off the curve is probably removed because the plug was set up too tightly, forcing the disks together and building up a false pressure with liberated nitrogen. In general, any effect preventing the escape of liberated nitrogen will increase the nitrogen content after decomposition; thus the greater the thickness of steel the less will nitride be decomposed at any temperature, for if the steel remains sound there can be no exit for the liberated nitrogen, except at the surface. The curve may be in error for this reason, as time allowed for decomposition never exceeded 5 min.; possibly

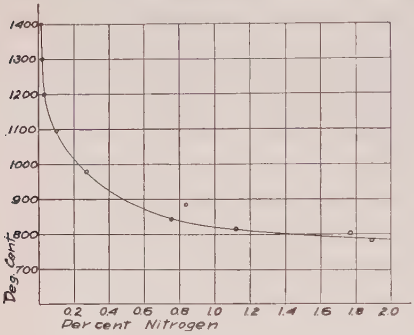

FIG. 6.

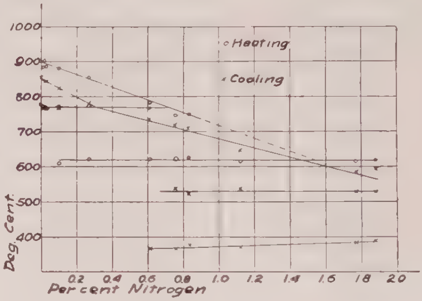

FIG. 7.

TABLE 9.—*Transformation Points for Nitrified Steel*

Sample Number	Per Cent. Nitrogen	*Ar*$_3$, Degrees C.	*Ac*$_3$, Degrees C.	*Ar*$_2$, Degrees C.	*Ac*$_2$, Degrees C.	*Ar*$_1$, Degrees C.		*Ac*$_1$, Degrees C.
Original........		844	901	770	770	....	....	....
After H_2 treatment........		857	900	777	777	....	....	....
HD	0.0131	837	884	766	770	....	....	743
GD	0.0198	843	904	769	769	....	....	....
FD	0.0306	843	888	768	768	....	....	....
ED	0.108	822	880	769	771	....	....	610
CD	0.269	780	854	....	774	....	....	621
BD	0.311	733	782	....	769	....	366	621
DD	0.755	719	749	....	....	538	366	621
AD	0.838	710	750	....	....	521	378	627
ID	1.12	647	....	....	....	538	371	616
KD	1.77	588	....	....	....	530	382	616
JD	1.89	595	....	....	....	530	388	619

Dotted lines indicate that no heat effect appeared.

the time should have been as long as that required for nitrification, but the slight thickness of the disks must have allowed rapid diffusion to their surfaces. Results from transformation-point determinations are given in Table 9. The terminology employed is that of the iron-carbon changes, though none of the sets of disks contained carbon, with the possible exception of set HD. Where confusion might exist, these changes are specially designated, as, for instance, Ac_1 for nitrogen, or Ac_1 for carbon.

One set of disks was treated in hydrogen at 680° C. for 1 hr. and furnace cooled; its transformation points were very sharp, the change at A_3 being marked with a large heat effect on both heating and cooling; absence of A_1 points indicated absence of carbon. A set of disks as punched from sheet iron showed no A_1 points, and the other points were not nearly so pronounced and clear as after hydrogen treatment. A set of disks HD heated to 1400° C. showed an abnormal depression of the A_3 points and also a slight absorption of heat at a temperature corresponding to Ac_1 for carbon. Both effects must be due to contamination by carbon during the heating in the carbon tube furnace. In general, the behavior of all points, except the Ar_1, was similar to those of carbon steel. Points indicated under the column of Ar_1, appear dual, and were faint and not suggestive of eutectoid changes; they are classified under Ar_1 for convenience in nomenclature. The Ac_1 points were of eutectoid character and their heat absorptions grew in magnitude with increasing nitrogen content. A faint point appeared at 305° C. and was independent of the character of the specimen; it may be the nickel point given as occurring at 327° C.

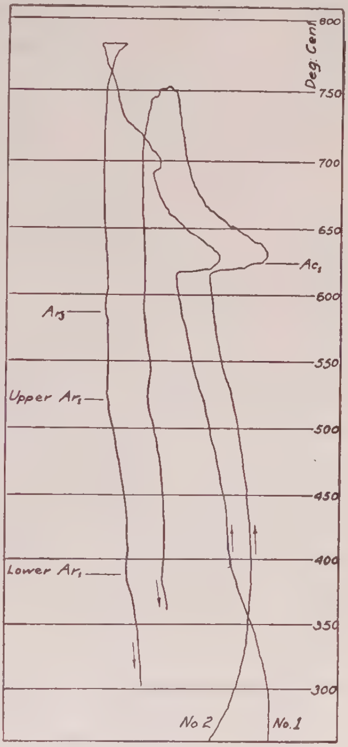

FIG. 8.

Fig. 7 shows the data of Table 9 assembled graphically and suggests a diagram for nitrogen and iron similar to that for carbon and iron, but with a much higher eutectoid composition. The Ac_3 points place this composition at 1.60 per cent. nitrogen and the Ar_3 points at 2.16 per cent. nitrogen; the average is 1.88 per cent. nitrogen. The Ac_1 point is much lower than for iron-carbon alloys and lies between 621° and 617° C.

The Ar_1 points were not clearly defined and did not behave like those of an eutectoid; their dual nature, especially, is hard to account for, see Fig. 8. If a set of nitrified disks having a large Ac_1 effect be cooled from above Ac_1 into the range between the two Ar_1 points and then reheated, its Ac_1 effect is much reduced in magnitude. Cooling below the lower Ar_1 point restores Ac_1 to its former magnitude. Possibly there is a slow formation of eutectoid between the two Ar_1 points, many of the cooling curves being irregular in this interval, as though eutectoid were separating unevenly. Moreover, the combined heat effects of the two Ar_1 points are small compared to that of Ac_1, as if part of the Ar_1 effect had been scattered over the range between. Were the lower Ar_1 point due to hydrogen, as described by Roberts-Austen,[12] it should have appeared with the hydrogen-treated disks.

Disks may be held, when being prepared for microscopical examination, by placing the end of the finger upon them. They may be mounted for photographing by pressing them against the moistened surface of an old photographic plate. A small wire magnet is convenient for manipulation during etching.

A thin outer layer of ferrite, probably formed by oxidation in the transformation-point apparatus, covers the true structure of the disk and must be removed. Black spots in some of the photomicrographs are probably caused by scale.

A crude bending test gave some idea of loss of ductility accompanying nitrification. A disk was held in a pair of pliers so that a diameter was coincident with the plier edge, and bent to the left and right so that the diameter was the vertex of an angle of 90° alternately on the left and right. The first bend swept through 90°, and the others 180° until the piece broke. The results are given in Table 10.

TABLE 10.—*Bending Tests on Nitrified Steel*

Nitrogen, Per Cent.	Total Bending, Degrees	Nitrogen, Per Cent.	Total Bending, Degrees
Orignal	1170	0.269	90
After H_2	1170	0.611	135
0.0131	1080	0.755	45
0.0198	630	0.836	70
0.0306	180	1.12	30
0.108	90	1.77	20

The photomicrographs do not indicate the existence of a nitride constituent until a concentration of nitrogen between 0.0308 and 0.108 per cent. has been reached. This is in accord with the previous findings on ingots, that nitrogen exists in solid solution in ferrite up to 0.030 per cent. The bend tests show a great loss of ductility for such solid solutions, increasing with increasing nitrogen content, as has been stated by Harbord,[13] Braune,[14] LeCarme,[15] Tschischewski,[16] Strohmeyer,[17] Gieson,[18] Herwig,[19] Strauss,[20] Dudley,[21] Ruder,[22] but doubted by Stead in the discussion of Tschischewski's paper.

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[12] Fifth Report of Alloys Research Com. (1899) 39.

Eutectoid first appears in set ED, containing 0.108 per cent. nitrogen, and thereafter increases in amount with the nitrogen content. At 100 diameters, it has the appearance of pearlite and to such nitrified steels may be assigned a corresponding, fictitious, carbon content, estimated by the eye, and convertible by proportion into the eutectoid composition for nitrogen. Thus Fig. 19 has the same appearance as the photomicrograph of a 0.30 per cent. carbon steel; knowing that it contains 0.611 per cent. of nitrogen, the proportion 0.30 : 0.85 = 0.611 : x, follows and the eutectoid composition x may be calculated. Table 11 shows results of such calculations. Nitrified irons containing about 1.70 per cent.

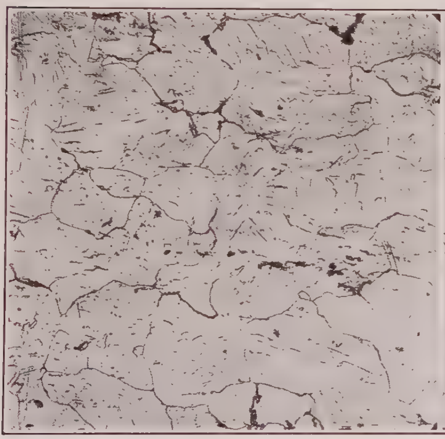

Fig. 9.—Original sheet iron; bending 1170°. Note oxide between grain boundaries; very little cementite. × 100.

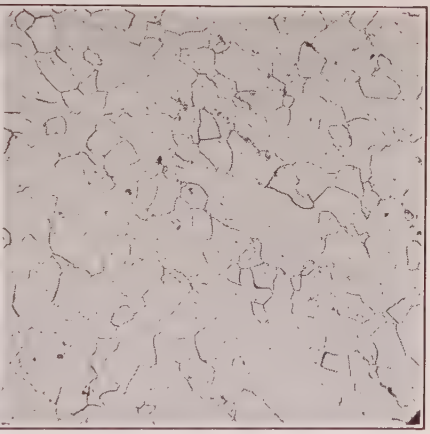

Fig. 10.—After treatment in hydrogen at 680° C. for 1 hr; furnace cooled in hydrogen; bending 1170°. Note absence of oxide between boundaries, and absence of carbon eutectoid. × 100.

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[13] *Jnl.* Iron and Steel Inst. (1896) **50,** II, 161.

[14] *Rev. de Metallurgie, Etraits.* (1905) 361.
Stahl und Eisen. (1907), **27,** II, 1395.
Bull. de L'Industrin Minerale (1907) **7,** 489.

[15] *Rev. de Metallurgie, Etraits.* (1905) I, 516.

[16] *Op. cit.*

[17] *Jnl.* Iron and Steel Inst. (1909) **79,** I, 404.

[18] Carnegie Scholarship Memoirs. (1909) **1,** 1.

[19] *Stahl und Eisen.* (1913) **33,** 1721.

[20] *Stahl und Eisen.* (1914) **34,** II, 1814.

[21] *Jnl.* Ind. and Eng. Chem. (1910), **2,** 299.

[22] G. F. Comstock and W. E. Ruder: *Chem. and Met. Eng.* (1920) **22,** 399.

of nitrogen, consist entirely of nitrogen eutectoid. This eutectoid etches like troostite, and is difficult to resolve under the microscope, even when slowly cooled.

TABLE 11.—*Eutectoid Compositions Calculated by Microscopic Estimation*

Figure	Apparent Carbon, Per Cent.	Nitrogen, Per Cent.	Eutectoid Composition, Per Cent.
19	0.30	0.611	1.73
	0.35	0.755	1.83
20	0.42	0.838	1.70
21	0.60	1.12	1.59
22	0.85	1.77	1.77

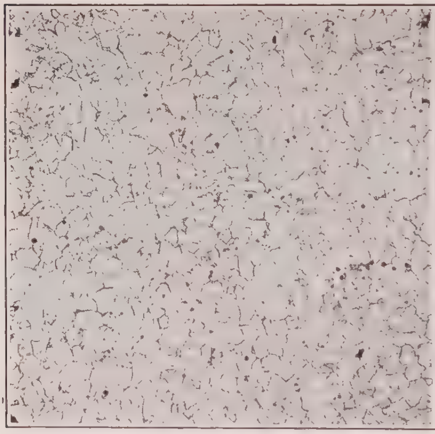

FIG. 11.—SET HD HEATED TO 1400° C. NITROGEN 0.0131 PER CENT.; BENDING 1080°. NOTE ABSENCE OF CARBON EUTECTOID. × 100.

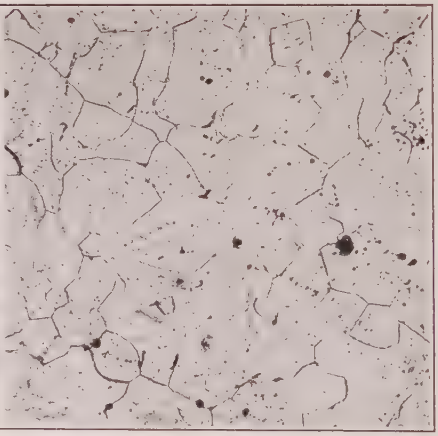

FIG. 12.—SAME AS FIG. 11; SPECKLED APPEARANCE POSSIBLY CAUSED BY FORMATION OF CAPILLARIES DURING DECOMPOSITION OF NITRIDE. × 400.

These values agree fairly well and place the eutectoid composition at 1.70 per cent. nitrogen, which is not far removed from the 1.88 per cent. determined by thermal analysis. It is at about this composition, too, that the curve, Fig. 6, of limiting temperatures for different nitrogen contents approaches a horizontal straight line, indicating a maximum concentration of nitrogen in solid solution. Any nitrides not in solution would be decomposed before reaching this temperature, while if the solution could exist at a greater nitrogen content (*i.e.*, if this were not the eutectoid point) the decomposition curve would not be horizontal.

Bend tests show great brittleness for all steels containing eutectoid, even though cooled slowly enough for Ac_1 to develop fully on heating; the eutectoid appears, therefore, to be inherently brittle. Brittleness of the eutectoid alone, however, should not render brittle those steels consisting mostly of ferrite, unless the ferrite also is brittle. The brittleness of ferrite is consistent with the approximately constant results of the bending tests for disks containing from 0.108 to 1.77 per cent. nitrogen, as though the results were independent of the proportion of eutectoid present.

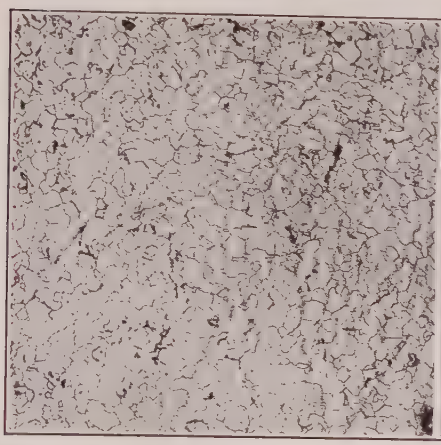

Fig. 13.—Set GD heated to 1300° C.; nitrogen 0.0198 per cent.; bending 630°. No nitride needles. × 100.

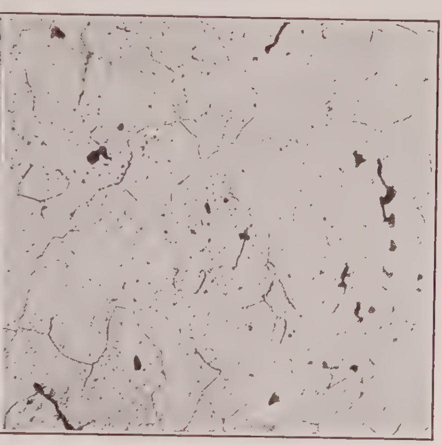

Fig. 14.—Same as Fig. 13. × 400.

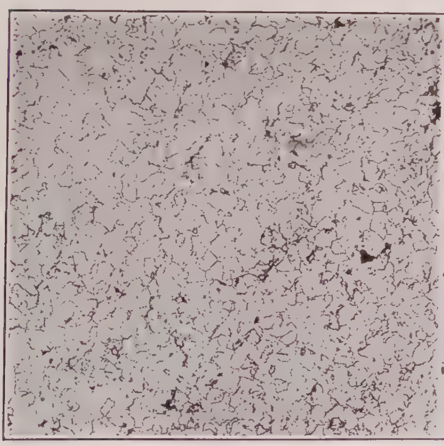

Fig. 15.—Set FD heated to 1198° C.; nitrogen 0.0306 per cent.; bending 180°. Note absence of nitride needles. × 100.

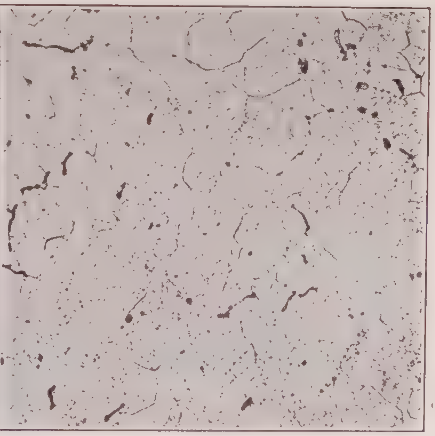

Fig. 16.—Same as Fig. 15. Note absence of nitride needles and presence of some oxide. × 400.

With reference to the transformation points, many of the heating curves (see Fig. 23) from disks of 2–4 per cent. nitrogen show an inflection at 380–390. Kido,[23] by magnetic analysis, has found transformation points in nitrified steel which he assigns to definite compounds as follows: Fe_2N at 250° C.; double carbide alpha at 350° C.; Fe_2N at 470° C.; double carbide beta at 620° C. The inflection may correspond to the completion of the change at 350° C.; Kido assigns this change to a double carbide, which is hard to reconcile with the negligible amounts of carbon found in the disks. That it is not necessary to postulate double carbides

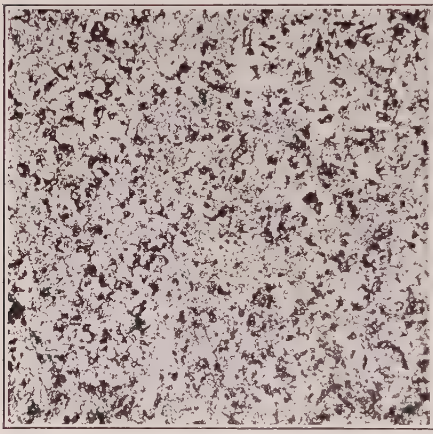

Fig. 17.—Set ED heated to 1096° C.; nitrogen 0.108 per cent.; bending 90°. Note presence of nitride eutectoid. × 100.

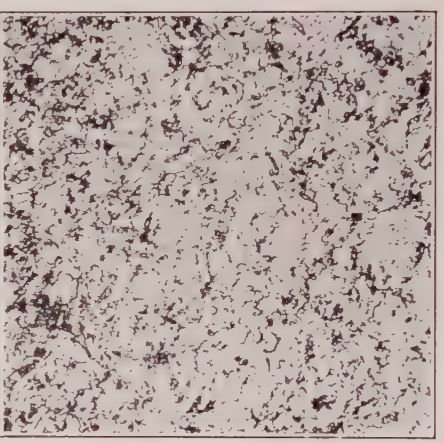

Fig. 18.—Set CD heated to 978° C.; nitrogen 0.269 per cent.; bending 90°. Nitride eutectoid increasing. × 100.

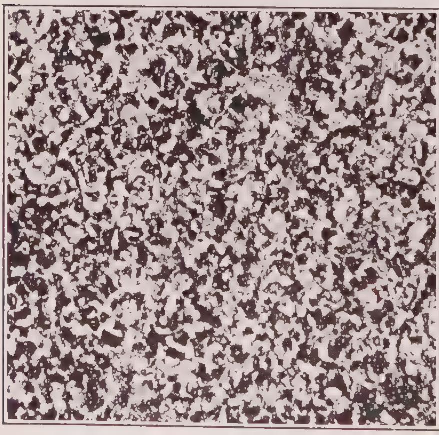

Fig. 19.—Set BD heated to 930° C.; nitrogen 0.611 per cent.; bending 135°. Nitride eutectoid increasing. × 100.

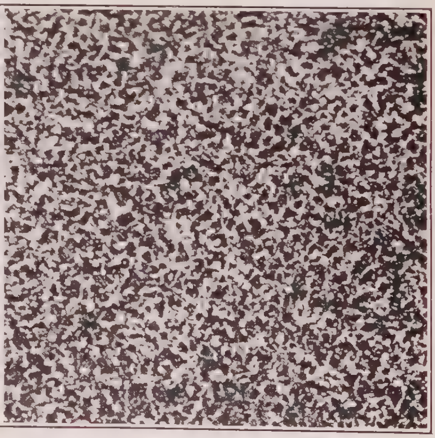

Fig. 20.—Set AD heated to 833° C.; nitrogen 0.838 per cent.; bending 70°. Nitride eutectoid increasing. × 100.

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[23] *Sci. Rept.* Tohoku Imp. Univ. (1921) **10, 471.** 1st series.

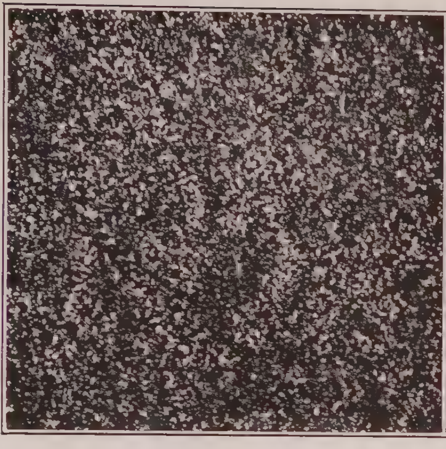

FIG. 21.—SET ID HEATED TO 816° C.; BENDING 30°; NITROGEN 1.12 PER CENT. NEARLY ALL EUTECTOID. × 100.

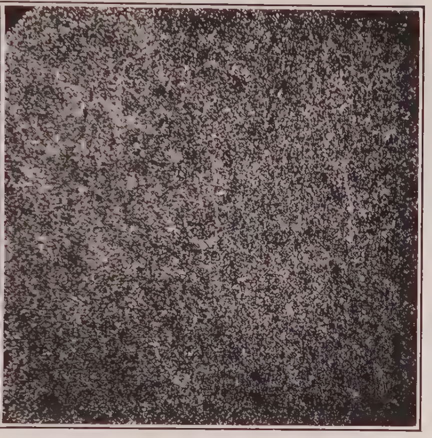

FIG. 22.—SET KD HEATED TO 803° C.; BENDING 20°; NITROGEN 1.77 PER CENT. WHOLLY EUCTECTOID. × 100.

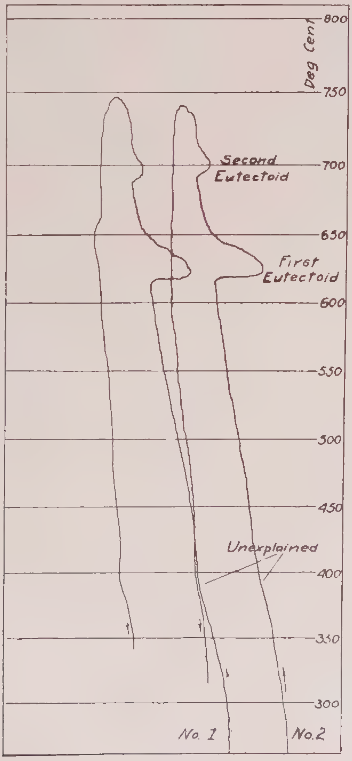

FIG. 23.

is indicated by the close correspondence of Kido's second double carbide at 620° C. with the nitride eutectoid already described. No heat effects were detected that correspond to the other points noted by Kido.

Such curves as those in Fig. 23, besides showing the first eutectoid transition, Ac_1 in Fig. 8, show another transition at 693° C., sharply defined and constant in temperature for varying nitrogen concentrations, but disappearing after heating to 780° C. Its behavior suggested a second eutectoid of iron with nitrogen; this was confirmed by microscopical examination. Such narrow cross sections as the disks afforded were difficult to prepare and the surface obtained was always rounded, preventing the taking of photographs. However, they showed: first, a dark center, corresponding to the first eutectoid, often with bright veins between the grains like cementite in a hypereutectoid carbon steel; second, on each side of the center portion and often reaching into it, a bright thin layer of that same material found between grains of the first eutectoid; third, this bright layer merging into a darker layer having the laminated appearance of a eutectoid and thought to correspond to the upper eutectoid point at 693° C.; fourth, outside of this second eutectoid, a second bright layer separated by a sharp line from the outermost layer; fifth, the outermost layer, of bright material showing many clearly and sharply defined twins. This outermost layer must have been quite fragile, as it was often lost in polishing.

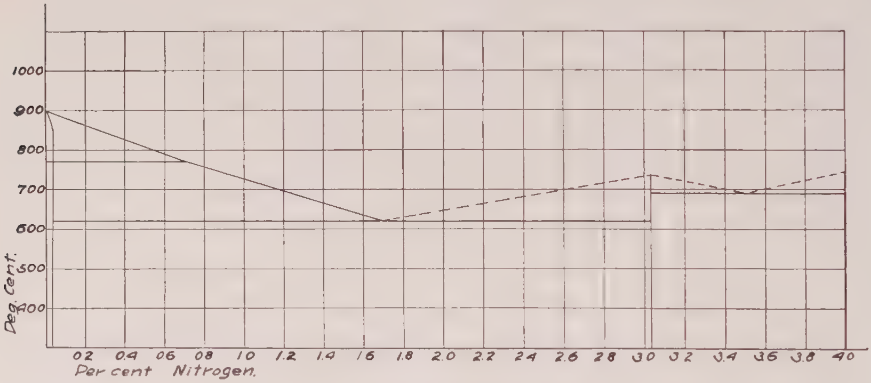

FIG. 24.

In a paper by Noyes and Smith,[24] on "The dissociation pressures of iron nitrides" results are presented which indicate the existence of the following nitrides:

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[24] *Jnl.* Am. Chem. Soc. (1921) **43,** 475.

	Per Cent. Nitrogen
Fe_8N	3.03
Fe_6N	4.00
Fe_4N	5.88
Fe_2N	11.1

These results tend to confirm the observations made on cross sections of disks and admit the possibilities of eutectoids between iron and Fe_8N, between Fe_8N and Fe_6N, and also of a compound Fe_4N corresponding to the outermost layer. All these compounds might exist together in a non-uniformly nitrified disk, containing, on the average, only 4 per cent. nitrogen. The compound Fe_2N has been noted by Fowler[25] and others. Combining the results and indications so far obtained, a tentative iron-nitrogen equilibrium diagram for heating may be constructed, as shown in Fig. 24, with hypothetical lines broken.

SUMMARY OF RESULTS

For obtaining the most accurate results with the distillation method for determining nitrogen in steel, a Kjeldahl trap and a glass-wool strainer should be inserted between condenser and distillation flask. The condenser tube should be of some material other than glass, which yields alkalies to the distillate.

A 2.42 per cent. gain in weight by thin sheet iron, because of cyanide hardening, is 97.5 per cent. accounted for by results of analyses for carbon and nitrogen, indicating the recovery by the distillation method of all nitrogen introduced into steel by case-hardening processes.

The combustion method for carbon is adapted to total nitrogen determinations. Results so far obtained by this method indicate complete recovery by the distillation method of all nitrogen introduced into steel by any method. A specially designed phosphorus pipette can absorb from 3 to 4 liters of oxygen at a rate of from 15 to 20 cc. per minute.

Iron melted under an atmosphere of nitrogen absorbs it in accord with the formula:

$$\%N = k\sqrt{P_N}$$

where $\%N$ = per cent. of nitrogen in cooled ingot;
k = a constant having value of 0.020;
P_N = pressure in atmospheres of nitrogen during melting and solidification.

The significance of this formula in steelmaking is discussed.

Microscopic examination of ingots melted under nitrogen discovers no trace of nitrogen up to approximately 0.03 per cent., and indicates a solid solution of nitrogen in ferrite. Above 0.03 per cent., many needles were found.

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[25] *Op. cit.*

Iron catalyzes the dissociation of ammonia at elevated temperatures and is denitrified by hydrogen so formed; it is therefore difficult to produce uniformly nitrified steel by passing ammonia over it.

Uniformly nitrified thin steel disks, containing up to 1.70 per cent. nitrogen, can be produced by heating non-uniformly nitrified disks, containing from 3 to 4 per cent. nitrogen, to different temperatures for different degrees of decomposition.

Decomposition so produced can be represented by a curve of per cent. nitrogen against temperature; its general shape is that corresponding to similar curves for volatile solutes in non-volatile solvents.

Transformation points obtained from sets of these disks clearly indicate an iron-nitrogen diagram of appearance similar to the iron-carbon diagram.

Results of photomicrographic examination of the disks are in accord with general findings of transformation-point determinations, and place the eutectoid point at 1.70 per cent. of nitrogen. No nitrogen constituent was visible in ferrite up to a content of 0.030 per cent.

Results of bend tests on disks show a great loss in ductility for nitrogen contents above 0.015 to 0.030 per cent., confirming statements in the literature.

In non-uniformly nitrified disks containing 3 to 4 per cent. nitrogen, the existence of a second eutectoid is indicated both microscopically and by thermal analysis.

ACKNOWLEDGMENT

The work herein described was carried out at the Massachusetts Institute of Technology, under the direction of Prof. Henry Fay, and at his suggestion. He has at all times given it his kindly and productive aid.

APPENDIX

DISTILLATION METHOD

The method is essentially that described by Hurum and Fay.[26] Preliminary tests with Hurum and Fay's distilling apparatus indicated: (1) That pieces of zinc placed in the distilling apparatus will prevent all bumping. (2) That an ordinary Erlenmeyer flask, loosely capped with a cork, serves as well for receiving the distillate as a more elaborate apparatus. Ammonia, pipetted into the distilling flask in quantities varying from 1 to 100 cc. of N/100 ammonia solution, was completely recovered as long as acid in excess was present in the receiver. (3)

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[26] *Op. cit.*

That a modification of the Kjeldahl trap and a plug of glass wool inserted between condenser and distilling flask entirely prevent occasional carrying over of alkali from the distilling flask, presumably the result of foaming. (4) That a condenser tube of ordinary soft glass yields sufficient alkali to 50 cc. of distillate to correspond to 0.003 per cent. of nitrogen in a 1-gm. sample (Tschischewski);[27] but that a condenser tube of fused silica yields no alkalies to the distillate.

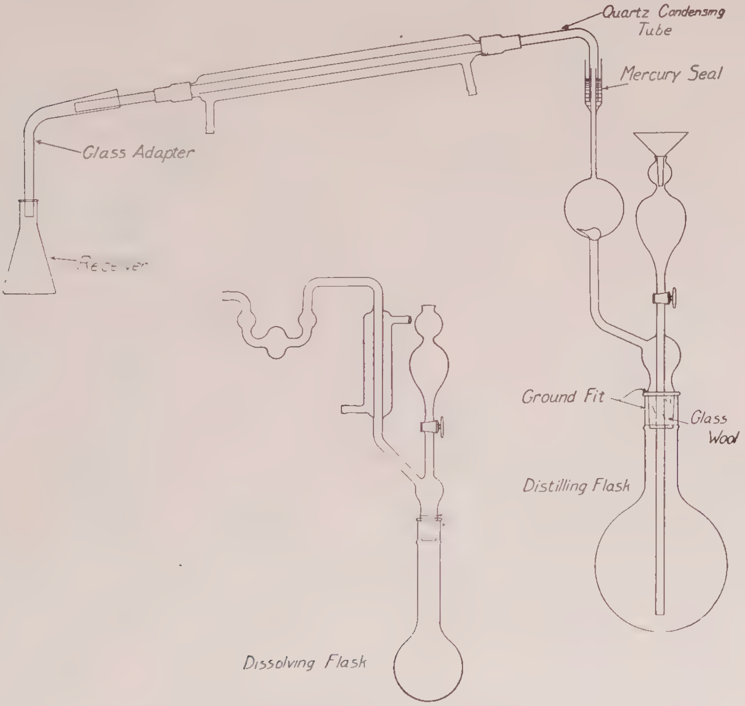

FIG. 25.—APPARATUS FOR NITROGEN ANALYSIS BY SOLUTION METHOD; ONE-EIGHTH ACTUAL SIZE.

The apparatus, with appropriate modifications, is shown in Fig. 25. To make an analysis, the weighed sample is placed in the dissolving flask with a sufficient quantity of redistilled hydrochloric acid. A dilute solution of the acid, placed in the wash bulbs, catches any spray carried up by hydrogen bubbles. When solution is complete, the wash liquor and solution are combined and transferred to the dropping funnel of the distilling flask.

This flask contains 400–500 cc. of distilled water, and 50 cc. of sodium-hydroxide solution, previously freed from nitrites, nitrates, and ammonia, and equivalent in strength to hydrochloric acid of sp. g. 1.12. The solution in the flask has been kept boiling until 1 cc. of N/100 sulfuric acid, pipetted into a 25-cc. fraction of the distillate, is completely recovered by iodate-iodide titration with N/100 sodium thiosulfate. The solution of the sample is then slowly run into the boiling contents of the flask and successive 25-cc. fractions of the distillate are titrated, until 1 cc. of N/100 acid is again completely recovered, when the solution of another sample may be run into the distillation flask. All alkalinity found in the distillate is attributed to ammonia derived from nitrogen in the steel, and is calculated as such.

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[27] *Op. cit.*

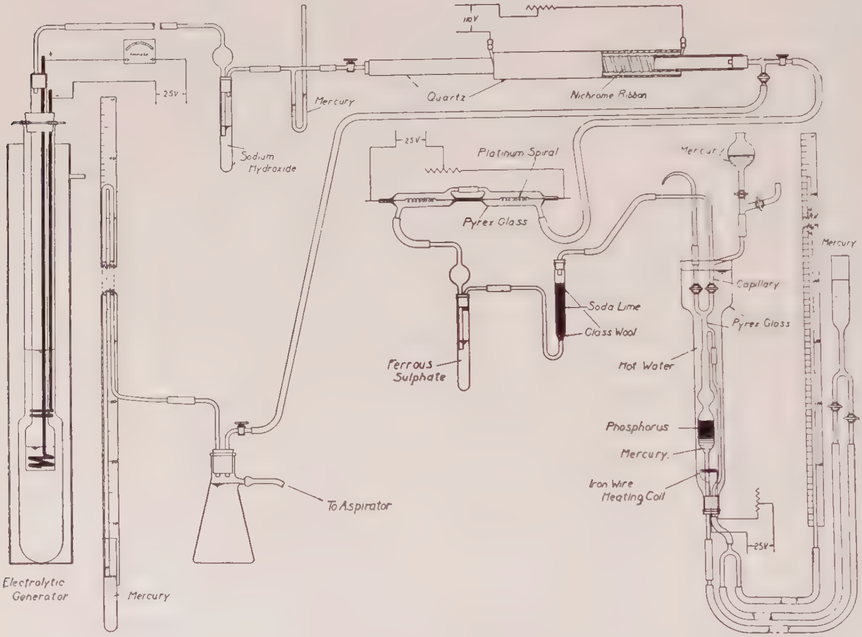

FIG. 26.—APPARATUS FOR NITROGEN ANALYSIS BY COMBUSTION METHOD; ONE-TWELFTH ACTUAL SIZE.

The ability of this method to check its own results appears to be limited only by the accuracy of the ordinary 50-cc. burette, when using N/100 standard solutions, and by the degree of freedom from contaminating ammonia gas of the laboratory air. Of course, check samples must be uniform. Thus, four adjacent samples of about 2 gm. each from a 1/4-in. drill rod gave the following results in per cent. of nitrogen: 0.0060, 0.0045, 0.0030, 0.0034. But three analyses, made on 2-gm. samples of carefully mixed turnings gouged with a dull lathe tool from the same drill rod, gave results in per cent. of nitrogen: 0.0032, 0.0031, 0.0031. The nitrogen equivalent of each of these 2-gm. samples is about 0.45 cc. of N/100 acid. Had 5-gm. samples been used, or had the nitrogen content of the samples been 2.5 times as great, the acid equivalent would have increased to about 1.05 cc., yielding three significant figures in the results. The size of the samples to be used is thus determined by their nitrogen contents and by the number of significant figures desired in the results.

Combustion Method

The ordinary combustion method for total carbon is adaptable to total nitrogen determination. Oxygen from an electrolytic generator enters the combustion tube, and combines with the iron, liberating its nitrogen in molecular form, which can then be swept out and estimated by gas analysis. Fig. 26, shows the apparatus used in obtaining the results stated below.

The oxygen generator is of the usual laboratory design, having electrodes of iron and electrolyte of strong sodium hydroxide. It operates on a 20-volt circuit taking a current of 5–8 amp., which is automatically made and broken when the pressure of oxygen in the center compartment falls or rises. In spite of its larger cross-sectional path for the current, the heat developed made water cooling necessary. Oxygen so generated always contained hydrogen, which had to be removed, and also a residual gas, probably nitrogen which necessitated a correction proportional to the oxygen used in an analysis. The purity of the yield increased with greater concentration of electrolyte and lower temperature, both tending to increase viscosity and to prevent the circulation of electrolyte. From the generator the oxygen passes into a sodium-hydroxide bubbler for indicating rate of flow, and then connects with a small mercury manometer.

The furnace consists of an ordinary quartz combustion tube, wound with nichrome ribbon over which is slipped a second quartz tube, providing a slight thermal insulation. The furnace required a current of about 10 amp. at 110 volts, and would heat from room temperature to 950° C. in 4 to 5 min. after the circuit was closed.

Oxygen enters the furnace through a glass stop-cock and rubber stopper, and leaves it by stop-cocks allowing connection either with a water aspirator and manometer, or with the gas analysis apparatus.

A platinum spiral maintained at red heat by an electric current insured complete oxidation of the gaseous products of furnace combustion. The device containing the spiral had two stages and was most efficient in the horizontal position shown. It removed hydrogen mixed with the oxygen and converted carbon monoxide to carbon dioxide. Leaving the spiral, the gas enters a slightly acid ferrous-sulfate washer, included as a test for nitrogen oxides. No darkening of this wash solution was observed, nor could Hurum, by the most careful tests, detect any trace of nitrogen oxides formed by the combustion of nitrified iron.

Next a soda-lime absorption tube removed all carbon dioxide and other acidic gases, leaving only nitrogen, oxygen, and water vapor; the soda lime used was that described by Wilson.[28] The usual sodium-hydroxide absorption tube is not suited to this purpose, as from 1 to 3 cc. of carbon dioxide can escape absorption following combustion of a high-carbon steel.

A specially designed phosphorus pipette absorbed oxygen and water vapor, thus completing the isolation of nitrogen. This pipette was of such capacity and construction that it could absorb from 3 to 4 liters of oxygen at a rate of 15 to 20 cc. per min. Sufficient heat is developed to melt sticks of phosphorus, excluding their use; therefore this pipette was designed to operate with melted phosphorus, floated on the surface of mercury and maintained at a temperature of about 60° C. by means of an electric heating coil and water bath. The pipette, constructed throughout of pyrex glass, has two capillary stop-cocks leading into the graduated body and bulb, with combustion chamber below. The capillary at the left provides for the introduction of liquid phosphorus and is sealed, by water, against inward leak of gas; that at the right allows the entrance of gas for analysis and is sealed by dropping mercury from a small reservoir above. A bypass outlet for gas is also provided. An inlet in the bottom of the combustion chamber allows the level of the mercury to be raised or lowered, and the volume of the gas space may be controlled by setting the phosphorus surface at any desired graduation mark. A side arm in the capillary inlet to the body of the pipette connects with an open manometer and indicates the pressure of gas above the phosphorus. Knowing the temperature, pressure, and volume, the quantity of gas can be calculated.

To make a combustion for nitrogen, the steel, weighed and placed in a boat, is introduced into the cold furnace. This is then closed and evacuated to the capacity of the aspirator (50 to 70 cm. of mercury depending on the water pressure at the moment) and filled with oxygen. After the furnace has been evacuated and filled with oxygen three or four times very little nitrogen should be left in the granular alundum or retained on the furnace walls. When the platinum spiral is at a bright red heat, oxygen is passed through the furnace and combustion train to the pipette bypass, for about 1 hr., when all nitrogen should be swept from furnace and train. The oxygen volume per minute is 22 to 30 cc., or that produced by a current of from 6 to 8 amp., exactly determined by an ammeter.

Meantime the phosphorus absorption pipette is prepared for use by heating the water bath to 60° C. and by drawing in about 10 cc. of melted phosphorus from a beaker of hot water. Some water always enters with the phosphorus, and should be, as nearly as possible, expelled before the stop-cock is closed. Withdrawing mercury from the pipette lowers the level of the phosphorus until it occupies the position shown. Admission of oxygen into the vacuum so formed induces combustion with accompanying production of phosphorus oxides and absorption of all remaining water. Absorption may, at first, proceed with difficulty, and can be helped by dropping mercury from the manometer side arm. But, with the drying up of water, absorption becomes smooth and will continue until phosphorus oxides, which appear to enter into solution with liquid phosphorus, impede absorption and necessitate replacement with fresh phosphorus. At this stage, the pipette should be cleaned with hot water. Liquid phosphorus will dissolve products of combustion remaining after hot water treatment. One charge of phosphorus usually suffices for two combustions.

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[28] *Jnl.* Ind. and Eng. Chem. (1920) **12,** 1000.

During the sweeping out process, the purity of gas coming from the absorption train is tested from time to time by 10-min. absorptions with the phosphorus pipette until determinations of cubic centimeters of residual gas agree at a satisfactory value. (About 0.20 per cent. residual gas by volume.) Products of phosphorus combustion deposited on the sides of the pipette may trap gas, and each pipette reading must therefore be checked. Mercury dropped from the manometer side arm sweeps the gas in the pipette over the phosphorus surface and insures absorption of any oxygen remaining in upper parts of the tube.

When blank determinations are satisfactory, the outlet is shut off and the pipette and absorption train are tested for leaks. If there are none, full current is turned on in the furnace, causing combustion to begin in less than 3 min. With quick heating, combustion proceeds slowly and evenly to completion; while with slow heating, the iron flashes and absorbs sufficient oxygen to produce a partial vacuum in the furnace. This would favor inward leak of nitrogen, to be avoided throughout, by always maintaining the pressure of oxygen slightly in excess of one atmosphere. The ampere-hours required to furnish oxygen for combustion are noted, and added to those required to furnish gas for sweeping the furnace gases into the phosphorus pipette, an operation begun toward the end of combustion. Usually 40 min. at 6 to 8 amp. suffice to sweep all nitrogen into the pipette, at the end of which time, analyses of gas coming from the absorption train should agree with one another and with the determinations made just before combustion.

From the total number of ampere-hours required for combustion and sweeping out is computed the total volume of residual gas, and this subtracted from the volume of gas isolated in the pipette (all volumes are reduced to 0° C. and 760 mm.) leaves the volume of nitrogen derived from the steel sample.

Certain changes in the apparatus would increase speed and accuracy; they could be effected as follows:

1. Allowing only a small vent for hydrogen in the generator, thereby excluding nitrogen from the hydrogen chamber whence it may diffuse through the electrolyte and become responsible for the residual gas.

2. Sealing in glass-tubing and eliminating all rubber connections and corks from the gas train.

3. Sealing the exit end of the silica combustion tube to glass, and making a gas-tight ground fit at the entrance end, for introducing the boat.

4. Replacing the heated platinum spiral with a plug of copper-oxide gauze or platinized asbestos, introduced into the exit end of the combustion tube and maintained at the required temperature by a separate heating coil.

5. Eliminating the ferrous-sulfate washer.

6. Providing a good vacuum pump.

7. Increasing the size of the combustion chamber of the pipette and the size of the capillary exit for phosphorus, which may become clogged with phosphorus oxides when drawing off used phosphorus.

DISCUSSION

James A. Aupperle*, Middletown, Ohio (written discussion).—The scope covered by the author's experiments is large and the results obtained indicate the importance of knowing the nitrogen content of various metals. It is an established fact that nitrogen plays an important part in the physical properties of steel. Some authorities claim that nitrogen has an embrittling effect nine times greater than phosphorus, so that a tool steel containing appreciable amounts of nitrogen will not stand much shock when extremely cold, as such metal is cold short.

The author shows that steel containing 0.03 per cent. nitrogen yields 0.48 cc. of the gas. It is unfortunate that the blank amounts to four times as much as the nitrogen found. The author realizes this fact and says that variations in results on 2-gm. samples of steel, containing no nitrogen, may equal 0.025 per cent.

In Table I, the nitrogen content of Armco iron is shown to be extremely low, when determined by the combustion method; in fact, no nitrogen was obtained by the combustion method on Armco iron as received. On the other hand, the distillation method on the same material yielded 0.0048 per cent. The author does not attempt to explain the reason for obtaining nitrogen by the distillation method in the Armco sample and obtaining no nitrogen in the same sample by the combustion method. It is difficult to conceive how the nitride of iron obtained by the distillation method exists in the iron-oxide residue, although this would be inferred from the fact that the nitrogen was not obtained by the combustion method. This is undoubtedly accounted for by the inaccuracy of the combustion method, as nitride nitrogen is apparently determined when it exists in large amounts. If this is a fact, the method cannot be used for determining the nitrogen as it exists in iron or steel, as in most varieties the total nitrogen does not exceed 0.01 per cent. while in other varieties it runs as high as 0.04 per cent. In oxyacetylene welds, the nitrogen runs about 0.04 per cent. in the welded metal; in welds produced by electric welding, it runs from 0.16 to 0.20 per cent.

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* Chief Chemist, American Rolling Mill Co.

The amount of nitrogen obtained on a 2-gm. sample of steel containing 0.03 per cent. nitrogen amounts to about ½ cc. The fact must not be overlooked that this amount of gas is more than twice the volume of the metal used in the determination.

It is of just as much importance to know the amount of dissolved nitrogen in ferrous metals as it is to know the amount of combined nitrogen as obtained by the distillation method. On page 814, the author refers to the preparation of disks for microscopical examination, from which it is inferred that he examined the face and not the edge section, as he refers to such narrow cross sections as disks being difficult to prepare for microscopical examination on account of always obtaining rounded surfaces which prevented taking photographs. It appears that attempts were made to polish a single specimen which caused the edge to become rounded. Had the author bolted several disks together with stove bolts and then paraffined them, he would have had no difficulty when polishing such samples for microscopical examination. An examination of the edge would have yielded more information as the depth of the penetration of nitrogen could have been observed.

The apparatus employed for determining nitrogen by the combustion method is very elaborate, and undoubtedly requires considerable skill in manipulation. The results obtained on the higher nitrogen samples seem to show that satisfactory agreement on different determinations is possible.

The information that nitrified steel decomposes at high temperatures is of interest and, from results obtained, the author has shown that it should be possible to obtain uniform nitrified steel by heating steel in which the nitrogen is segregated.

The effect of nitrogen on the transformation points seems very pronounced. The nitrogen has a similar effect to carbon on such transformation points, although the nitrogen effect does not seem to be as pronounced as the carbon effect.

We have studied the effect of nitrogen in steel from a corrosion standpoint, and have found that the combined nitrogen, as obtained by the distillation method, has an important bearing on the rate of corrosion. Our results show that steel containing three times as much nitrogen as another sample of steel will corrode approximately three times as fast when the metal is exposed to atmospheric conditions. We have studied the failures of other forms of ferrous metals in the form of pipe and spikes;

both of these materials corroded very rapidly. The pipe contained 0.041 per cent. nitrogen and the spikes contained 0.033 per cent. nitrogen; this is almost ten times as high as the nitrogen content of commercially pure iron.

Many investigators are studying the gas content of ferrous metals at the present time, and we can look forward to interesting data, both as to the methods for determining various gases in iron and steel and their effect on the physical properties and resistance to corrosion.

C. Baldwin Sawyer (author's reply to discussion).—I wish to emphasize again that the combustion method for total nitrogen is only the start of a method capable of much refinement. Neither time nor apparatus were available for this refinement, but fortunately sufficient results were obtained to place reliance on the more rapid and accurate distillation method, when used on the steels of this research. This achievement was sufficient for the present research. The greatest service of the combustion method or of any other method for total nitrogen will be rendered in defining the conditions under which the distillation method determines all of the nitrogen present in steel.

The Armco iron, as received, contained too little "nitride nitrogen" for determination by the combustion method, which accounts for the discrepancy with the distillation method. The significance of this single analysis, by the combustion method, would have remained unaltered had the numerical result assumed any value between ±0.025 per cent. That it should have been exactly zero was purely accidental. This single analysis was undertaken only to show that no large amount of "uncombined nitrogen," say 0.05 per cent., existed in commercial Armco iron.

In general, disks were prepared for photomicrography by polishing face sections. By varying the amount of grinding of the face surface on emery paper, it was determined that the nitrification was quite uniform. Owing to the extreme brittleness of the nitrified disks, it was not possible to bolt them together, as suggested. However, had time permitted, they would have been copper- or nickel-plated, and their edge sections thus protected from rounding during polishing.

Some Commercial Alloys of Iron, Chromium, and Carbon in the Higher Chromium Ranges

By C. E. MacQuigg, E. M., Long Island City, N. Y.

(Canadian Meeting, August, 1923)

In this paper it is impossible to more than touch on many of the commercial alloys of iron, chromium, and carbon, therefore the discussion is confined to the properties of some of the less well-known alloys with upwards of 20-per cent. chromium. The discussion has been based largely on the strictly engineering aspects of the alloys.

Chromium has well-defined effects when added to the iron-carbon system. These effects depend on the amount of chromium added and on the composition of the iron-carbon alloy to which the addition is made, *i.e.* whether low- or high-carbon steel, cast iron, etc. In the case of the low-carbon steels, the chromium addition, when not in excess of several per cent., appears to exert an influence on the properties by its influence on the critical range. In the higher chromium alloys, the physical properties are influenced chiefly by the formation of new complexes, such as iron-chromium solid solutions and, with higher carbon contents, iron-chromium carbides of great hardness.

Examination of the published data[1] on the critical range phenomena of chromium-bearing steels up to several per cent. chromium indicates an agreement on the following points:

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[1] The Properties of Some Chromium Steels. R. D. Report No. 51 Res. Dept. Woolwich.

Takejiro Murakami: On the Structure of Iron-carbon-chromium Alloys. Sci. Reports Tohoku Imper. University (1918) **7,** 217.

Edwards, Sutton, and Oishi: Properties of Iron-chromium-carbon Steels. *Jnl.* Iron and Steel Inst. (I, 1920).

Edwards, Greenwood, and Kikkawa: Initial Temperature and Critical Cooling Velocities of Chromium Steel. *Jnl.* Iron and Steel Inst. (I, 1916).

On the Constitution of Chromium Steels. *Car. Res. Mem. Jnl.* Iron and Steel Inst. (II, 1921) 247.

1. Alloys of iron and chromium without carbon have their magnetic change point A_2 depressed, the depression being a function of the chromium content. The elements iron and chromium form an unbroken series of solid solutions, the lag between Ar and Ac is quite small, and the position of neither is much affected by previous heat treatments.

2. Chromium raises the thermal critical range, the elevation of the range being a direct function of the amount of chromium present and the amount of carbon contained in the alloy.

The outstanding effects on steel of chromium additions up to several per cent. are the development of hardness without brittleness in the treated specimens, a finer grain, and the lessening of the mass effect in heat treatment. The effect of lowering the carbide change, by even moderately slow rates of cooling, is to cause the chromium steels to assume desirable physical properties with treatments much less drastic than are required for plain carbon steels of like carbon content. This results from the suppression of the critical range by undercooling.

Much investigation has been conducted on ternary steels, from those containing chromium in a subordinate degree (say less than 1 per cent.) to those used for case hardening in the ball-bearing industry and into the ranges around 12 to 14 per cent., *e. g.*, stainless steels. Until recently there has been a gap in the plain chromium steels (from about 1.5 per cent. chromium) to the stainless-steel range. Some excellent steels have been described by Aitchison[2] and unusually good properties obtained with 2.5 to 3 per cent. chromium. The combination of hardness, tensile strength, and toughness suggests the use of such compositions for rails. It has been found that the air-hardening properties of chromium give a refinement of grain, even with large masses air cooled; this might be expected to improve greatly the structure at the center of rail head over that common to plain carbon-steel rails.

Chromium as an alloying metal in high-speed steels has long been indispensable, many standard brands containing it in quantities up to 4.5 per cent. Satisfactory tools have been made with about 18 per cent. chromium and no other alloying elements but vanadium. There has been much controversy as to whether tungsten or chromium is chiefly responsible for the self-hardening properties of high-speed steels.

With the exception of the plain nickel steels, most of the well-known engineering steels are alloys containing chromium and one or more alloying elements in addition to the carbon, such as nickel, vanadium, molybdenum, etc. Their properties are fully described in several standard references.[3]

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[2] Aitchison: Chromium Steels and Irons. The Institution of Automobile Engineers (1921).

The place of so-called "stainless steel" in cutlery has become well established. The composition is generally 12 to 13.5 per cent. chromium with 0.3 to 0.4 per cent. carbon.[4] Since the development of the stainless steels, stainless or rustless irons have been experimented with. With from 9 or 10 per cent. up to 14 or 15 per cent. chromium and less than 0.10 per cent. carbon, this class of alloys does not require heat treatment to develop stainless qualities.

Alloys with 12 or 15 per cent. or more chromium and the balance iron and one or more elements, as nickel, cobalt, tungsten, molybdenum, etc., have been brought to quite a useful stage of development in a number of compositions. Many of these have received well-known trade names in this country and have been extensively used for electrical resistance, high-temperature resistance, tools, etc. Examples that may be cited are Nichrome, Rezistal, and Stellite.

Alloys with 20 Per Cent. or More Chromium

Because the alloys already mentioned have been rather extensively discussed in the literature, it is proposed here to present a few facts on commercial alloys containing above 20 per cent. chromium and discuss briefly their useful engineering qualities. Although this investigation has extended over a number of years, some points are still of a controversial nature because of a lack of knowledge of the behavior of metals and alloys in general from the standpoint of physical chemistry. The information is therefore presented largely from the standpoint of engineering interest.

It may be well to picture alloys between 20 and 30 per cent. chromium as having, in a general way, similarities to the plain carbon steel-iron series. The alloys low in carbon resemble steels in being forgeable; they gradually shade over into the white-iron or chilled-iron alloys in the high-carbon range. While the similarity is somewhat distorted, the two series are in many qualities comparable. No graphite will be found, however, in the iron-carbon-chromium series of this range. For convenience in discussion, the alloys may be divided into three purely arbitrary groups, according to carbon content: Low-carbon, up to about 0.5 per cent.; medium-carbon, 0.5 to 1.5 or 2 per cent.; high-carbon, 2 per cent. and more.

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[3] Hand Books, The Am. Soc. Automotive Eng.

C. N. Dawe: Chromium and Nickel-molybdenum Steels. *Iron Age* (1922) Mar. 16.

R. D. Report No. 55, Woolwich Arsenal (1923).

[4] R. G. Hall: Stainless Steels and the Making of Cutlery. *Trans.* A. S. S. T. (1922).

Stainless Steels. *Iron and Coal Trade Rev.* (1921) Oct. 28.

Stainless Steel—Its Properties and Some of Its Engineering Apparatus. *Iron and Coal Trade Rev.* (1923) Mar. 9.

Chemical Characteristics

The outstanding chemical property is resistance to oxidation at high temperatures, or in fact to any oxidizing action. Exposure to air at 1000° C. will cause, on bright samples, a light tarnish, or at most a thin and toughly adherent scale that acts as a protective coating that arrests or prevents further attack. The mechanism of oxidation has been discussed elsewhere[5] on the basis of the quality of the oxide formed and, as has been pointed out by Bancroft, the property of non-corrodibility is a function of the quality of the scale or product formed. It has been noted in tests that the maximum amount of scale is formed early and that indefinite exposure does not seem to increase materially the percentage of scale in many cases.

Carbon content does not seem to influence the resistance to oxidation. On the other hand, increase of chromium from 16 or 18 up to about 20 or 22 per cent. increases the oxidation resistance to an appreciable extent. Above about 22 per cent., the relative improvement in oxidation resistance with increase in chromium is not so marked. This effect is believed to be due to the tendency to form a tougher and more adherent scale if alloys are above 20 per cent. chromium than if they are below 20 per cent.

All of the alloys are remarkably resistant to nitric acid in all concentrations and temperatures. They are not resistant to hydrochloric acid and are not recommended for resistance to sulfuric acid, more particularly at temperatures much higher than 50° C.

Electrical Properties

The specific electrical resistance of the alloys is high and the temperature coefficient is large. An alloy with 0.27 per cent. carbon, 23.49 per cent. chromium gave the following data: 17.3 microhms per cubic inch at room temperature and 2.6 times this value at 1100° F. This test was made on 0.025-in. wire.

The permeability of the alloys is rather low, but is higher the softer the alloy. In general, the permeability is less than one-half that of cast iron. Carbon lowers the permeability to some extent but manganese lowers it greatly.[6]

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[5] Philling and Bedworth: Mechanism of Metallic Oxidation at High Temperatures. Bureau of Standards *Sci. Paper* 118 (1922).

Philling and Bedworth: The Oxidation of Metals at High Temperatures. Inst. of Metals, London (1923).

[6] F. M. Becket, U. S. Patent 1333151 (1920).

Physical Properties

The mean specific heat between room temperature and 200° C. was found to be 0.15; between room temperature and 600° C. it was 0.165 on the same sample.

The thermal coefficient of expansion is not far from that of steel, being about 0.0000066 per degree F., in the rolled alloy. The relation is linear between 0 and 1500° F.

The thermal conductivity between room temperature and 100° C. is about one half that of soft steel, as shown by some rather approximate tests.

Hot- and Cold-working Properties

Low-carbon Alloys

Alloys with 0.3 to 0.5 per cent. carbon have a fracture in the cast condition which depends on the rate of freezing. A casting may have a large, shiny or coarse-grained fracture with single facets extending nearly across a thin section; some grains will show a cubic crystallization under a hand glass, like galena. The transverse strength of such metal is likely to be low. The same metal, cast under different conditions and more rapidly cooled, will show a fine-grained fracture like a well-forged low-carbon steel. This refining of grain is naturally attended by an improvement in strength and these castings will be quite tough. On the other hand, in one or two instances a low-carbon fine-grained casting was brittle; investigation showed this alloy to be full of minute inclusions, in other words the metal would correspond to a "dirty steel." It is believed that these non-metallic inclusions act as crystallization neuclei and bring about a fine-grained structure but one without good mechanical strength or resistance to shock stresses.

The low-carbon alloys are readily forgeable and will stand forging through nearly the same temperature range as a steel. Because of their greater hardness and toughness at elevated temperatures, they require a higher low limit for hot working than do the steels—namely, about 800° to 850° C. They work best at 950° to 1050° C., depending on the operation. Plate and rod have been rolled on a commercial scale with practically the same procedure as for steel, the factors affecting production being about the same as for ordinary medium-carbon steel. Wire drawing is quite practicable but has certain differences from steel practice. More care seems to be required in the early draws, but after a few draws have been made the drafts can be considerably increased.

Hot pressing has been done on an experimental scale and has demonstrated its practicabilty and there are indications that cold stamping is also feasible. Seamless tubing has also been forged; the process appears to be entirely commercial.

High-carbon Alloys

The difficulty of hot and cold working increases with increase of carbon content; the workability seems to depend to a much greater degree on the carbon content than on the chromium. It is possible to forge alloys with 2.5 or 3 per cent. carbon and 26 to 28 per cent. chromium, but the forging is difficult and the metal must be worked quite hot, say about 1110° C. Castings are readily made. Shrinkage allowance is about ¼ in. per foot. In the range of 0.6 to 1.5 or 1.75 per cent. carbon, the cast alloys are machinable or may be made so by heat treatments. Brinell hardness varies, with carbon content, from 200 to 600 in the cast condition.

STRUCTURAL CONSTITUTION

Low-carbon Alloys

The structure of the very low-carbon alloys, below about 0.20 per cent. carbon, consists of the iron-chromium solid solution with the usual grain structure found in any metal or alloy with the same degree of previous work. It is believed that the limit of solubility of carbon in the cast alloys is about 0.20 or 0.22 per cent. carbon. Alloys with 20 per cent. chromium or more will not show the presence of any structurally free carbide with less than 0.2 per cent. carbon even though given various heat treatments. That the solid solution always holds some carbon dissolved accounts, in part, for the fact that the hardness of these alloys may not be brought down to equal that of annealed plain carbon steels of like or even higher carbon content. On the other hand, the simple iron-chromium solid solution, if it were possible to secure it free from

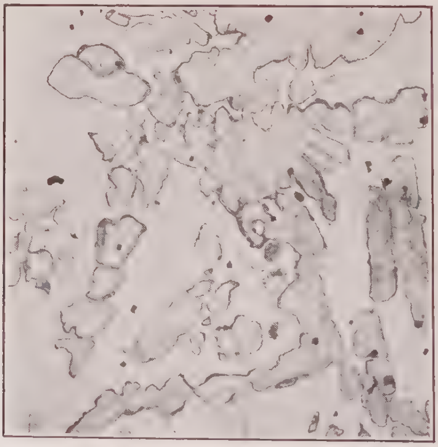

Fig. 1.—Cast alloy containing 0.17 per cent. carbon. × 100.

carbon, would be much harder than ferrite, as accounted for by the theory of the hardening of solid solutions.[7]

Fig. 1 illustrates the structure of an alloy containing 0.17 per cent. carbon, in the cast condition. This shows irregular shaped grains in a white structureless ground mass which, in turn, is broken by grain boundaries. Probably in freezing there was some microscopic segregation of carbon, which darkens certain grains due to differences in behavior toward the etching reagent. The metals in the low-carbon ranges are soft, even in the forged and quickly cooled conditions, although harder than plain carbon steels. The Brinell hardness number of the alloys with 0.15 to 0.25 per cent. carbon is near 175, or about equal to a 0.57-per cent. steel in the "dead soft" state.

Medium-carbon Alloys

Increase of carbon content is manifested in the structure by the appearance of carbide particles. These are small white bodies in a matrix of solid solution, in the grain boundaries in cast alloys and scattered throughout the granular solid solution after forging. The appearance of the cast structure is depicted in Figs. 2 and 2*a*. The alloy contains 0.46 per cent. carbon and 21.91 per cent. chromium. It has not been found possible to absorb the carbides by heating, as in the case of ordinary steels, except to a minor degree. High-carbon alloys may be heated to near the melting point and quenched, when the carbides will show a partial solution. The grain, as shown by fracture, is not broken up by annealing, as in the case of plain carbon steels; this is

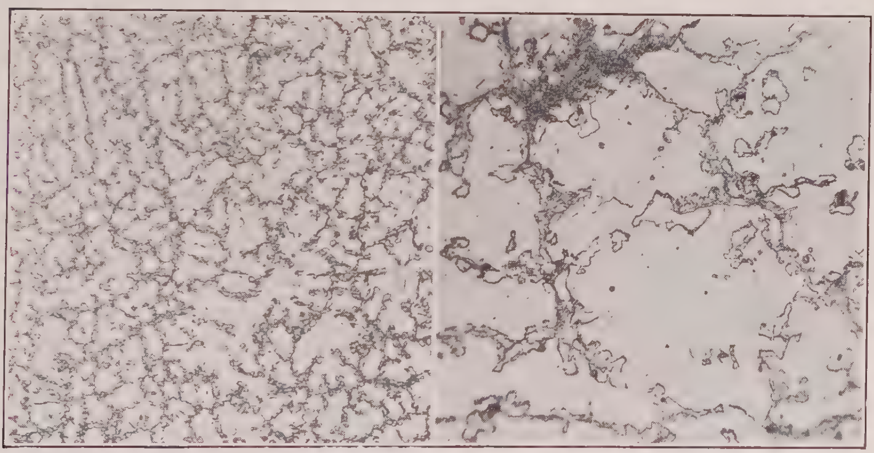

FIG. 2.—CAST ALLOY CONTAINING 0.46 PER CENT. CARBON AND 21.91 PER CENT. CHROMIUM. × 100.

FIG. 2*a*.—SAME AS FIG. 2. × 425.

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[7] Walter Rosenhain: Hardness of Solid Solution. *Chem. and Met. Eng.* (1921) **25,** 243.

because chromium slows up the thermal reactions of the alloys to which it is added in any appreciable degree. It is believed that the heat treatment effects in the high-chromium alloys would conform more closely to those encountered in steels if the time element were considerably increased.

Fig. 3 shows a characteristic appearance of a forged alloy with 0.36 per cent. carbon and 29.46 per cent. chromium held at 600° C. for 24 hr. and water quenched. The carbide particles are easily distinguished as small rounded white bodies.

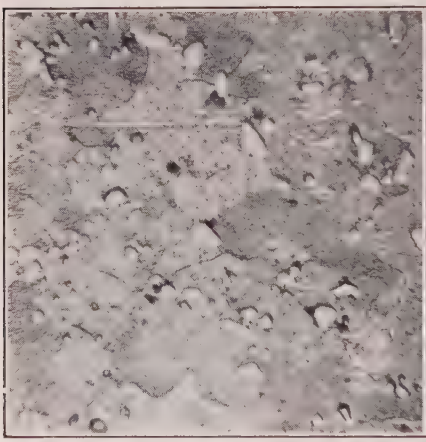

Fig. 3.—Forged alloy containing 0.36 per cent. carbon and 29.46 per cent. chromium. × 100.

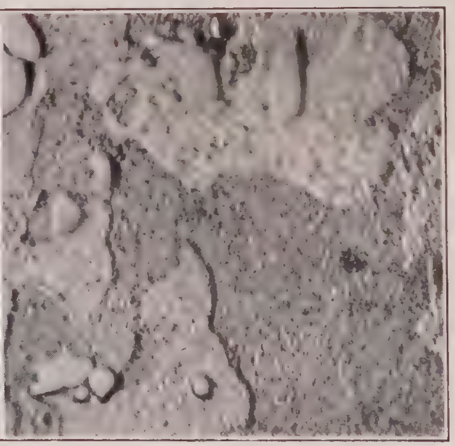

Fig. 3*a*.—Same as Fig. 3. × 1000.

High-carbon Alloys

The solid solution of iron and chromium containing carbon is progressively lessened in relative quantity as the carbon content of the alloy increases. The carbide present in the low-carbon alloys occurs in networks that gradually thicken into lace-like patches with increasing carbon; Fig. 4 illustrates the structure of an alloy with 0.98 per cent. carbon and 28.87 per cent. chromium as cast; with forging, the network is broken and the carbides scattered into small rounded particles. With about 1 to 2.5 per cent. carbon, the carbides appear as dendrites in the cast alloys; Fig. 5 shows a cast alloy with 2.15 per cent. carbon and 27.04 per cent. chromium. The solid solution appears here as the white dendrites; the eutectic forms a characteristic structure, the foliations alternating with small patches of cellular form. Further increase in carbon suppresses the structurally free solid solution until, at about 2.75 per cent. carbon, the eutectic structure of solid solution and carbide occurs as depicted in Fig. 6. Alloys of this composition have a structure analogous to plain carbon steel of eutectoid composition. As increasing carbon in the latter gradually suppresses the structurally free solid solution (ferrite), until at 0.85 per cent. carbon the whole of the structure is composed of carbide and solid solution, similar changes take place in the iron-chromium-carbon alloys of this range of composition. The physical properties, also, are somewhat analogous, combining great hardness and toughness. Alloys with about 2.75 per cent. carbon have a marked sheen on the fracture and the polished and etched samples have the appearance of watered silk and possess a pearly luster. The carbides

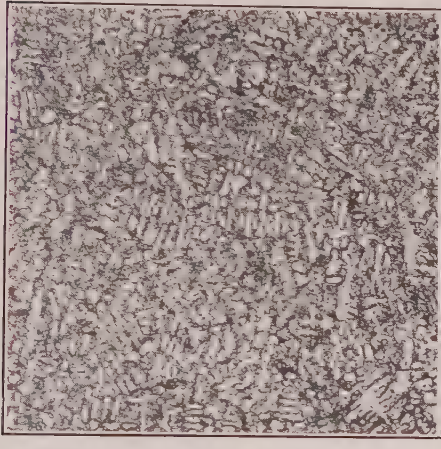

Fig. 4.—Cast alloy containing 0.98 per cent. carbon and 28.87 per cent. chromium. × 100.

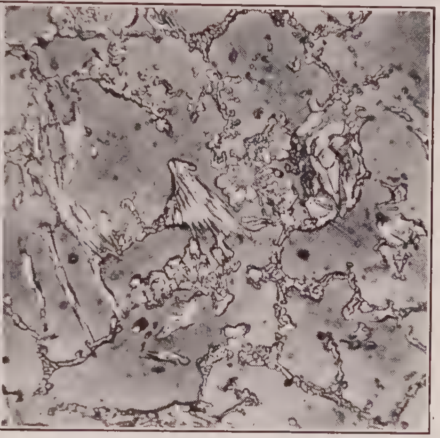

Fig. 4*a*.—Same as Fig. 4. × 425.

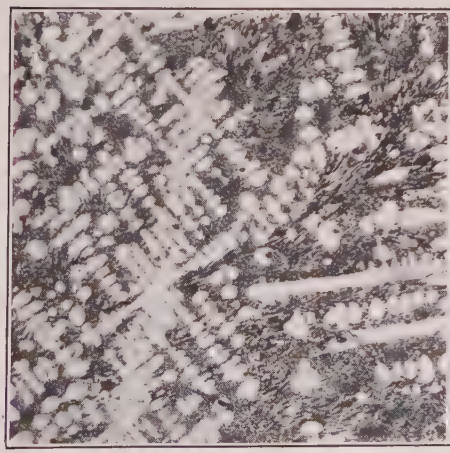

Fig. 5.—Cast alloy containing 2.15 per cent. carbon and 27.04 per cent. chromium. × 100.

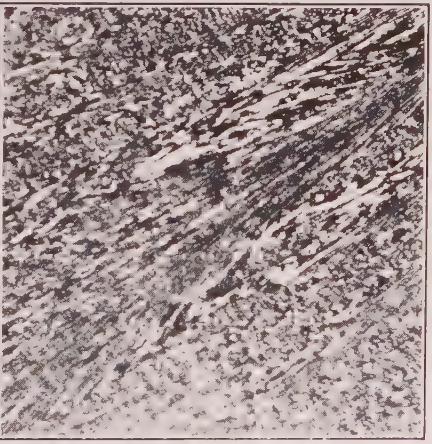

Fig. 6.—Cast alloy containing 2.75 per cent. carbon. × 100.

have a characteristic foliated structure like palm fronds. This interlacing between the tough solid solution and the extremely hard carbides probably accounts for the wear resistance of this type of alloy.

Just as a new structural constituent appears above 0.85 per cent. carbon in the pearlitic plain carbon steels, *i. e.*, free cementite, the analogy may be maintained here. Structurally free carbides are seen as needlelike plates or in small polygonal bodies in Fig. 7, an alloy with a little over 3 per cent. carbon and 27.35 per cent. chromium. The field shows the eutectic ground mass in the characteristic palm leaf and cellular form, with the interspersed excess carbide crystallites. It is believed that the composition of the carbides changes as the carbon content increases. Murakami has discussed three carbides, and differentiated them by magnetic analysis and by their behavior toward his etching reagent.[8] These he calls alpha, beta, and gamma double carbides and estimates their compositions as follows:[9]

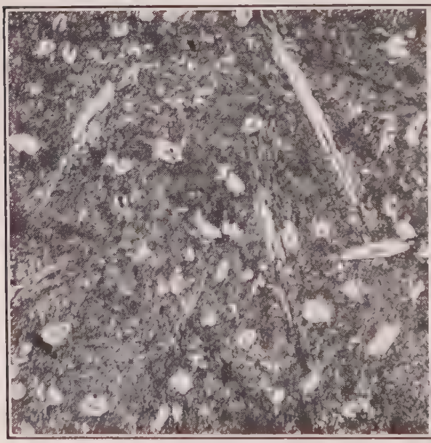

Fig. 7.—Structurally free carbides in alloy steels containing over 3 per cent. carbon and 27.35 per cent. chromium. × 100.

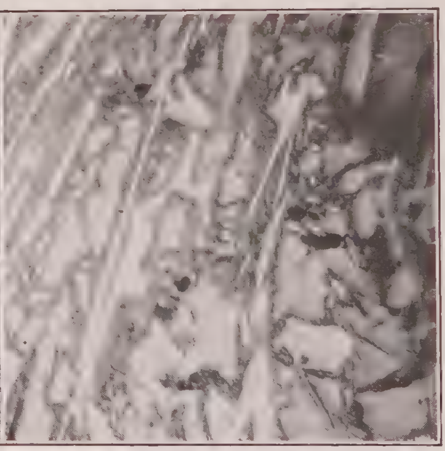

Fig. 9.—Cast alloy containing 2.88 per cent. carbon and 27.35 per cent. chromium. × 425.

	Per Cent. Chromium	Per Cent. Iron	Per Cent. Carbon
Alpha double carbide, $(Fe_3C)_{18}.Cr_4C$........	6.01	87.40	6.59
Beta double carbide, $(Fe_3C)_9.Cr_4C$.........	11.30	82.18	6.52
Gamma double carbide, $(Fe_3C)Cr_4C$........	52	42	6

According to his scheme, the alloys below 1 per cent. carbon consist of the solid solution of iron and Cr_4C (the latter dissolving chromium);

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[8] Murakami's reagent is composed of potassium ferricyanide and potassium hydroxide 10 gm. each, dissolved in 100 c.c. of water.

[9] Murakami, *loc. cit.* 249.

for higher carbon ranges, the alloys contain possibly some Cr_4C but the bulk is the gamma carbide.

Differential chemical analysis, as by the method of Arnold and Read, has not been attempted so far in this investigation, hence no formulas are assigned to the structural constituents noted. Our work has indicated, however, that the carbide changes materially in behavior toward etching reagents and physical form as the carbon increases in the alloys with a given chromium content.

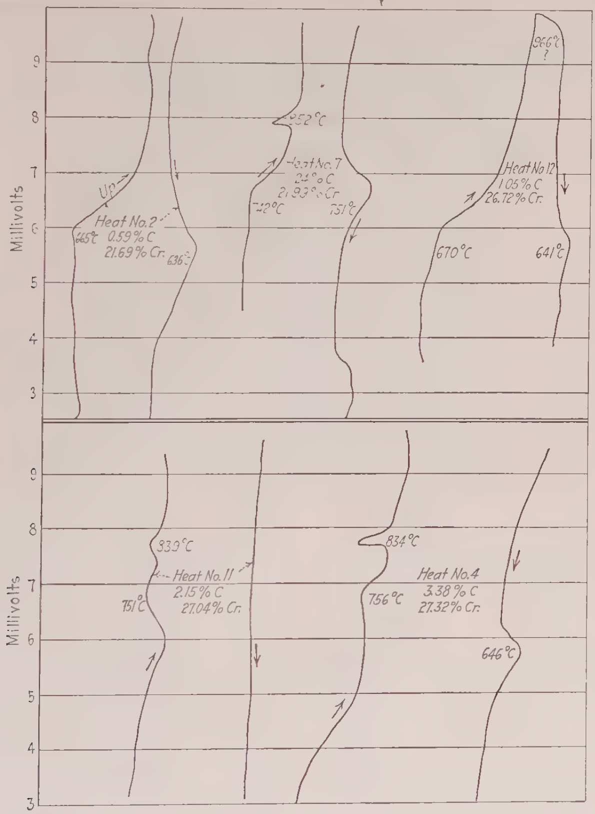

Fig. 8.—Temperature arrests in chrome-iron alloys; curves made by Leeds & Northrup transformation-point apparatus.

Heat Treatment

Thermal Arrests

The alloys with around 0.3 per cent. carbon have a feeble thermal critical point at about 575° C. on cooling. Murakami found no thermal arrest in an alloy with 27.5 per cent. chromium and 0.28 per cent. carbon, but did find a magnetic arrest point A_2, the position of which was not affected by the initial temperature. Some results of our own tests are shown in Fig. 8; these curves were made by Leeds & Northrup transformation point apparatus.

Thermal Treatment

Alloys up to about 0.5 per cent. carbon cannot be hardened by heating and quenching although some alloys may be made brittle as described and this treatment will show as a slight increase in scleroscope hardness. Higher carbon, 0.7 per cent. or more, may be readily hardened by heating to 850° C. or above, and quenching. With higher carbon alloy (1.5 per cent.), the quenching may be omitted and hardening obtained by air cooling. Softening will result from heating to about 800° C., slow cooling to 600° C., then more rapid cooling.

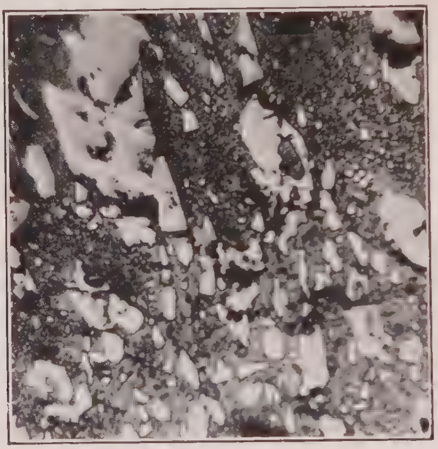

FIG. 10.—SAME AS FIG. 9 AFTER HEAT TREATMENT. × 425.

In the experimental work on heat treating the alloys in the lower carbon ranges, that is to harden or soften them, in general it has not been possible to affect to any extent the appearance of the alloy under the microscope. The solid solution and the carbides do not change their appearance to any material degree.

In the carbon ranges above 2.5 per cent. carbon, the structural changes in heat treatment are more defined. Fig. 9 shows the structure of an alloy with 2.88 per cent. carbon and 27.35 per cent. chromium. As cast this alloy had a Shore number of 58. After heating to 1073° C. for 1 hr., cooling very slowly (112 hr.) to below 600° C., and then more rapidly (3 hr.) to room temperature, the Shore number dropped to 35 and the structure was altered to that shown in Fig. 10. The carbides have lost their long spine-shaped appearance and are more rounded.

Considering the lower carbon alloys, the question naturally arises, what is the nature of the solid solution? By definition, this should be termed austenite, which is a solution of carbon or carbide in gamma iron. The alloys under some conditions are magnetic and under others only feebly magnetic or entirely non-magnetic. The structural changes largely associated with the transformations of austenite through heat treatments are not shown in these alloys, namely the formation of martensite and other transition products by annealing. We are then compelled to consider that the austenite is not affected by heat treatments, even though they extend over several days, or that this austenite does not have the properties of the solid solution met with in the iron-carbon alloys, or that this is not an austenitic ground mass. We are inclined to believe that the bulk of the alloy consists of a solid solution containing dissolved carbon or carbide of such great stability that it maintains its entity through the numerous changes undergone during heat treatment. There are objections to this hypothesis, however, because of the change in physical properties brought about by heat treatment, which would seem to require more outstanding changes in the appearance of the alloys. We have not found the usual forms of martensite, troostite, sorbite, and pearlite in these alloys, although several investigators have mentioned their occurrence in somewhat similar compositions. The metallographic changes occurring on heat treatment are discernible in the higher carbon alloys, in particular the appearance of the carbide bodies. These are rounded up somewhat and coalesced by high temperature anneals if given sufficiently long treatment, say 24 hr. or more, for the changes as depicted above to take place.

PHYSICAL PROPERTIES

Transverse Strength

The transverse strength of castings has been found to vary considerably with the method of casting and the composition of the alloy. As the compositions used in castings have been higher carbon as a rule, the amount of deflection is not high, being only a few hundredths on the standard arbitration bar. Low-carbon alloys show less transverse strength but an increase in deflection. The following tabulation will indicate the general properties tested transversely:

SPECIMEN	SIZE	DISTANCE BETWEEN SUPPORTS, INCHES	BREAKING LOAD, POUNDS	DEFLECTION, INCH
0.26 per cent. carbon, 27 per cent. chromium, cast at high temperature	¾ in. sq.	12	1,650	0.2
Ditto, cast at low temperature	¾ in.	12	3,350	0.53
Carbon 1.16 per cent., chromium 21.17 per cent., average five tests	1⅛ in.	12	8,370	none
Carbon 2.63 per cent., chromium 26.72 per cent.	¾ in. sq.	4	12,510	none

Tensile Tests

The physical properties of one forged alloy are shown in Fig. 11 which depicts the results of tensile tests on an alloy containing 0.53 per cent. carbon, 24.45 per cent. chromium, 0.39 per cent. silicon, and 0.10 per cent. manganese. The metal for these tests was rolled to ¼-in. plate from 5 by 5-in. ingots, the plate being sheared to strips, which were then side milled to give flat tensile specimens; the flat faces were not machined. They were then drawn at the various temperatures indicated, for 30 min. for each temperature. As the carbon content is raised, the tensile strength increases and ductility decreases.

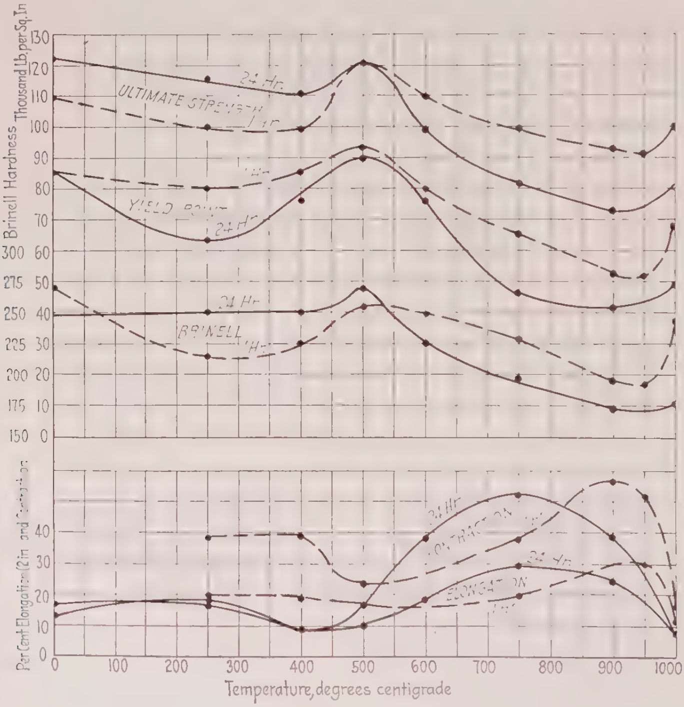

Fig. 11.—Strength and ductility of low-carbon, chrome-iron alloy; specimens air cooled after 1 and 24 hr. at temperature indicated.

The results of some tensile tests made at high temperatures are shown in Fig. 12. Comparison of the results with those obtained by investigators for various steels shows a high tensile strength at elevated temperatures. Experiments are being carried on to determine the effect of several additional elements on the tensile properties at high temperatures, such as cobalt, vanadium, uranium, copper and molybdenum. Increase of carbon raises the tensile strength, that is, the knee of the curve characteristic of the temperature-tensile tests of steels is pushed toward the right. The property of high tensile strength at elevated temperatures is useful and important for an oxidation-resistant alloy and it is becoming of increasing importance, especially in the oil-cracking industry. It is thought that the useful properties of alloys of this nature are as yet only partly realized.

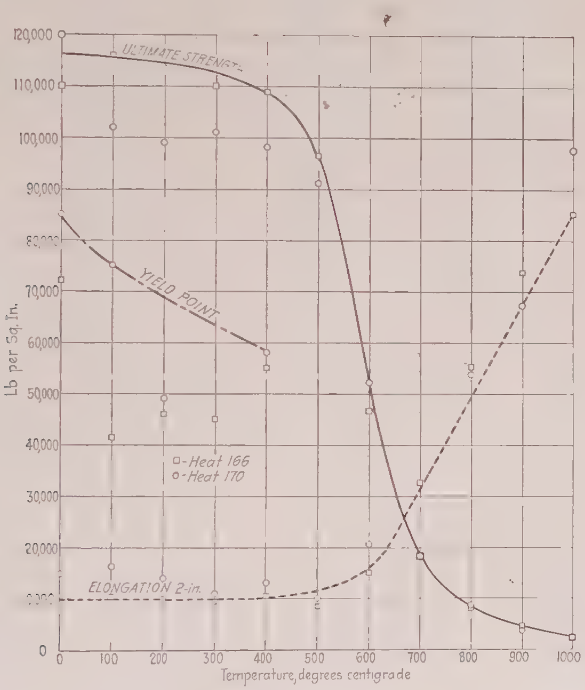

Fig. 12.—Results of high-temperature tensile tests.

Tensile properties of cold-drawn wire without subsequent treatment have shown breaking stresses of 112,000 lb. per sq. in. and 11.5 per cent. elongation in 2 in. for 0.045-in. wire, and 180,000 lb. per sq. in. with 1.5 per cent. elongation on 0.021-in. wire. Both wires were low-carbon alloys.

Miscellaneous Properties

While the outstanding property of the high-chromium alloys has been their stability under oxidizing conditions at high temperatures, they have other useful characteristics. Because of the interest in rolled alloys resistant to chemical action, for use in the canning and preserving industries, tests were made on a small sheet-metal container, the various mixtures of solutions being cooked for 4 hr. or more, after which the respective mixtures were analyzed for iron and chromium. Table 1 indicates the results obtained.

The tests were made on a small rectangular box about 5 by 4 by 3 in., made by welding 16-gage sheet. The weight of the box was 1.5 lb. and its composition about 0.3 per cent. carbon with 27 per cent. chromium. Even prepared mustard, which is an active tarnishing agent on many alloys, is practically without effect.

In many localities the action of acid mine water is quite destructive of equipment. Some extensive tests[10] recently made have demonstrated the practical immunity of this alloy toward both weak and strong acid mine water. The sheet tested was hot-rolled, low-carbon alloy with no subsequent heat treatment.

Table 1

Fruit, Vegetable, or Acid	Weight or Volume Used	Time, Hours	Grams Extracted		Per Cent. Extracted	
			Iron	Chromium	Iron	Chromium
Rhubarb	2 lb.	4–5	0.0100	0.0010	0.0015	0.0002
Tomatoes	2 lb.	4–5	0.0055	trace	0.0008	trace
Apples	3 lb.	5	0.0040	trace	0.0006	trace
Prepared mustard	1½ lb.	5	0.046	0.0035	0.007	0.0005
10-per cent. acetic	500 cc.	5	0.024	0.0045	0.0036	0.0007
1-per cent. acetic	500 cc.	10	0.0021	trace	0.0003	trace
1-per cent. tartaric	500 cc.	4–5	0.008	trace	0.0012	trace
1-per cent. oxalic	500 cc.	5	0.010	0.0025	0.0015	0.0004
1-per cent. formic	500 cc.	5	0.012	0.0012	0.002	0.0002
1-per cent. malic	500 cc.	5	0.0045	0.001	0.0007	0.0002
1-per cent. citric	500 cc.	5	0.0035	0.0008	0.0005	0.0001

The high-carbon alloys combine a peculiar toughness with high mineral hardness. Some tests were made using an alloy with 2.72 per cent. carbon and 26.41 per cent. chromium for bottom plates in a 9-ft. pan mill. Working on hard minerals, such as zirkite, electrically sintered magnesite, etc., the iron-chromium alloy greatly excelled chilled cast iron in wear resistance and equalled manganese steel. Under conditions of wear combined with shock, chilled iron has a tendency to spall; as soon as the white chill is penetrated, the metal fails rapidly. Manganese steel, on the other hand, while of low initial hardness, will harden under the effect of work. The chromium alloy seems to possess qualities of both the foregoing, a combination of high mineral hardness with a degree of toughness found in few alloys.

When considering the diverse requirements of the mining and metallurgical industries for materials of high resistance to various forms of attack, it is impossible to recommend any given composition as a cure-all; each case must be decided in the light of the particular service conditions. Among the many alloys on the market, some of which have given many years of satisfactory service, it is possible to fit most of the conditions up to 1000° C. or more. One alloy may be forgeable but is exceeded in strength at high temperatures by an alloy that is non-forgeable; still another alloy may excel in acid resistance and fail in oxidation at more than red heat, and so on. To date, no one combination has been found to possess the maximum merit under all conditions of service. It is believed by the writer, however, that the general possibilities of numerous alloys in the high-chromium ranges are not yet sufficiently appreciated by the engineering profession. It is also believed that many technical processes can be carried out at a higher efficiency by the installation of equipment which will, in general terms, permit the more efficient transfer of heat, the boosting of temperature and pressure, and the less frequent necessity of shutdowns for repairs. Already installations of the high-chromium or chromium-nickel alloys have been made, or are being discussed, in the walls and hearths of roasters, for heat interchangers of various types, in oil-cracking stills, in the manufacture of ammonia, fuel-burning equipment, chemical manufacturing plants, and in lines where we have not conceived of the use of metals or alloys.

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[10] Selvig and Enos: Corrosion Tests on Metals and Alloys in Acid Mine Waters from Coal Mines. Bureau of Mines *Bull.* 4 (1922).

Thanks are due for assistance in preparing this paper, to F. M. Becket, chief metallurgist of the Union Carbide and Carbon Corpn., whose interest in and knowledge of the chromium alloys has inspired extensive research work for a number of years, particularly in the high-chromium ranges; also to H. S. George and other members of the staff of this laboratory, especially S. M. Norwood who has personally supervised much of the detail work discussed here.

Some Effects of Zirconium in Steel

By Alexander L. Feild,* M. S., New York, N. Y.

(Canadian Meeting, August, 1923)

This paper contains an account of experimental results obtained in an extensive investigation of steels containing zirconium. There recently appeared[1] an announcement by F. M. Becket, at whose suggestion and under whose general direction this investigation was made, a statement of the outstanding features of the work and a brief explanation of the factors that led to its inception. It is not possible, in a paper of this character, to record all of the operating and test data collected, which includes the production of more than 350 heats of steel. The difficulty of a complete presentation is increased by the fact that several lines of experimentation are too incomplete to warrant publication of the results at the present time.

In addition to the methods of manufacture and test employed in the work, there is described experimental evidence on which are based certain definite conclusions regarding the remarkable effects brought about by the presence in steel of relatively small percentages of zirconium. When added to a molten steel bath, zirconium combines chemically with dissolved oxygen, nitrogen, and sulfur, in the order named, and is capable of neutralizing the embrittling effect of phosphorus in whole or in part; although in the case of phosphorus there is no evidence so far of actual chemical combination. These effects are not in the nature of suppositions supported by theory, nor is their existence deduced merely on the basis of analytical evidence. In each phase of its relation to or reaction with the four impurities mentioned, zirconium affects beneficially the mechanical properties of the steel to which it is added, in a manner which it is one of the purposes of this paper to explain in terms of a tentative working theory.

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* Research Metallurgist, Electro Metallurgical Co.

[1] *Iron Age* (1923) 1321–3.

METHODS USED IN EXPERIMENTAL STEEL MANUFACTURE

Electric Steel Furnace and Its Operation

The steels were made by melting cold scrap steel in a basic-lined electric furnace of the single-phase direct-arc type. The usual melting charge consists of 200 to 275 lb. of light scrap, together with the proper amounts of lime and silica rock (7 lb. of lime and 5 lb. of silica rock frequently gave a satisfactory slag). From 20 to 40 min. after melting is completed, the bath is in proper condition to permit the tapping off of this first, oxidizing slag. The surface of the metal bath is scraped as free as possible from slag, which can be done very efficiently because of the relatively low melting point and low viscosity of the latter. The second, or finishing, slag is then immediately built up, using approximately 10 to 15 lb. of lime and 1½ to 4 lb. of silica rock.

From this point on, the practice depends entirely on the composition of the steel and, chiefly, on the carbon content desired. Normally, the practice is to close the door of the furnace, plug up around the door with fireclay, and, by the judicious addition of coke* screened to approximately ¼ in. by 8 mesh, to bring about the formation of the well-known "carbide" slag. When carbide formation begins, the carbon content of the bath is apt to be close to 0.18 per cent. Under a continuance of these conditions, even though no more coke is added, the carbon content of the bath rises. At the same time, sulfur is eliminated from the metal bath and enters the slag with appreciable rapidity.

As the first carbide slag conditions are obtained when there is about 0.18 per cent. carbon in the steel bath, and as the subsequent high-carbon ferromanganese addition carries with it 0.04 to 0.06 per cent. carbon, it becomes difficult, at least in an electric furnace of this capacity, to produce a steel, with all silicon additions made in the ladle, of less than 0.22 to 0.25 per cent. carbon, unless a practice resembling basic open-hearth practice is adopted. In the present work, these low-carbon steels have been produced by controlling the furnace conditions during the finishing period in such a way as to approach but not realize the formation of a true carbide slag. One method is to seal the furnace, not immediately after the second slag is built up but toward the end of the run, and to add only sufficient coke to eliminate excess oxide from the slag. With proper precautions, it is possible to cover the range of carbon content between 0.08 and 0.23–0.25 per cent.

In all cases, the metal is tapped from under the carbide, or high-lime, slag. When the slag does not solidify rapidly enough during the short interval required to raise the electrodes, lime is added just prior to tapping and to that section of the bath in front of the pouring spout. Tapered ingot molds were made by a foundry that specializes in the manufacture of cast-iron ingot molds. The ingots were top poured with the small end up, removed from the mold while at a red or yellow heat, and allowed to cool in air to atmospheric temperature.

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* From 12 to 24 oz. is usually required.

The details of ladle and ingot practice depend on the weight of metal required for test purposes and on whether the ingots were to be forged or rolled. In the case of steels designated for heat treatment, two 150-lb. ladles were tapped in quick succession from each heat, one ladle (ladle A) being treated with a silicon-zirconium alloy, the other (ladle B) with an equal quantity of silicon as 50-per cent. ferrosilicon. Each ladle was teemed into a mold 20 in. deep, 1½ in. wall thickness, 5¼ by 5¼ in. in cross-section at the bottom and 4¾ by 4¾ in. at the top. The ingots weighed 110–120 lb. and were, as a rule, forged to 1-in. round bars.

When the steels were to be tested in the pearlitic state, the general practice was to tap three 50-lb. ladles from each heat, treated (ladle A) with a silicon-zirconium alloy containing approximately 10 per cent. zirconium and 70 per cent. silicon, (ladle B) with an equal amount of silicon as 50-per cent. ferrosilicon, and (ladle C) with a silicon-zirconium alloy containing 33–38 per cent. zirconium and 44–50 per cent. silicon, also in amount corresponding to the same addition of silicon as was used in ladles A and B. For the production of steels to be forged, these 50-lb. ladles were teemed into molds 14 in. deep, 1 in. wall thickness, 3¼ by 3¼ in. in cross-section at the bottom and 2¼ by 2¼ in. at the top, inside dimensions. When the ingots were to be rolled, they were cast in slab molds, 8 in. deep, 1¼ in. wall thickness, 2⅞ by 6⅛ in. at the bottom and 2⅝ by 5⅞ in. at the top, inside dimensions. Both types of ingots teemed from the 50-lb. ladles weighed approximately 30 to 34 pounds.

Ferro-alloy Practice

All of the steels subjected to mechanical tests were of the "piping" or "killed" type, with additions of ferrosilicon, or of silicon-zirconium alloy, made in the ladle. Except in a few instances,* additions of high-carbon ferromanganese were made to the furnace, the weight of the addition being based on the known weight of the charge. Weights of ladle additions are based on the rated capacity of the ladle, but in all cases where values for percentage recoveries or for percentages of added silicon or zirconium are given, they are based on the actual weight of metal tapped into the various ladles as obtained by weighing both ingot and excess metal. In this way corrections are made for unavoidable differences in ladle weights caused by slight over or underpouring. Ladle additions were weighed on a torsion balance to the nearest 1/16 oz. The silicon-zirconium and ferrosilicon were, in all cases, crushed to small lumps (1/4–3/8 in.) of roughly uniform size. They were added, in a paper envelope, to the ladle when the bottom of the latter was covered by a couple of inches of the issuing stream of metal. The steel was held in the ladle until it reached the proper temperature for pouring, the measured length from the moment when the ladle was filled being 1 to 3 min., in the case of the 120-lb. ladles, and 1/2 to 1 1/2 min. in the case of the 50-lb. ladles. During this time the surface of the metal was carefully scraped free of slag, practically all of which came from the oxidized ladle additions and the fireclay ladle lining. Little or no furnace slag followed the metal during tapping into the ladle. Heats that gave ingots showing any indication of "rising" or "raising" in the ingot molds were considered off-grade and the ingots laid aside. No additions of aluminum were made.

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* Of the steels reported here, only those of heats 105, 107, 115, 120, and 121 were made with ladle additions of high-carbon ferromanganese.

As it was one of the purposes of this investigation to determine the effect of zirconium in steels containing phosphorus and sulfur up to percentages far exceeding the limits imposed by existing specifications, it was necessary at times to modify the ordinary practice. One of the commercial advantages of the electric steel furnace is its ability to eliminate both phosphorus and sulfur from the initial charge—the first-mentioned element during the melting and oxidizing period of the run, the second during the finishing or reducing period—so that it became necessary to adopt certain standardized procedures to obtain these elements in the desired amount in the finished steel, without adversely affecting the state of deoxidation of the bath.

In the production of the high-phosphorus steels described later, the calculated amount of ferrophosphorus in lump form was added to the furnace 5 to 10 min. before tapping. Under conditions of carbide-slag formation, the recovery of phosphorus in the metal bath was close to 100 per cent. Where such slag conditions did not prevail, percentage recoveries fell, in some cases, to 75 per cent. even under satisfactory furnace conditions. The recovery of phosphorus in the case of high-phosphorus low-carbon heats was a good index of the degree of deoxidation of the slag and steel bath. Off-grade low-carbon heats showing low phosphorus recovery were not rare, but in all such cases the ingots were discarded as being undesirable for test purposes.

The production of steels in a manner capable of meeting a designated high-sulfur specification presented another problem in electric-furnace practice. Control of recovery of sulfur, added to the furnace as iron sulfide in lump form, required some care, especially when carbide-slag conditions strongly and definitely prevailed during the finishing period, as in the case of steels containing more than 0.23–0.25 per cent. carbon.

Experience served as the best guide, the expected recovery being deduced from previous log sheets. In the production, for instance, of a 0.29-per cent. sulfur steel of 0.37-per cent. carbon content, the percentage recovery of sulfur was 50 per cent. when the iron sulfide was added 5 to 10 min. previous to tapping. This was about the lowest recovery obtained. In most cases, the high-sulfur steels were not made with such a high percentage of sulfur and as the recovery increased with decreasing weights of sulfur added (the other conditions remaining the same) recoveries in most cases averaged around 60 to 80 per cent. Table 1 gives the analyses of the steel scrap, alloys, and furnace additions used in the production of the steels to which reference is made in this paper.

TABLE 1.—*Analyses of Raw Material*

	Carbon, Per Cent.	Silicon, Per Cent.	Manganese, Per Cent.	Phosphorus, Per Cent.	Sulfur, Per Cent.	Zirconium, Per Cent.	Chromium, Per Cent.	Iron, Per Cent.	Titanium, Per Cent
Steel scrap	0.07	0.02	0.23	0.011	0.038				
Ferromanganese, H. C.	6.40	1.45	77.84	0.284	nil				
Ferrosilicon, 50 per cent.	0.05	49.76		0.041					
Washed metal	3.72	0.01	0.008	0.016	0.029				
Ferrophosphorus				26.65					
Iron sulfide					25.98				
Silicon-zirconium No. 1	0.20	72.14	0.96	nil		9.97	0.12	12.75	0.39
Silicon-zirconium No. 2A	0.40	47.47				28.09		23.49	0.40
Silicon-zirconium No. 2B	0.07	48.74				36.65		12.97	0.43
Silicon-zirconium No. 2C	0.00	43.58				37.87			
Silicon-zirconium No. 2D	0.48	50.77				44.53		2.54	0.70

Mechanical and Thermal Treatment

Forging temperatures were controlled by means of a Leeds & Northrup optical pyrometer, the filament of which was sighted against the surface of the ingot. The maximum temperatures chosen lay within the range 1050-1300° C., the chief factor determining the particular temperature chosen being the carbon content.

During forging, all ingots were worked down in finished size to 1-in. round, corresponding to a 90-per cent. reduction, and cut into bars ranging in length from 15 to 18 in. The forged bars, during annealing, were held for 1 hr. at the selected annealing temperature and allowed to cool in the furnace. Rates of cooling, in degrees per minute, as determined from a cooling curve, taken after an annealing run at 815° C., were as follows:

TEMPERATURE, DEGREES C.	RATE OF COOLING, DEGREES PER MIN.
815	10.5
750	4.7
720	2.8
600	1.8

Normalizing treatment of forged bars consisted in heating to the desired normalizing temperature, removing the bars from the furnace, and permitting them to cool freely in air. The annealing temperatures selected were taken from the curve given by Sauveur,[2] the carbon content alone being the determining factor. Annealing and normalizing temperatures were measured by means of an iron-constantan thermocouple and a Leeds & Northrup potentiometer indicator. The same method of temperature measurement was used in all the procedures involving the quenching and drawing (tempering) of heat-treated test pieces.

Quenching and drawing operations on forged material were carried out on the tensile and Izod impact test pieces after they had been machined to finished size, except that the notches in the impact test pieces were always cut after quenching and drawing.

All slab ingots were rolled to ½-in. plate in a 26-in. cogging mill at the plant of a company engaged in the manufacture of high-quality carbon and alloy steels of crucible and electric-furnace grade. Rolling operations were conducted by skilled operators, under the supervision of the rolling-mill superintendent. Rolling practice employed on the heats herein referred to is given in Table 2.

TABLE 2.—*Rolling Practice on ½-in. Plate*

Heat Number	Maximum Rolling Temperature, Degrees C.	First Pass Draft, Inch	Total Number of Passes	Remarks
286	1105	1	5 or 6	
305	1065–1093	½	7	2d and 3d pass ½ in.; next three passes ¼ in. each; final pass 3⁄16 in.
318	1093	¾		Remaining passes ½ in.
331	1093	⅞	5	Remaining passes ½ in.
332	1093	⅞	5	Remaining passes ½ in.
333	1093	⅞	5	Remaining passes ½ in.

The finished plate, ½ in. thick, measured approximately 7¼ in. wide and 34 in. long. As the original ingot possessed an averaged cross-section of 2¾ by 6 in., the total reduction in rolling was 78 per cent. The ingots were straight rolled, butt first, without cross-rolling.

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[2] "Metallography and Heat Treatment of Iron and Steel," 234. Cambridge, Mass., 1916. Sauveur & Boylston.

In all forging and rolling operations, each lot of ingots from a given heat were treated in an identical manner, as far as possible. The annealing and normalizing of the forged bars from the ingots of each heat were conducted simultaneously and in the same furnace on all the bars. The same principle was followed in all quenching and drawing procedures on machined test pieces.

Selection of Test Pieces and Methods of Test

Forged bars were marked in such a way that their exact position in the original ingot was known. In conducting tests on any given heat of steel, whether poured into two or three ingots, an equal number of each particular type of test piece was taken from the bars of each ingot and in such a way as to correspond to identical positions in the original ingot. All test data herein reported were obtained on sound material free from primary or secondary pipe. Where the ingot furnished more sound material than was needed for purposes of test, the metal was taken from the bars corresponding to the base of the ingot upward. The test pieces from the forged steels were taken axially from the 1-in. round bars.

In the case of the ½-in. rolled plate, tensile and Izod impact test pieces were taken, both longitudinally and transversely, from that part of the plate which corresponded to the base of the original slab ingot.

Tensile test pieces from the 1-in. round forged bars were of standard dimensions (0.505 in. diameter, 2 in. gage length) and in all cases made with threaded shoulders, ¾ in. diameter. The tensile test pieces from the ½-in. plate were necessarily of a special form, having a diameter of 0.300 in. and a gage length of 2 in. In order that these test pieces might be screwed into the regular threaded grips of the Olsen machine, they were first machined with shoulders having a cross-section of ½ by ¾ in. and then threaded to ¾ in., the cross-section of the finished shoulder being a portion of a circular area bounded by two chords, each parallel to and equidistant from a diameter of a ¾-in. circle. The impact test pieces, 10 by 10 mm. in cross-section, were furnished with a 45° notch, 2 mm. deep, having a radius of ⅔ mm. at its base.

Experiments on fatigue resistance have so far been confined to annealed steels, using test pieces 0.300 in. in diameter, 13 in. long, tested to destruction in a Farmer machine under rotary alternating stress. The bars were not tapered nor reduced in diameter at any part of their length to fix the position of greatest fiber stress, our own work on such steels indicating that it is possible to obtain remarkably close checks on plain rods, regardless of the fact that breakage usually occurs close to or within one of the two collars that rotate with the rod and to which, through ball bearings, the loads are applied.

Tensile tests were made on a 100,000-lb. Olsen machine, the holders of which rest in spherical seats. When locating the yield point, a cross-head speed of 0.03 in. per min. was used; above the yield point, the speed was 0.23 in. per min. When determining yield point, readings of strain were made after each increment of 250 lb. beam load for the 0.505 in. diameter test pieces and after each increment of 50 lb. beam load for the 0.300 in. diameter test pieces. A Berry strain gage, reading to $^1/_{5000}$ in. was used in practically all of the tests on heat-treated steels, as well as for a considerable part of the tests on pearlitic steels. Otherwise, a Hayes-Lewis extensometer with a dial reading to $^1/_{10,000}$ in. was used. The first measuring device was more satisfactory in testing hard steels and in measuring permanent set than the latter.

For reasons that will be explained later, the yield point was taken as that stress at which the increment in strain per unit increment of stress attained a value equal to four times that which characterized the lower, completely elastic portion of the stress-strain curve.

Impact tests were made on a 120-ft.-lb. Izod machine of the pendulum type. The length of the cantilever was 28 mm., measured from the notch, and the striking distance was 22 mm. (0.866 in.).

Brinell tests were made on an Aktiebolaget Alpha machine, using a 10-mm. ball. Except where stated to the contrary, measurements were made on the longitudinal surfaces of the Izod impact test pieces after the latter were broken. Pressure was allowed to act upon the ball for 30 sec. before release.

Final analyses were made on the lathe turnings taken from the tensile test pieces, and in all cases correspond to average values based on two to five samples.

Deoxidizing Power of Zirconium

That type of zirconium alloy that has proved to be most efficacious is an alloy of zirconium, silicon, and iron of very low carbon content (0.05–0.30 per cent. carbon). Analyses of samples from representative lots of the silicon-zirconium alloys used in this investigation are given in Table 1. Such alloys may be regarded as ferrosilicons in which zirconium replaces a portion of the iron content. They resemble ferrosilicon in their general appearance but are more compact and less friable than ordinary 50-per cent. ferrosilicon. As customarily made, the range of silicon content is from 45 to 70 per cent. and of zirconium content from 10 to 40 per cent.

Percentage recoveries obtained in ferro-alloy practice depend on a number of factors. One of the most important, in the case of those elements that function as deoxidizers, is rate of chemical reaction with the oxygen content of the molten steel. When the added ferro-alloy contains more than one deoxidizing element, as do the silicon-zirconium alloys, that element which displays the greatest rate of reaction will show the lowest recovery. Table 3 contains data on percentage recoveries of silicon and zirconium from silicon-zirconium alloys containing 33 to 38 per cent. zirconium and approximately 50 per cent. silicon, together with corresponding recoveries of silicon from 50 per cent. ferrosilicon. the alloys being added to two ladles from the same heat in each instance.

For the heats listed the average recovery of silicon from the silicon-zirconium alloy is 97 per cent., as against 86 per cent. from ordinary ferrosilicon. This increase in recovery results from the protective action of the zirconium content of the former alloy. Recovery of zirconium averages 59 per cent.; the individual values do not vary from this average much more than do the individual values for silicon recovery vary from their mean value.

The fact that the recovery of zirconium is lower than that of silicon, when the two elements are present in the same alloy, is strong evidence in favor of the greater deoxidizing power of the former. This evidence is strengthened when it is recalled that silicon-zirconium alloys become incorporated in the molten bath as readily and as quietly as does ferrosilicon and that the boiling point of zirconium is very high. Low recoveries in the case of certain other reactive elements might be justly attributed to their relative insolubility or low boiling point; but these objections cannot be raised in the case of silicon-zirconium alloys.

Final proof as to the combined deoxidizing and scavenging action of a deoxidizer must lie in an accurate determination of the total oxygen content of the steel. Fortunately, it was possible, through a coöperative agreement with the Bureau of Standards, to have a number of gas determinations made on steels, treated with zirconium and untreated, by means of its recently devised method involving fusion in vacuo in contact with carbon and analysis of the evolved gases. It is understood that a description of this method is to be published by the Bureau. The analyses were made by C. E. Plummer, of these Laboratories, working in The Metallurgical Division of the Bureau under the supervision of Louis Jordan. The values for oxygen arrived at by this method include not only that which exists as the relatively easily reduced oxide and silicate of iron, but also the oxygen existing as the oxide and silicate of manganese and zirconium.

In Table 4 are given the values for oxygen and hydrogen obtained on a number of zirconium-treated steels, and, except for heat number 307, the corresponding values for untreated steels (*i.e.* treated with ordinary ferrosilicon) from the same heats.

TABLE 3

Heat Number	Ladle B Treated with Ferrosilicon			Ladle C Treated with Silicon-zirconium					
	Per Cent. Silicon		Per Cent. Silicon Recovery	Per Cent. Silicon		Per Cent. Zirconium		Per Cent. Silicon Recovery	Per Cent. Zirconium Recovery
	Added	Recovered		Added	Recovered	Added	Recovered		
105	0.24	0.19	79	0.26	0.21	0.15		81	
107	0.27	0.26	96	0.26	0.23	0.15	0.07	88	47
115	0.20	0.18	90	0.19	0.17	0.11	0.07	90	64
120	0.18	0.13	72	0.19	0.16	0.11	0.08	84	73
121	0.19	0.17	89	0.22	0.20	0.17	0.07	91	41
125	0.25	0.07	28	0.25	0.20	0.19	0.10	80	53
133	0.25	0.22	88	0.23	0.20	0.17	0.12	87	71
134	0.22	0.20	91	0.21	0.21	0.16	0.09	100	56
135	0.20	0.18	90	0.18	0.18	0.13	0.11	100	84
137	0.21	0.24	114	0.18	0.27	0.14	0.10	150	72
138	0.18	0.21	117	0.21	0.20	0.16	0.09	95	56
142	0.20	0.19	95	0.19	0.21	0.14	0.07	110	50
144	0.20	0.22	110	0.21	0.28	0.16		133	
147	0.22	0.22	100	0.21	0.24	0.16	0.15	114	94
148	0.20	0.19	95	0.21	0.21	0.16	0.12	100	75
153	0.29	0.25	86	0.19	0.27	0.14	0.06	142	43
183	0.23	0.20	87	0.21	0.21	0.16	0.11	100	69
198	0.17	0.12	70	0.16	0.16	0.12	0.05	100	42
199	0.20	0.13	65	0.18	0.14	0.13	0.04	78	31
200	0.24	0.18	75	0.22	0.20	0.16	0.06	91	37
201	0.22	0.19	85	0.19	0.20	0.14	0.07	105	50
202	0.21	0.15	72	0.19	0.16	0.14	0.07	84	50
203	0.23	0.19	83	0.19	0.18	0.14	0.11	95	79
204	0.21	0.17	81	0.17	0.17	0.13	0.09	100	69
205	0.19	0.18	95	0.18	0.21	0.13	0.08	116	62
206	0.22	0.15	68	0.22	0.15	0.17	0.10	68	59
208	0.23	0.17	74	0.21	0.20	0.16	0.10	95	62
211	0.21	0.20	95	0.17	0.18	0.13	0.07	106	54
214	0.22	0.20	91	0.25	0.20	0.19	0.11	80	57
215	0.23	0.19	83	0.20	0.20	0.15	0.13	100	87
217	0.22	0.22	100	0.20	0.15	0.15	0.10	75	67
220	0.23	0.20	87	0.23	0.23	0.17	0.11	100	65
222	0.28	0.25	89	0.22	0.20	0.16	0.09	91	56
225	0.22	0.17	77	0.23	0.16	0.17	0.07	69	41
Av......	0.219	0.188	85.8	0.206	0.199	0.150	0.089	97.0	59.7

NOTE.—Recoveries of silicon are based on a zero silicon content of the steel as tapped into the ladle; hence the fact that sometimes the value for percentage recovery exceeds 100 per cent.

For the four heats where a comparison of the effect of silicon-zirconium and ferrosilicon is possible, the average oxygen content of the steels treated with the former alloy is 0.0049 per cent. as against 0.0106 per cent. for those treated with the latter. The data on the three steels of heat number 307 show the decrease in oxygen content with an increase of zirconium.

TABLE 4

Heat Number	Treated with	Gas Content, Per Cent.		Analysis, Per Cent.					
		Oxygen	Hydrogen	Zirconium	Carbon	Silicon	Manganese	Phosphorus	Sulfur [a]
207	SiZr	0.0025	nil	0.09	0.50	0.23	0.57	0.020	0.036
	FeSi	0.0152	0.0010	none	0.50	0.21	0.57	0.020	0.038
285	SiZr	0.0047	0.0004	0.15	0.98	0.15	0.37	0.018	0.116
	FeSi	0.0096	0.0004	none	0.99	0.21	0.38	0.018	0.110
309	SiZr	0.0085	0.0010	0.10	0.27	0.17	0.77	0.023	0.022
	FeSi	0.0096	0.0002	none	0.27	0.17	0.77	0.023	0.022
310	SiZr	0.0040	nil	0.11	0.50	0.17	0.30	0.011	0.015
	FeSi	0.0081	0.0003	none	0.49	0.18	0.31	0.012	0.016
307	SiZr	0.0059	0.0006	0.08	0.62	0.94	0.43	0.037	0.069
	SiZr	0.0015	0.0006	0.18	0.64	1.15	0.43	0.036	0.058
	SiZr	0.0014	0.0002	0.32	0.64	1.37	0.43	0.034	0.041

[a] Sulfur determined by gravimetric method involving oxidation.

The Electro Metallurgical Co., during a number of years prior to its development of commercial processes for manufacturing zirconium alloys, accumulated considerable information regarding zirconium and its behavior. The tenacity with which it resisted electric-furnace reduction to metallic form and its great affinity for oxygen and nitrogen in particular were noted. Mr. Becket had observed, for instance, that when a molten alloy containing aluminum, zirconium, titanium, silicon, and iron was exposed to the action of an oxidizing slag, the rate of elimination of these elements was in the order named. Aluminum was the most rapidly oxidized, differing in rate, however, only slightly from zirconium. Titanium was oxidized at a rate not much exceeding that of silicon.

The data of Gillett and Mack[3] are in agreement with these observations. Crop ends of zirconium steels when remelted gave the data shown below:

Heat Number	Present in Crop Ends at Start		Analysis of Finished Steel	
	Per Cent. Zirconium	Per Cent. Titanium	Per Cent. Zirconium	Per Cent. Titanium
1204	0.25	0.03	none	0.005
1205	0.20	0.03	none	0.008
1317	0.44	0.034	0.03	0.01

These investigators also found that the average recoveries of titanium, zirconium, nickel, and aluminum when present together in the alloys with which they worked were as follows: Aluminum 40 per cent., zirconium 52 per cent., titanium 79 per cent., nickel 80 per cent.

EFFECT OF ZIRCONIUM ON AMOUNT AND MODE OF OCCURRENCE OF NITROGEN IN STEEL

Occurrence of Zirconium Nitride

It was early observed that in many zirconium-treated steels there occurred a new crystalline microconstituent of a bright lemon-yellow color, resistant to acid attack but blackened by a boiling alkaline sodium-picrate solution. In view of the great affinity zirconium is known to exhibit for nitrogen and the general resemblance the microconstituent bears to the pink crystals of titanium nitride (or cyanonitride) often occurring in steels containing titanium, it was concluded that these yellow cubic crystals were nitride of zirconium.

A number of zirconium nitrides are reported in the literature. In appearance the above-mentioned microconstituent would seem to correspond most closely to a product prepared, in 1839, by Mallet, who describes it as a golden-yellow substance having the form of microscopic cubes. More recently (1912), Wedekind has assigned the formula Zr_3N_2 to a nitride prepared at high temperatures from metallic zirconium and nitrogen.

As seen in a polished and etched section, the yellow crystals display the form of the various sections of a cube (square, rectangle, triangle hexagon, see Fig. 1) and are not ordinarily noticed under the microscope at magnifications below 500 diameters. Once their presence and location are known, they may be detected at 100 diameters, especially against the dark background of pearlite in steels at or near the eutectoid composition.

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[3] Bureau of Mines *Bull.* 199, 51.

It appears to be generally true that in the type of steel practice followed in this investigation, visible zirconium-nitride crystals do not occur in steels of less than 0.23 per cent. carbon. This conclusion is based on careful observations made on a series of forty heats of varying carbon content treated with approximately 0.15 per cent. zirconium. The steel of lowest carbon content in which the crystals were observed was from heat number 205, containing 0.232 per cent. carbon, 0.024 per cent. phosphorus, and 0.032 per cent. sulfur. No crystals were observed in the steel of heat number 202, containing 0.236 per cent. carbon, 0.026 per cent. phosphorus, and 0.022 per cent. sulfur. In the case of the remaining steels, the crystals occurred whenever the carbon was above 0.23 per cent. and were absent when the carbon content fell below this figure. The range of carbon content in the series was 0.07–0.73 per cent.

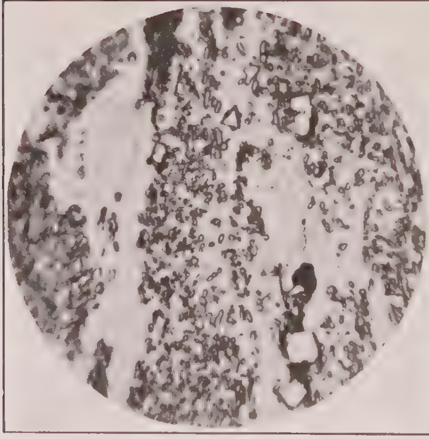

FIG. 1.—ZIRCONIUM-NITRIDE CRYSTALS. × 825.

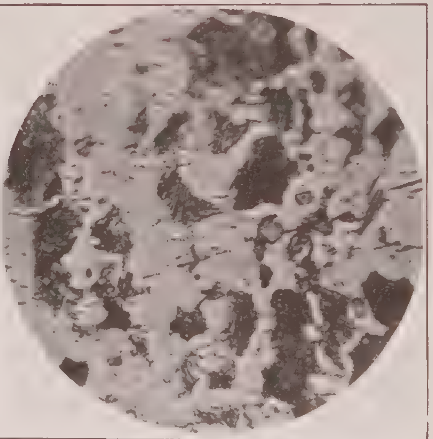

FIG. 2.—ZIRCONIUM-NITRIDE CRYSTAL EMBEDDED IN SLAG INCLUSION. × 500.

Increase in phosphorus up to 0.17 per cent. did not affect the amount of carbon content at which the crystals began to make their appearance, although in a steel (heat number 148) containing 0.262 per cent. phosphorus, yellow crystals were observed for a carbon content of 0.187 per cent.

The nitride constituent begins to appear in steels of 0.23 per cent. carbon and above when the amount of added zirconium exceeds approximately 0.04–0.06 per cent. Up to about 0.15 per cent. added zirconium, there is the expected tendency toward increase in its abundance. Additions of zirconium above 0.15 per cent. fail to bring about any corresponding increase in the number of yellow crystals.

A number of counts, made by a careful survey of thirty-three fields at 500 diameters for each specimen examined, showed the average field to contain 1.27 crystals. This is equivalent to 22 crystals per square millimeter. Deviation from this average value is not so great as might be expected, indicating that it is in the nature of a constant for any given family of steels.

Nitrogen Content of Zirconium-treated Steels

In Table 5 are given the results of analytical determination of "nitride nitrogen" by the Allen method for four heats of steel, each heat being treated with silicon-zirconium and with ferrosilicon in equivalent amounts in duplicate ladles. By "equivalent amounts" is meant those corresponding to equal additions of silicon. Table 6 shows nitride-nitrogen values for two other heats, treated in three ladles with different amounts of zirconium.

TABLE 5.—*Nitride-nitrogen Content of Steels with and without Zirconium*

Heat Number	Ladle	Per Cent. Added Zirconium	Per Cent. Zirconium Recovered	Per Cent. Nitride-nitrogen
211	B	none		0.0062
	C	0.13	0.07	0.0034
217	B	none		0.0072
	C	0.15	0.10	0.0023
215	B	none		0.0086
	C	0.15	0.13	0.0059
222	B	none		0.0072
	C	0.16	0.09	0.0024

TABLE 6.—*Nitride-nitrogen Content of Zirconium-treated Steels*

Heat Number	Ladle	Per Cent. Added Zirconium	Per Cent. Zirconium Recovered	Per Cent. Nitride Nitrogen
243	A	0.32		0.0022
	B	0.20		0.0031
	C	0.14		0.0031
252	A	0.39		0 0021
	B	0.28		0.0024
	C	0.20		0.0028

Considering average values, the steels treated with ferrosilicon contained 0.0073 per cent. nitride nitrogen, those treated with 0.15 per cent. zirconium as silicon-zirconium contained only 0.0035 per cent. nitride nitrogen. With heavier zirconium additions, there is a tendency toward lower values for nitride nitrogen, but the decrease is not proportional to the increase in zirconium. As the procedure in analyzing steels by the Allen method decomposes completely the zirconium-nitride constituent, it would logically follow that the yellow cubic crystals observed in steels treated with zirconium represent simply that residuum of nitrogen which has been "fixed" by zirconium but not fluxed off.

Nitrogen Elimination

An insight into the mechanism by which zirconium eliminates nitrogen from steel is afforded by the not infrequent occurrence of a nitride crystal within a slag inclusion. Fig. 2 shows this phenomenon. During those slagging reactions that occur immediately after additions are made to the ladle, a certain proportion of the zirconium-nitride particles simultaneously forming in and crystallizing from the steel are entrapped in the molten slag masses rising through the liquid steel and mechanically removed therefrom.

As the amount of nitrogen present in the steel, as tapped into the ladle, is strictly limited and must tend to vary around some mean value, the maximum number of crystals formed is also limited. Similarly, the number of crystals remaining in the steel tends to vary around some mean value, the additional controlling factor here being the volume of the slagging processes accompanying deoxidation.

Assuming that the reaction between zirconium and nitrogen may be represented by the equation $3Zr + N_2 = Zr_3N_2$, one part by weight of zirconium available for the purposes of the reaction is capable of "fixing" 0.10 part of nitrogen.

In Table 7 the ferrosilicon-treated steels show an average of 0.0073 per cent. nitride nitrogen. If all of this is fixed by zirconium and if this is the total nitrogen capable of reaction with zirconium, only 0.073 per cent. zirconium is required to combine with all the nitrogen present. Adding to this value the 0.04–0.06 per cent. zirconium required for deoxidizing reactions, a total of 0.113–0.133 per cent. zirconium is obtained. This checks very satisfactorily, all things considered, with the value of 0.15 per cent. added zirconium at which further increase in the number of nitride crystals ceases.

As already stated, occurrence of the nitride microconstituent is limited to steels of 0.23 per cent. carbon and above. This critical value corresponds exactly with the minimum carbon content that experience showed it was possible to obtain in a steel made under carbide-slag conditions.

Steels of 0.23 per cent. carbon and under may possess an initial oxygen content that places them in a class separate from the higher-carbon steels, so far as the volume of the slagging processes in the ladle are concerned. If it is true that these lower-carbon steels had, prior to deoxidation, an oxygen content of an order of magnitude somewhat higher than that of the steels containing more than 0.23 per cent. carbon and prepared under more strongly reducing conditions in the furnace, the absence of yellow crystals from the former class might be ascribed either to a deficiency of zirconium available for reaction with nitrogen or to a practically complete "clean up" of the nitride crystals formed through the agency of the large volume of slag produced by deoxidation reactions. The writer is inclined to favor the latter view, realizing at the same time that in the case of a very highly oxidized steel the former may be a factor, if not the determining one. In the present practice, however, certain effects are produced by zirconium upon the mechanical properties of steels, containing less than 0.23 per cent. carbon and free from yellow nitride crystals, which it is concluded are properly laid to zirconium and its capacity for "fixing" nitrogen. These effects will be discussed later.

Effect of Zirconium on Amount and Mode of Occurrence of Sulfur in Steel

Occurrence of an Acid-insoluble Compound of Zirconium and Sulfur

When added in sufficient amount to steel, zirconium combines chemically with sulfur to form a compound insoluble in 1:1 hydrochloric acid. Sulfur present in this form is not detected by nor included in an analysis performed by the ordinary "evolution" method. In the gravimetric method involving oxidation, however, it is quantitatively determined.

Numerous analyses have been made on steels of varying composition treated with different amounts of zirconium, as silicon-zirconium, to ascertain the relation between the amount of sulfur "fixed" in acid-insoluble form and the amount of added zirconium. As a result, it has been established that, over a wide range of values, the following relation holds good to a surprisingly close approximation:

$$\text{Per cent. sulfur ``fixed''} = \frac{\text{Per cent. added zirconium} - 0.15}{10}$$

In other words, every part by weight of zirconium, over and above 0.15 per cent. added zirconium, combines with and fixes in acid-insoluble form 0.10 part by weight of sulfur. This critical value (0.15 per cent. added zirconium), it is to be observed, corresponds to that value above which zirconium ceases to form nitride due to exhaustion of the available supply of nitrogen. Table 7 exhibits analytical data on eight heats of steel, each treated in three ladles with different amounts of silicon-zirconium. The agreement between the observed values for sulfur determined by the evolution method and the corresponding values calculated by means of the foregoing equation is good. Additional data are given in Table 8, for heats treated in duplicate ladles with silicon-zirconium and ferrosilicon in equivalent amounts.

TABLE 7.—*Sulfur Contents of Zirconium-treated Steels by the Evolution Method, Observed and Calculated*

Heat Number	Ladle	Per Cent. Added Zirconium	Per Cent. Sulfur by the Evolution Method	
			Observed	Calculated
241	A	0.38	0.006	0.006
	B	0.21	0.019	0.023
	C	0.15	0.029	
242	A	0.45	0.014	0.018
	B	0.28	0.026	0.035
	C	0.20	0.043	
237	A	0.42	0.015	0.012
	B	0.21	0.028	0.033
	C	0.14	0.039	
248	A	0.35	0.015	0.016
	B	0.21	0.032	0.030
	C	0.15	0.036	
243	A	0.32	0.009	0.013
	B	0.20	0.023	0.025
	C	0.14	0.030	
253	A	0.31	0.018	0.027
	B	0.28	0.032	0.030
	C	0.20	0.038	
252	A	0.39	0.017	0.019
	B	0.28	0.032	0.030
	C	0.20	0.038	
262	A	0.54	0.071	0.081
	B	0.41	0.092	0.094
	C	0.26	0.109	

Table 9 contains sulfur percentages as determined by the evolution method for thirty heats, treated in duplicate ladles with: (1) approximately 0.15 per cent. zirconium as silicon-zirconium and (2) an equivalent amount of ferrosilicon. The close agreement between the two sets of average values was one reason why 0.15 was chosen as the second term of the numerator of the right-hand member of the equation relating percentage of sulfur fixed with percentage of added zirconium.

TABLE 8.—*Sulfur Contents of Steels, Treated with Ferrosilicon and Silicon-zirconium; Determined by the Evolution Method*

Heat Number	Ladle	Treated With	Per Cent. Added Zirconium	Per Cent. Sulfur by Evolution Method	
				Observed	Calculated
280	A	SiZr	0.15	0.071	0.075
	B	FeSi	none	0.075	
285	A	SiZr	0.16	0.103	0.109
	B	FeSi	none	0.110	
286	A	SiZr	0.42	0.096	0.085
	B	FeSi	none	0.112	
287	A	SiZr	0.42	0.075	0.080
	B	FeSi	none	0.107	
328	B	FeSi	none	0.008	
	C	SiZr	0.19	0.005	0.004
331	A	SiZr	0.40	0.006	nil
	B	FeSi	none	0.023	
	C	SiZr	0.22	0.022	0.016

NOTE.—Ladles B and C of Heat 331 were treated with equivalent amounts of silicon, ladle A with a heavier addition.

As the first 0.15 per cent. of zirconium added in the ladle is used in its chemical reaction with oxygen and nitrogen, ladle additions of silicon-zirconium, to a steel already treated in the furnace with additions of silicon-zirconium sufficient to take care of such oxygen and nitrogen, ought to bring about fixation of sulfur in direct proportion to the amount of zirconium used. In other words, the equation formerly given would have to be modified in the circumstances to the form

$$\text{Per cent. sulfur “fixed”} = \frac{\text{Per cent. added zirconium}}{0.10}$$

Table 10 contains sulfur data for heats numbers 307, 316, and 317, treated, in the furnace 15 min. before tapping, with 0.71, 0.35, and 0.70 per cent. zirconium as silicon-zirconium, respectively, and thereafter treated in three ladles with silicon-zirconium in the amounts shown. Two sets of calculated values for sulfur are given, the first set being obtained by use of the original equation, the second by means of the modified equation.

TABLE 9.—*Sulfur Contents of Steels, Treated with Ferrosilicon and with Silicon-zirconium; Determined by the Evolution Method*

Heat Number	Per Cent. Sulfur by the Evolution Method		Per Cent. Zirconium Added to Ladle C
	Ladle B, Treated with Ferrosilicon	Ladle C, Treated with Silicon-zirconium	
105	0.030	0.031	0.15
107	0.047	0.046	0.15
115	0.016	0.019	0.11
120	0.021	0.023	0.11
121	0.027	0.026	0.17
133	0.049	0.044	0.17
134	0.022	0.021	0.16
135	0.030	0.027	0.13
137	0.028	0.027	0.14
138	0.061	0.059	0.16
142	0.041	0.040	0.14
147	0.067	0.062	0.16
183	0.029	0.029	0.16
198	0.024	0.023	0.12
199	0.029	0.027	0.13
200	0.025	0.026	0.16
201	0.031	0.033	0.14
202	0.022	0.022	0.14
203	0.031	0.033	0.14
204	0.026	0.025	0.13
205	0.032	0.032	0.13
206	0.036	0.036	0.17
208	0.033	0.037	0.16
211	0.054	0.053	0.13
214	0.044	0.043	0.19
215	0.042	0.036	0.15
217	0.044	0.040	0.15
220	0.029	0.030	0.17
222	0.030	0.028	0.16
310	0.016	0.015	0.14
Average.........	0.0339	0.0331	0.147

The values given in Table 10 afford conclusive proof that, when silicon-zirconium is added in the ladle to steels already treated in the furnace with this alloy, the extent of sulfur fixation per unit of added zirconium

TABLE 10

Heat Number	Ladle	Per Cent. Zirconium Added in Ladle	Per Cent. Sulfur by Evolution Method		
			Observed	Calculated by First Equation	Calculated by Second Equation
307	A	none	0.062		
	B	0.13	0.053	0.062	0.049
	C	0.34	0.036	0.043	0.028
316	A	none	0.055		
	B	0.16	0.048	0.054	0.039
	C	0.30	0.036	0.040	0.025
317	A	none	0.016		
	B	0.15	0.006	0.016	0.001
	C	0.37	0.008	nil	nil

is greater than under ordinary conditions. The observed values are, it is true, somewhat higher than those calculated by means of the modified equation, but they consistently fall below those calculated from the original equation.

Sulfur prints of steels treated with zirconium in amounts above 0.15 per cent. furnish evidence of considerable interest, especially where there remains only a small percentage of sulfur not combined with zirconium.

Sulfur Elimination

It is possible, by means of fairly heavy additions of zirconium, to bring about elimination of sulfur in the ladle. To prove the elimination of sulfur, gravimetric analyses must be made on samples taken before and after the zirconium addition is made. Probably the best elimination obtained in our small ladle practice was in the case of heat number 262. By the addition of 0.41 per cent. zirconium, 0.017 per cent. sulfur was eliminated; and by the addition of 0.54 per cent. zirconium in another ladle, 0.024 per cent. sulfur was removed from the steel. These values refer to the actual elimination as determined from gravimetric analyses, not to the question of the fixation of sulfur in acid-insoluble form.

EFFECT OF ZIRCONIUM ON DISTRIBUTION OF IMPURITIES

Persistence in the pearlitic state of non-uniform carbon distribution inherited from the pre-existent austenitic solid solution is at least an indication of the presence of undesirable oxidized impurities characterized by low diffusion rates. Whereas the writer is not prepared to maintain that all instances of "banded" or "rolled" structure observed in longitudinal sections of rolled and forged products are due to the presence of such oxidized impurities and to no other cause, it is reasonable to infer that when two ingots, poured in duplicate ladles from the same heat of steel and thereafter treated in an identical manner, show consistent differences, the forged bar or rolled plate that shows the least indication of banded structure possesses the greater freedom from such impurities. Figs. 3 to 8 are photomicrographs of a number of steels, each

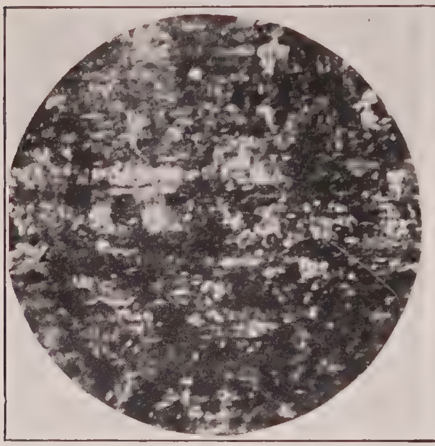

Fig. 3.—Longitudinal section, as rolled; heat No. 285; 0.99 per cent. C, 0.19 per cent. Si, 0.37 per cent. Mn, 0.017 per cent. P, 0.110 per cent. S. × 50.

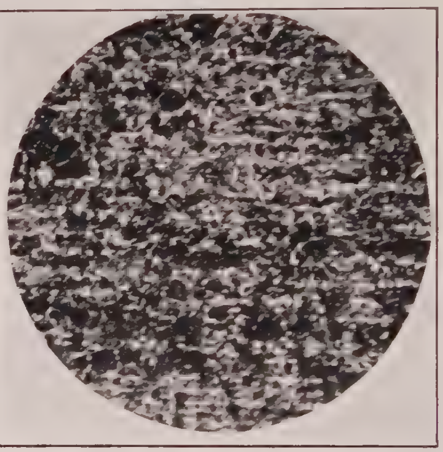

Fig. 4.—Longitudinal section, as rolled; heat No. 285; same composition as for Fig. 3, but treated with 0.16 per cent. Zr. × 50.

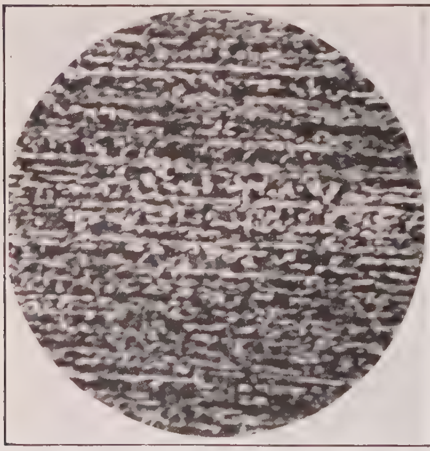

Fig. 5.—Longitudinal section, as rolled; heat No. 287; 0.58 per cent. C, 0.43 per cent. Si, 0.19 per cent. Mn, 0.016 per cent. P, 0.107 per cent. S. × 50.

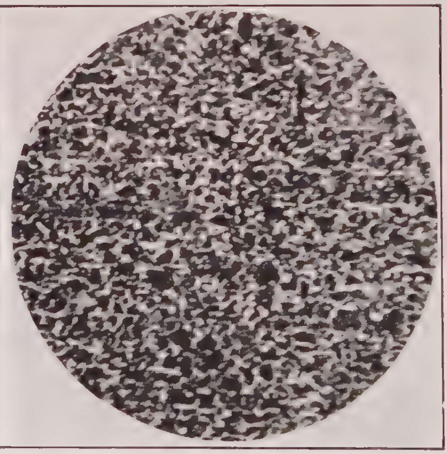

Fig. 6.—Longitudinal section, as rolled; heat No. 287; same composition as for Fig. 5, but treated with 0.42 per cent. Zr. × 50.

pair from the same heat and similar in every way except that one contains zirconium and the other contains none. The photomicrographs are taken from typical sections parallel to the direction of forging or rolling, the steels being in the "as forged" or "as rolled" condition in every case. This ability of zirconium to prevent or reduce in extent the formation of "banded" structure is no doubt due to its deoxidizing and scavenging action.

The solid, non-metallic inclusions, or sonims, occurring in steel can be divided into two classes. The first class includes all sonims that are the insoluble end products of reactions between two components both of which were, previous to reaction, dissolved in the molten-steel bath. They are in effect precipitated from the steel solvent. The class includes manganese sulfide, manganese oxide, silica, etc. Because of their mutual fluxing action, the two last-mentioned sonims form a manganese silicate, with or without incorporated iron silicate, if suitable opportunity is afforded. Zirconium nitride belongs to this class, although the fact that it crystallizes idiomorphically gives it a close relationship to certain carbides that crystallize as a primary phase, for instance, primary cementite.

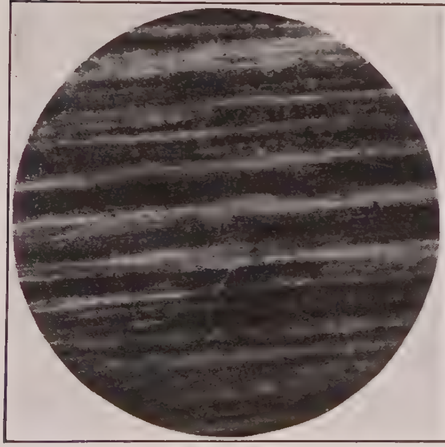

FIG. 7.—HIGH-SPEED, CHROMIUM-TUNGSTEN STEEL, AS FORGED; LONGITUDINAL SECTION; HEAT No. 303. × 50.

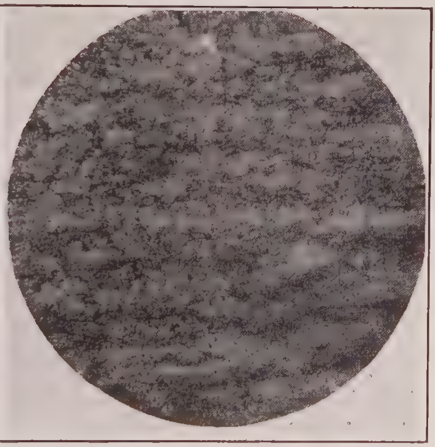

FIG. 8.—HIGH-SPEED CHROMIUM-TUNGSTEN STEEL, LONGITUDINAL SECTION, AS FORGED; SAME AS IN FIG. 7 BUT TREATED WITH ZR; HEAT No. 303. × 50.

The second class of sonims embraces those particles that remain dissolved in the molten steel up to the point of solidification but which are then precipitated therefrom because of their insolubility in solid austenite. It numbers among its members the oxide and sulfide of iron. Because of their mode of genesis, sonims of this second class should be subject to a high degree of dispersion and probably constitute a large proportion of the emulsified inclusions that occur in steel. Such a classification makes it more readily possible to account for the observed marked effect of zirconium in causing what would appear to be a coagulation of inclusions of the emulsified type.

It is not reasonable to suppose that coagulation can occur because of the direct action of zirconium on inclusions of the second class. These inclusions are formed *in situ* in austenite coincidently with solidification of the steel. Nor is it to be expected that zirconium or its oxidized products possess an entirely unique fluxing relationship with respect to finely divided inclusions of the first class; the term fluxing is here used in its usual meaning of promoting fluidity by mutual solution. Nevertheless, it has been consistently observed in the case of a series of high-sulfur steels which in the ordinary untreated condition show a ferrite rich in emulsified inclusions, that treatment with zirconium brings about the formation of ferrite of a much higher order of cleanness, in many instances as free from emulsion as is low-sulfur steel of good quality. Photomicrographs illustrating this effect of zirconium are difficult to obtain, for the emulsified particles are barely visible at 500 diameters.

The logical inference to be drawn is that zirconium treatment is able to eliminate emulsified slag by preventing its formation; that is, by removing the last traces of iron oxide and sulfide from the liquid steel, before solidification occurs. This gives the same end effect as if zirconium had coagulated the emulsified slag and permitted it to escape from the molten steel. Neither zirconium oxide nor other infusible, oxidized product of zirconium has been observed in any of the numerous steels treated with zirconium.

MECHANICAL PROPERTIES OF ANNEALED CARBON STEELS CONTAINING ZIRCONIUM

Low-phosphorus Series

An examination of the results of tensile, Izod, and Brinell tests on annealed carbon steels throws additional light on the effects of zirconium in steel. Tables 11, 12, and 13 contain such data for twelve heats of carbon content varying from 0.14 to 0.70 per cent., each heat treated in triplicate ladles in accordance with the following practice: Table 11, ladle A, 0.20 per cent. silicon and 0.04 per cent. zirconium as a silicon-zirconium alloy analyzing approximately 10 per cent. zirconium and 70 per cent. silicon. Table 12, ladle B, 0.20 per cent. silicon as 50-per cent. ferrosilicon. Table 13, ladle C, 0.20 per cent. silicon and 0.15 per cent. zirconium as a silicon-zirconium alloy analyzing approximately 48 per cent. silicon and 36 per cent. zirconium.

There are no striking nor significant differences among the individual or average values for the three sets of steels, except in the yield-point values for the steels of ladle C, treated with 0.15 per cent. zirconium. Here the difference is of large magnitude and indicates deep-seated action. The yield point of heat number 310 was lowered from 37,500 lb. per sq. in., in the ferrosilicon-treated steel of ladle B, to 27,500 lb. per sq. in. in the case of the steel of ladle C, treated with 0.15 per cent. zirconium as silicon-zirconium, without significant change in ultimate strength, percentage elongation and reduction of area, Brinell hardness or Izod number; nor, as will be shown later, is there any appreciable change in the endurance or fatigue limit.

The method used in the determination of yield point has already been described. It consisted in observing that stress at which the increment in strain per unit increment in load attained a value four times as great as it possessed along the lower branch of the stress-strain curve. For all ferrosilicon-treated annealed carbon steels so far examined, the yield point as thus defined coincides with the rapidly accelerated movement of the extensometer needle and also with the drop of the beam. It also coincides, practically within experimental error, with Johnson's apparent elastic limit, and possesses over the latter the advantage of substituting the tangent for the slope of the stress-strain curve as a measure of deviation from linearity. For steels treated with 0.04 per cent. zirconium, the same condition holds true. In the case of annealed carbon steels treated with 0.15 per cent. zirconium, however, yield point as measured by the drop of the beam may be ill-defined and in some instances entirely masked. Attainment of yield point, as determined by change in the slope of the stress-stain curve, is realized below that stress which brings about such acceleration of extensometer needle or drop of beam as does occur. Expressed in another way, such steels as these show a smaller "drop in stress at yield" and in some cases the drop in stress may be practically absent.

Determinations of yield point by means of the "visible stretch" method fail to show any decided difference between the ferrosilicon-treated steels and those treated with 0.15 per cent. zirconium. For instance, in the case of heat number 310, the visible stretch method gives a yield point of 37,500 lb. per sq. in. for the ferrosilicon-treated steel and of 33,500 lb. per sq. in. for the steel treated with 0.15 per cent. zirconium. Yield by the method of visible stretch is defined as the stress corresponding to a strain of 0.01 in. in a 2-in. gage length. Heat number 310 shows a much greater difference in yield point by this method than most of the steels examined.

Obviously the yield point in the latter type of steel is not subject to the same general rules that experience has shown to be true for ordinary steels. Measurement of yield by any of the several commonly used methods gives closely concordant results on ordinary annealed steels.

Steels treated with 0.15 per cent. zirconium, on the other hand, give widely varying values for yield depending on the method selected.

It has remained for fatigue tests on zirconium-treated steels to disclose how complete is the lack of a direct relation between endurance limit and yield point as determined from the slope or tangent of the stress-strain curve. Table 14 gives experimental results obtained in fatigue tests on a number of the steels in Tables 11, 12, and 13. Endurance limit was calculated by the formula proposed by Moore and Seely.

The steels treated with 0.04 per cent. zirconium consistently show a higher endurance limit than the ferrosilicon-treated steels, while those treated with 0.15 per cent. zirconium have an endurance limit equal to that of the ferrosilicon-treated steels, in spite of their plastic behavior at relatively low stresses.

These and other test data have led to the conclusions (*a*) that the increase in the endurance limit of the steels treated with 0.04 per cent. zirconium is possibly due to elimination of traces of iron oxide by zirconium, and (*b*) that the effect of zirconium on the yield point in the case of the series of steels treated with 0.15 per cent. zirconium is due to fixation and elimination of nitrogen.

TABLE 11.—*Mechanical Properties of Annealed Carbon Steels, Low-phosphorus Series, Ladle A, Treated with* 0.20 *Per Cent. Silicon and* 0.04 *Per Cent. Zirconium as Silicon-zirconium*

Heat Number	Percentage		Yield Point, Lb. per Sq. In.	Ultimate Strength, Lb. per Sq. In.	Per Cent. Elongation	Per Cent. Reduction of Area	Brinell	Izod Number, Ft.-lb.
	Carbon	Phosphorus						
120	0.140	0.028	29,460	52,805	38.4	61.8	70.5	
202	0.234	0.025	33,810	66,254	32.8	52.3	103.8	33.9
205	0.248	0.021	39,199	67,192	38.1	56.2	99.5	36.4
121	0.316	0.027	36,364	69,459	31.8	51.0	105.9	38.9
208	0.308	0.015	35,820	70,105	36.5	55.7	112.0	30.3
115	0.314	0.008	38,077	68,301	32.3	56.9	109.7	36.4
206	0.359	0.016	37,025	73,992	32.2	51.2	105.6	26.4
203	0.390	0.024	40,286	76,791	33.5	52.1	125.9	20.6
204	0.444	0.019	40,010	80,541	28.6	47.4	132.6	23.5
310	0.517	0.014	40,943	90,920	20.8	33.6	148.0	10.0
211	0.610	0.023	44,055	98,468	24.4	40.6	177.0	13.8
217	0.703	0.020	42,392	102,431	19.9	31.4	172.8	5.8
Av.	0.382	0.020	38,120	76,438	30.8	49.2	122.0	

TABLE 12.—*Mechanical Properties of Annealed Carbon Steels, Low-phosphorus Series, Ladle B, Treated with* 0.20 *Per Cent. Silicon as* 50-*Per Cent. Ferrosilicon*

Heat Number	Percentage		Yield Point, Lb. per Sq. In.	Ultimate Strength, Lb. per Sq. In.	Per Cent. Elongation	Per Cent. Reduction of Area	Brinell	Izod Number, Ft.-lb.
	Carbon	Phosphorus						
120	0.110	0.016	30,156	47,850	38.1	64.4	71.0	
202	0.230	0.025	34,580	64,206	35.4	51.6	99.5	35.4
205	0.254	0.022	38,905	63,412	34.6	56.0	98.0	34.5
121	0.288	0.029	37,311	66,391	33.6	52.0	104.6	40.6
208	0.308	0.014	36,561	68,200	36.2	55.5	113.4	31.9
115	0.322	0.008	38,713	70,800	31.4	52.1	108.6	38.5
206	0.359	0.014	38,229	70,900	33.2	54.8	110.4	26.5
203	0.418	0.025	39,804	74,367	32.1	54.8	125.0	24.9
204	0.440	0.019	42,866	76,479	31.4	52.9	130.0	24.7
310	0.495	0.012	37,500	86,975	23.0	33.6	143.0	9.3
211	0.626	0.023	47,213	99,295	24.2	38.1	171.8	11.2
217	0.724	0.020	44,036	97,689	22.8	39.4	169.8	6.2
Av.....	0.382	0.019	38,823	73,880	31.3	50.4	120.4	

TABLE 13.—*Mechanical Properties of Annealed Carbon Steels, Low-phosphorus Series, Ladle C, Treated with* 0.20 *Per Cent. Silicon and* 0.15 *Per Cent. Zirconium as Silicon-zirconium*

Heat Number	Percentage		Yield Point, Lb. per Sq. In.	Ultimate Strength, Lb. per Sq. In.	Per Cent. Elongation	Per Cent. Reduction of Area	Brinell	Izod Number, Ft.-lb.
	Carbon	Phosphorus						
120	0.112	0.032	24,725	49,059	42.3	68.0	76.1	39.5
202	0.236	0.026	30,281	64,960	34.9	57.4	112.5	44.7
205	0.234	0.022	31,917	62,990	36.5	58.0	106.2	39.5
121	0.274	0.008	19,767	62,086	36.3	54.6	90.5	
208	0.320	0.014	28,249	67,193	31.8	57.8	105.8	34.1
115	0.326	0.008	36,292	69,225	32.6	54.5	108.0	42.4
206	0.358	0.017	29,167	71,783	32.0	52.4	114.6	30.2
203	0.406	0.026	31,561	73,995	31.2	51.5	121.8	24.9
204	0.446	0.022	34,423	76,783	31.5	49.4	130.0	24.1
310	0.495	0.011	27,500	84,334	26.0	37.5	144.0	10.3
211	0.662	0.023	36,000	106,930	25.5	36.9	168.8	12.1
217	0.730	0.019	29,766	97,133	21.0	32.5	178.9	5.9
Av.....	0.383	0.019	29,971	73,873	31.8	50.9	121.4	

TABLE 14.—*Fatigue Test Data on Annealed Carbon Steels, with and without Zirconium*

Heat Number	Ladle	Maximum Fiber Stress, Lb. per Sq. In.	Revolutions to Failure	Endurance Limit (Calculated), Lb. per Sq. In.	Observed Static Yield Point, Lb. per Sq. In.
120	A	28,000	582,500	19,630	29,460
	B	28,000	422,000	18,850	30,156
	C	22,000	4,254,700	19,770	24,725
202	A	28,000	3,132,500	24,210	33,810
	B	28,000	593,700	19,670	34,580
	C	28,000	602,300	19,710	30,280
205	A	35,000	286,800	22,450	39,199
	B	35,000	206,800	21,550	38,905
	C	30,000	511,300	20,690	31,917
208	A	32,000	1,060,100	24,170	35,820
	B	32,000	567,500	22,350	36,561
	C	28,000	644,100	19,870	28,249
206	A	29,000	1,433,600	22,750	37,025
	B	29,000	1,281,200	22,430	38,229
	C	29,000	557,000	20,220	29,167
203	A	35,000	688,500	25,050	40,286
	B	35,000	294,600	22,530	39,804
	C	30,000	2,302,700	24,970	31,561
204	A	34,000	411,000	22,810	40,010
	B	34,000	241,200	21,350	42,866
	C	34,000	257,800	21,520	34,423
211	A	40,000	140,200	23,470	44,055
	B	40,000	115,300	22,900	47,213
	C	35,000	760,000	25,360	36,000
217	A	37,000	465,400	25,210	42,392
	B	37,000	222,500	23,000	44,036
	C	28,000	2,258,600	23,250	29,766

	LADLE A, 0.04 PER CENT. ADDED ZIRCONIUM	LADLE B, NOT TREATED WITH ZIRCONIUM	LADLE C, 0.15 PER CENT. ADDED ZIRCONIUM
Average endurance limit, lb. per sq. in....	23,309	21,625	21,707

It is supposed that, whereas elimination of oxygen by zirconium raises the endurance limit, fixation and elimination of nitrogen causes a drop in yield, acting in this respect similar to phosphorus. When sufficient zirconium to take care of both nitrogen and oxygen is added, the two effects neutralize each other and the resulting steel has an endurance limit equal to that of an untreated steel. Zirconium additions above 0.15 per cent. fail further to lower the yield point, as measured by the tangent of the stress-strain curve, as would be expected if the effect on yield is due to reaction between zirconium and nitrogen.

High-phosphorus Series

A favorable effect of zirconium on the Izod number of annealed high-phosphorus steel was observed early in the present work. Accordingly, the mechanical properties of steels containing carbon, phosphorus, and zirconium in varying proportions were investigated. Tables 15, 16, and 17 contain the results obtained for a series of eleven heats with carbon contents covering the range 0.17 to 0.79 per cent. Ferro-alloy practice was the same as in the low-phosphorus series.

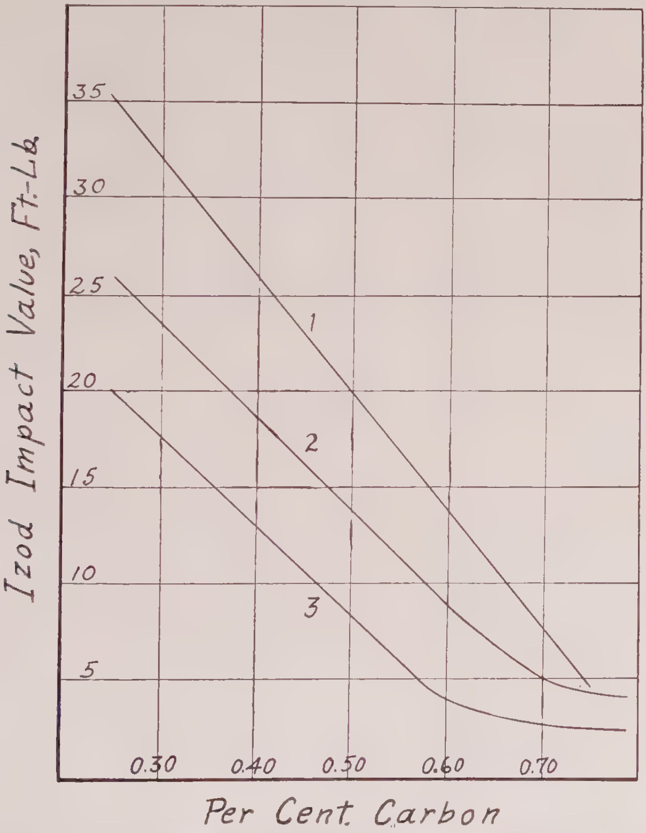

Fig. 9.—Relation between Izod impact values and carbon content for: (1) ordinary low-phosphorus steels, (2) high-phosphorus (0.140 per cent.) steels treated with zirconium, (3) same high-phosphorus steels not treated with zirconium; tests made in annealed state.

The effect of 0.15 per cent. added zirconium in increasing Izod number is especially to be observed. Fig. 9 shows graphically the relation between carbon content and Izod number of ordinary low-phosphorus

TABLE 15.—*Mechanical Properties of Annealed Carbon Steels, High-phosphorus Series, Ladle A Treated with* 0.20 *Per Cent. Silicon and* 0.04 *Per Cent. Zirconium as Silicon-zirconium*

Heat Number	Percentage		Yield Point, Lb. per Sq. In.	Ultimate Strength, Lb. per Sq. In.	Per Cent. Elongation	Per Cent. Reduction of Area	Brinell	Izod Number Ft.-lb.
	Carbon	Phosphorus						
200	0.172	0.141	40,887	65,406	35.5	60.8	109	35.6
107	0.170	0.144	46,658	70,772	35.6	59.6	119	33.0
142	0.184	0.124	45,894	68,983	34.6	63.2	119	45.3
201	0.264	0.154	46,297	75,363	32.3	57.0	130	30.4
220	0.258	0.165	45,335	76,582	32.1	56.9	126	25.5
183	0.302	0.174	49,695	82,500	31.3	52.4	127	15.2
147	0.420	0.122	55,002	94,901	25.8	43.2	159	19.0
215	0.482	0.171	48,611	90,233	26.4	40.2	163	10.3
222	0.586	0.118	53,440	104,653	21.4	41.4	174	11.8
137	0.624	0.162	54,297	110,770	18.0	27.0	207	4.1
135	0.788	0.128	55,960	120,773	17.5	27.3	192	3.5
Av.....	0.377	0.146	49,280	87,358	27.3	48.1	148	

TABLE 16.—*Mechanical Properties of Annealed Carbon Steels, High-phosphorus Series, Ladle B, Treated with* 0.20 *Per Cent. Silicon as* 50-*Per Cent. Ferrosilicon*

Heat Number	Percentage		Yield Point, Lb. per Sq. In.	Ultimate Strength, Lb. per Sq. In.	Per Cent. Elongation	Per Cent. Reduction of Area	Brinell	Izod Number, Ft.-lb.
	Carbon	Phosphorus						
200	0.132	0.147	42,500	63,781	36.0	65.2	109	44.7
107	0.162	0.149	46,875	72,968	35.6	59.6	119	35.3
142	0.186	0.124	44,059	67,008	35.8	61.7	119	41.9
201	0.236	0.153	43,307	72,079	34.1	60.4	130	34.1
220	0.254	0.161	46,420	74,466	31.7	56.1	126	18.5
183	0.288	0.166	48,231	77,735	32.6	55.0	130	22.2
147	0.424	0.129	56,623	95,422	24.3	40.1	163	12.7
215	0.468	0.169	51,821	91,143	27.6	42.9	153	8.2
222	0.566	0.119	55,226	103,578	22.6	36.7	174	10.0
137	0.632	0.168	60,417	114,383	18.8	30.7	199	3.2
135	0.788	0.133	52,061	112,453	16.9	22.7	217	2.3
Av.....	0.376	0.147	49,776	85,911	28.7	48.3	149	

steels, of steels containing on an average 0.14 per cent. phosphorus and treated with 0.15 per cent. zirconium, and of the same high-phosphorus steels not treated with zirconium. These curves are based on much more complete data than are given in Tables 15–17.

TABLE 17.—*Mechanical Properties of Annealed Carbon Steels, High-phosphorus Series, Ladle C, Treated with* 0.20 *Per Cent. Silicon and* 0.15 *Per Cent. as Silicon-zirconium*

Heat Number	Percentage		Yield Point, Lb. per Sq. In.	Ultimate Strength, Lb. per Sq. In.	Per Cent. Elongation	Per Cent. Reduction of Area	Brinell	Izod Number, Ft.-lb.
	Carbon	Phosphorus						
200	0.130	0.153	39,457	64,803	37.1	65.3	109	47.9
107	0.162	0.149	47,880	72,333	35.0	64.0	119	46.0
142	0.186	0.124	34,303	66,104	36.7	61.5	119	50.6
201	0.268	0.154	45,356	73,886	35.6	58.8	119	35.6
220	0.232	0.160	35,300	76,048	32.9	57.6	126	24.8
183	0.278	0.166	35,341	76,411	32.4	59.3	137	28.3
147	0.448	0.126	44,956	94,275	26.6	43.8	159	14.6
215	0.490	0.162	41,875	84,338	27.0	40.5	160	13.5
222	0.568	0.117	40,897	99,117	24.2	40.8	170	12.1
137	0.632	0.167	45,560	111,191	19.2	30.0	199	6.1
135	0.782	0.132	47,917	116,805	17.7	26.4	210	4.1
Av.....	0.380	0.146	41,713	85,028	29.5	49.8	148	

The effect of zirconium upon the Izod number is essentially a neutralization of the embrittling effect of phosphorus, as the Izod number of ordinary low-phosphorus steels is not appreciably affected. No evidence of actual chemical combination between zirconium and phosphorus has been obtained by chemical or microscopic examination. As notch toughness is intimately related to the properties of the intercrystalline material, there is reason to infer that zirconium in high-phosphorus steels favorably modifies the structure and composition of this material, possibly because of its concomitant action on oxygen and nitrogen.

MECHANICAL PROPERTIES OF HIGH-SULFUR ROLLED PLATE CONTAINING ZIRCONIUM

Table 18 gives the values obtained in a number of high-sulfur rolled plate and also the corresponding values for the low-sulfur rolled plate of heat number 331. The test data refer to longitudinal test pieces in the "as rolled" condition. The carbon content of all these heats lies between 0.32 and 0.43 per cent.

TABLE 18.—*Mechanical Tests on High-sulfur Rolled Plate, with and without Zirconium*

Heat Number	Per Cent. Added Zirconium	Carbon	Per Cent.		Sulfur (Evolved)	Yield Point, Lb. per Sq. In.	Ultimate Strength, Lb. per Sq. In.	Per Cent.		Izod Number Ft.-lb.	Brinell
			Silicon	Manganese				Elongation	Reduction of Area		
331	none	0.37	0.22	0.62	0.023	45,977	82,754	23.2	47.0	17.6	141
	0.19	0.36	0.19	0.62	0.022	29,364	79,841	26.2	46.5	19.4	131
	0.40	0.43	0.43	0.58	0.006	31,936	86,164	23.2	49.7	15.3	147
318	none	0.36	0.25	0.33	0.184	39,508	71,713	22.3	44.0	42.7	119
	0.24	0.36	0.27	0.32	0.183	29,631	71,128	24.9	35.7	34.4	112
305	none					(could not be rolled)					
	0.23	0.36	0.23	0.13	0.192	37,390	68,197	24.8	46.3	35.7	116
	0.40	0.37	0.40	0.13	0.184	35,415	69,642	20.5	39.9	40.4	124
286	none	0.36	0.44	0.25	0.112	46,366	75,087	23.0	39.6	25.8	
	0.42	0.40	0.44	0.26	0.096		72,698	21.7	43.4	25.9	
332	none					(could not be rolled)					
	0.25	0.34	0.22	0.14	0.278	23,712	67,619	25.0	40.2	23.3	107
	0.47	0.35	0.44	0.16	0.272	31,130	73,410	20.2	35.7	30.4	125
333	none					(could not be rolled)					
	0.22	0.32	0.20	0.15	0.256	22,712	66,312	25.4	41.8	24.3	106
	0.43	0.33	0.38	0.14	0.252	32,648	69,851	27.2	40.0	27.6	115

In Table 19 are given the ratios of longitudinal to transverse values for elongation, reduction of area, and Izod number. Yield point and ultimate strength are but slightly affected by directional differences and are not included. It is of much interest to observe the comparatively

TABLE 19.—*Ratios of Longitudinal to Transverse Properties of Rolled Plate, with and without Zirconium*

Heat Number	Per Cent. Added Zirconium	Per Cent. Elongation	Per Cent. Reduction of Area	Izod Number Ft.-lb.	Heat Number	Per Cent. Added Zirconium	Per Cent. Elongation	Per Cent. Reduction of Area	Izod Number Ft.-lb.
331	none	1.01	0.99	1.09	286	none	1.13	0.88	1.06
	0.19	1.14	0.94	1.23		0.42	1.07	1.41	1.33
	0.40	1.02	1.03	1.23					
318	none	1.02	1.58	2.03	332	0.25	1.18	1.91	1.44
	0.24	1.14	1.40	1.26		0.47	1.37	2.31	1.90
305	0.23	1.31	1.78	1.78	333	0.22	0.94	0.99	1.07
	0.40	1.30	1.82	1.51		0.43	1.73	2.67	1.60

excellent mechanical properties possessed by the high-sulfur zirconium-treated steels. Ductility is not materially lower than that of ordinary low-sulfur steels and the Izod number is considerably higher.

In order to obtain the full beneficial effect of zirconium on the hot-rolling properties of steel, it is not necessary to realize actual sulfur elimination. The first step in sulfur fixation by zirconium consists in chemical combination with the minute amount of iron sulfide present in all steels, then the remaining zirconium combines with the sulfur present as manganese sulfide. The latter process brings with it no particularly beneficial effects. The idea is first to remove the last traces of iron sulfide; this usually requires only a small amount of zirconium (0.15–0.20 per cent.). Next, to form zirconium sulfide at the expense of manganese sulfide and permit the former to slag off, provided actual sulfur elimination is sought.

Ingots containing 0.185–0.200 per cent. sulfur and only 0.15 per cent. manganese can be rolled to plate or sheet free from cracks and seams when the steel has been treated with 0.22 per cent. zirconium. On increasing the sulfur up to 0.260–0.290 per cent., the same thing can be done by the addition of 0.43 per cent. zirconium. The untreated ingots of these heats broke to pieces in every case on their first pass through the rolls.

MECHANICAL PROPERTIES OF HEAT-TREATED CARBON STEELS CONTAINING ZIRCONIUM

Carbon steels containing relatively small amounts of zirconium show, in the heat-treated condition, tensile properties closely approaching those of the well-known alloy steels. This effect of zirconium is especially noticeable when steel containing it is drawn in that range of tempering temperatures associated with the formation of troostite. The values given in Table 20 are typical of what may be obtained by the addition of 0.15 per cent. zirconium to a steel analyzing 0.70 per cent. carbon, 0.26 per cent. silicon, 0.62 per cent. manganese, 0.026 per cent. phosphorus, and 0.027 per cent. sulfur. The values in the last column were obtained on the same steel treated in a duplicate ladle from the same heat with an equivalent amount of ferrosilicon.

Figures 10 and 11 show how percentage elongation and percentage reduction of area are related to drawing temperature for both steels, untreated and zirconium-treated. A similar effect has been observed on the addition of zirconium to certain alloy steels, although this phase of the work is not yet ready for publication. The beneficial action of zirconium in heat-treated steels is probably caused by its reaction with oxygen and nitrogen and to its general deoxidizing and scavenging properties.

TABLE 20.—*Mechanical Properties of a 0.70-Per Cent. Carbon Steel, with and without Zirconium, Quenched from 825° C. in Water and Drawn at Temperatures Indicated*

	0.15 Per Cent. Zirconium	Without Zirconium
Drawing temperature, degrees C	375	375
Per cent. elongation	8.3	5.2
Per cent. reduction of area	23.3	6.6
Yield point, pounds per square inch	185,952	128,125
Ultimate strength, pounds per square inch	227,203	197,800
Brinell number	414	433
Drawing temperature, degrees C	412	412
Per cent. elongation	12.7	7.5
Per cent. reduction of area	45.8	22.9
Yield point, pounds per square inch	172,620	180,180
Ultimate strength, pounds per square inch	198,828	207,144
Brinell number	407	418
Drawing temperature, degrees C	440	440
Per cent. elongation	13.0	8.5
Per cent. reduction of area	46.0	30.0
Yield point, pounds per square inch	160,100	171,935
Ultimate strength, pounds per square inch	183,700	197,512
Brinell number	388	387
Drawing temperatures, degrees C	540	540
Per cent. elongation	18.0	14.7
Per cent. reduction of area	48.2	39.4
Yield point, pounds per square inch	122,220	128,890
Ultimate strength, pounds per square inch	140,704	147,007
Brinell number	288	298
Drawing temperature, degrees C	600	600
Per cent. elongation	21.2	19.6
Per cent. reduction of area	54.7	51.2
Yield point, pounds per square inch	105,632	110,164
Ultimate strength, pounds per square inch	125,085	127,663
Brinell number	233	238

SUMMARY

In this paper, there have been presented some of the results obtained in the experimental manufacture and testing of steels treated with zirconium. Data are given in support of a number of conclusions regarding the chemical behavior of zirconium toward the major impurities in steel, namely, oxygen, nitrogen, sulfur, and phosphorus, and the resultant effects produced on mechanical properties. It is shown that:

1. Zirconium combines chemically with oxygen, nitrogen, and sulfur, in the order given, and is able to neutralize the embrittling effect of phosphorus in whole or in part.

2. A relatively small addition of zirconium makes possible the satisfactory rolling of steels containing as high as approximately 0.30 per cent. sulfur, which without zirconium treatment break to pieces on the first pass through the rolls.

3. Plain carbon steels treated with approximately 0.15 per cent. zirconium exhibit, in the heat-treated condition, tensile properties that closely approach those of alloy steels, especially in that range of drawing temperatures between 300 and 450° C.

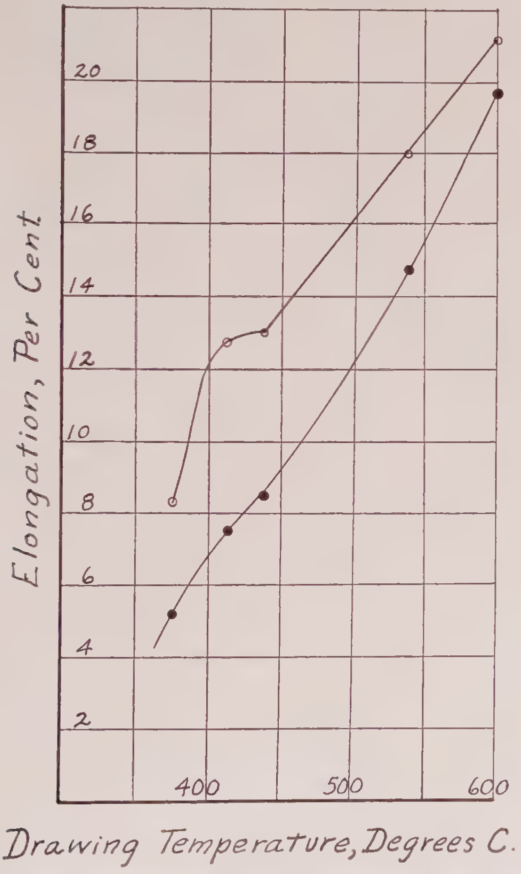

FIG. 10.—RELATION BETWEEN PER CENT. ELONGATION AND DRAWING TEMPERATURE FOR 0.70-PER CENT. CARBON STEELS QUENCHED FROM 825° C. IN WATER; UPPER CURVE, WITH 0.15 PER CENT. ZIRCONIUM; LOWER CURVE, NO ADDED ZIRCONIUM.

4. Zirconium reduces the total oxygen content of steel and operates to prevent the occurrence of emulsified slag and "banded" structure.

5. Zirconium combines with the nitrogen dissolved in molten steel to form a crystalline microconstituent of a bright lemon-yellow color, which is, in large part, removed from the steel while the latter is still molten.

6. Zirconium forms, with the sulfur content of steel, an acid-insoluble sulfide and eliminates from high-sulfur steels the last traces of iron sulfide.

Besides the work reported in this paper, the Electro Metallurgical Co. has accumulated during a period of several years information on the commercial application of zirconium; these results it is intended to publish at some future date.

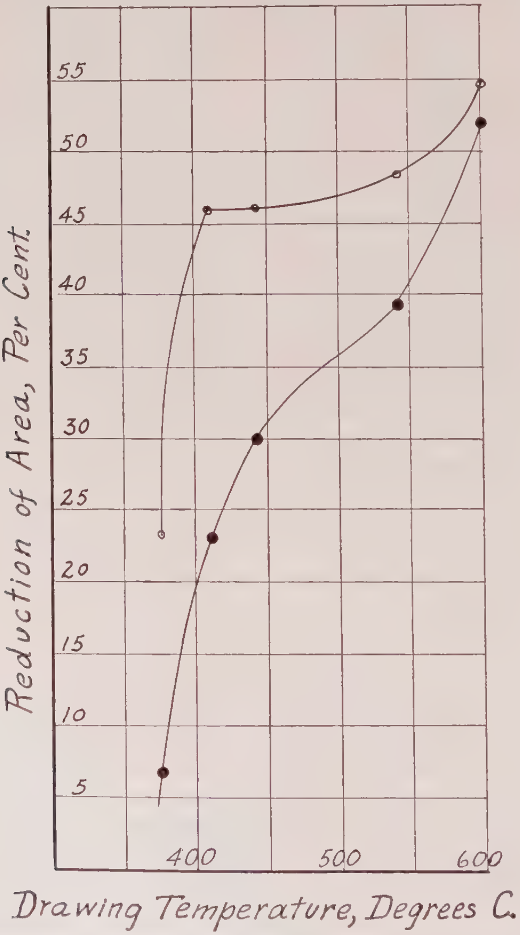

Fig. 11.—Relation between per cent. reduction of area and drawing temperature for 0.70-per cent. carbon steels quenched from 825° C. in water; upper curve, with 0.15 per cent. zirconium; lower curve, no added zirconium.

Acknowledgments

In addition to F. M. Becket, directing head of the research staff of the Electro Metallurgical Co., the author wishes to acknowledge his indebtedness to the other members of the staff for their valuable suggestions and assistance; particularly to J. H. Critchett, W. L. Harbrecht

J. A. Holladay, and C. E. MacQuigg. Mr. Critchett gave valuable assistance in connection with the design of the electric steel furnace, and has made many useful suggestions throughout the progress of the investigation. For the past year, Mr. Harbrecht has operated the electric steel furnace, and in numerous ways has facilitated the progress of experimental steel production. All of the chemical analyses reported herein were carried out under the supervision of Mr. Holladay, to whom is due much credit for developing a method for the accurate determination of zirconium in steel and in zirconium ferro-alloys. In connection with the metallographic phases of the work, Mr. MacQuigg's experience and suggestions were of material help.

DISCUSSION

I. A. Billiar, Burnham, Pa. (written discussion).—This paper shows clearly the effects brought about by the presence in steel of small percentages of zirconium. Some of these effects have been noticed by us in the manufacture of acid and basic open-hearth steels. All our steels are of the "killed" type and the heats vary from 50 to 90 tons. The silicon-zirconium alloy is added in the ladle in amounts from 0.04 to 0.08 per cent. After the zirconium is added, the steel seems to become a deep pink, which color persists quite clearly during the first half of the teeming period—about 35 min.

The recovery of zirconium ranges from 45 to 75 per cent. with an average recovery of 55 per cent. The average recovery of silicon in heats made with the addition of silicon-zirconium was higher than the average recovery of silicon in heats made with the addition of titanium and 50-per cent. ferro-silicon, which seems to indicate the greater deoxidizing power of zirconium.

We found very little difference between the mechanical properties of steels treated with zirconium and steels treated with titanium and 50-per cent. ferro-silicon. No difference was observed in the drop test on finished products but a slight decrease in scrap produced during rolling was observed in heats that had been treated with zirconium; this may or may not have been due to the addition of zirconium. All our steels are made on a quality rather than a quantity production and it was not expected that a marked change would be shown by the addition of 0.04 to 0.08 per cent. zirconium.

Besides the open-hearth commercial steels, we have made crucible steels with sulfur up to 0.25 per cent. and additions of zirconium up to 1.0 per cent. The results of tests on these steels check closely the results shown by Mr. Feild regarding the actual elimination and "fixation" of sulfur and beneficial effect of zirconium on the hot working of high-sulfur steel.

H. W. GILLETT, Ithaca, N. Y. (written discussion).—This paper presents some real evidence for the value of zirconium. The reduction in oxygen content, the fixing of sulfur, the decrease of banded structure, the Izod tests on high-phosphorus steels, and the ability to roll high-sulfur steels, are all quite definite.

The theories on the cleansing and the coagulating effects of zirconium on emulsified inclusions are interesting. It would have been helpful had some unetched photomicrographs been shown to back up the argument as to greater cleanliness of zirconium-treated steel. It would also be interesting to know whether the zirconium-sulfide particles remaining in the steel are plastic at rolling or forging temperatures and draw out into threads or plates like manganese sulfide.

As the radius at the base of the notch in the Izod test bar is greater than the ¼ mm. or 0.01 in. in the British Engineering Standards test piece, and generally used in this country, the Izod figures may be 50 to 100 per cent. higher than they would be on standard bars, with the results of which these figures might be compared. The value of the comparison between zirconium-treated and untreated high-phosphorus steels is, however, probably not decreased by the use of the blunter notch.

In general, the thoroughness of the work recorded in the paper deserves high commendation, but the fatigue tests deserve equal condemnation; they prove nothing. Notwithstanding the recent work of Moore, McAdam, and various British investigators, what are now accepted as the essentials of endurance testing have been disregarded.

In the first place, Moore's work at the University of Illinois has shown that the localized stresses at the collets in an unnecked bar are great and cannot be calculated. Hence the calculated maximum fiber stress figures mean nothing in absolute values and little in comparative values. It is not clear just how the endurance limit was calculated from the Moore and Seely formula,[4] as the coefficient B in that formula must be obtained from a plot of tests of more than one bar of a given material. But at any rate the nominal maximum stress on an unnecked bar is not the true maximum stress.

The substitution of an incorrect stress or of a coefficient derived from it in a formula will necessarily give incorrect results. Moore states that the formula "applies to members with all corners at points of high stress generously filleted." It does not apply to an unnecked bar.

Moreover, the formula itself, suggested by Moore and Seely in 1915–16, involves the assumption that failure will ultimately occur at any stress however small, and Moore's later work has led him to the exactly opposite conclusion, *i.e.* that there is a true endurance limit.

Moore's old formula does not give the endurance limit as this is now defined. It gives instead a stress at which the specimen should withstand a certain number of alternations, were the formula accurate. This number must be assumed at some definite value and inserted in the formula, and Feild does not state the number assumed.

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[4] *Proc.* Am Soc. Test. Materials (1915) **15,** II, 437; (1916) **16,** II, 471.

Moreover, the best endurance tests, made with all proper precautions as to necking of bars, etc., show so much variation between duplicate tests that calculation of an endurance limit from a single bar, even were there an accurate formula for such calculation, would be absurd.

That the endurance limit calculated on such a basis shows no relationship to the elastic limit is not surprising, for even when a series of tests are made from which a true endurance limit may be found, the proportionality found by all recent workers is between endurance limit and tensile strength, not elastic limit. All recent work leads to the conclusion that properly made endurance tests on steels of this class would show endurance limits of 40 to 50 per cent. of the tensile strength. One would expect that these specimens, properly tested, would run from about 22,000 lb. per sq. in. on No. 120 to 50,000 lb. per sq. in. on No. 217, and would show no more difference between zirconium and untreated steels than the irregular and negligible variation in tensile strength.

If the calculated endurance limit is therefore neglected, and only the life given by bars tested at the same nominal stress be considered, it might be thought that the data do show a faint indication that the material treated with 0.04 per cent. zirconium was a shade better than untreated material. But, considering the lack of record of any duplicate tests and the incorrect form of test bar used, even this conclusion rests on a very rickety basis.

Some years ago I wasted a good deal of time in endurance tests on unnecked bars, which showed absolutely nothing, while the samples tested in necked form gave results in agreement with those of recent work by Moore and others. This experience has prompted the above criticism.

That on the other points covered by Feild a very strong case has been made out for benefits due to zirconium additions cannot be denied. But from the results of a long series of nickel-silicon steels with and without zirconium, in which no beneficial effects could be traced to zirconium, it would seem necesary that the value of additions of that element to any particular alloy steel, for example, be proved for the particular case.

Alexander L. Feild.—That the practical value of additions of zirconium to any particular type of steel should be proved for that particular case has always been a self-evident proposition to the writer, and was one of the premises on which the present work was based. A sort of corollary to the proposition, namely, that the failure of an investigator to obtain a favorable effect by adding zirconium to a given type of steel, nickel-silicon steel for example, furnishes no valid grounds for adverse conclusions of a generalized nature, does not appear to have been emphasized particularly in previous investigations.

The use of a notch with a radius of ⅔ mm. at its base in the Izod tests was adopted in our investigation only after a careful study of the subject. At the time of its adoption the test piece having a notch with a radius of ¼ mm. had been only recently standardized by the British Aeronautical Inspection Directorate. It was next adopted by the Air Board and still later by the British Engineering Standards Association. The types of test piece then most commonly used were prepared either with the Izod-Mesnager notch having a radius of 1 mm., or with the Charpy notch having a radius of ⅔ mm. While it is true that the British standard notch referred to above has within the last few years been widely adopted in this country for Izod tests, its choice was originally made quite arbitrarily, and not without considerable criticism. There is at present no standard impact test piece recognized by the American Society for Testing Materials. The anomalous character of the situation in this country is accentuated by the fact that, while the British standard notch has been quite widely adopted for Izod tests, Charpy tests are still commonly made with a blunter notch, varying usually from ½ to 1 mm. in radius at the base.

Philpot,[5] in his investigation of notched bars, has given comparative data that bear directly on the point in question. His conclusion is that the notch with ¼-mm. radius gives, in the case of square Izod test pieces, values approximately 7 ft.-lb. lower than are obtained under similar conditions with a notch of ⅔-mm. radius. The data for the 10 heat-treated steels on which this conclusion is based cover a range of values from 58.3 to 9.4 ft.-lb. in the case of the former type of notch, and from 65.1 to 14.9 ft.-lb. in the case of the latter type. If Philpot had investigated steels of lower Izod number, it seems certain that the advantage of the latter type of notch would have made itself felt.

In view of this evidence it would appear that the notch of ⅔-mm. radius, such as was used in our investigation, gives the wider scale of measurement and, in the range of the more brittle steels, makes it an easier experimental problem to measure and observe differences in Izod brittleness. Incidentally, it is the type of notch recommended by the Tinius Olsen Testing Machine Co. for square Izod bars.

If any impact values on annealed carbon steels covering a satisfactory range of carbon content have been published, which show the use of the British standard notch on square Izod test pieces, I would be glad to learn of them. The values given recently by Langenberg[6] seem to come nearest to filling the requirement, and these were made on Charpy bars with a notch of ⅔-mm. radius.

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[5] *Proc.* Institution of Automobile Engineers, (1917–18) **12**, 235.

[6] *Chem. & Met. Eng.* (1921) **25**, 910.

It probably should have been pointed out that the tensile and Izod values given in Tables 11–13 and 15–17, inclusive, represent the average values obtained on five tensile test pieces and eight Izod notches, respectively. These thirteen test pieces drew heavily on the available material of the ingot, so that of sound material there was left for fatigue tests only enough for one fatigue bar. Of course, this part of the original 1-in. round bar could have been forged down to supply additional fatigue bars. It was decided, however, not to do this, in order to have all tests conducted on bars that had undergone the same percentage of reduction from the original ingot.

The investigation was not considered to be primarily a study of endurance limit, but it was desired to know whether zirconium had any appreciable effect on endurance limit. Therefore, at a time considerably prior to the publication of Bulletin 124 of the University of Illinois Experiment Station,[7] or of McAdam's paper,[8] a schedule of tests was begun on a Farmer machine. Before they were made, however, a number of duplicate tests were run on straight, unnecked bars, using for these preliminary runs a group of miscellaneous steels available outside of the investigation proper. The values given below were obtained.

Test No.	Maximum Fiber Stress, Lb. per Sq. In.	Revolutions, to Failure	Test No.	Maximum Fiber Stress, Lb. per Sq. In.	Revolutions to Failure
1a	27,500	1,758,900	4b	37,500	503,200
1b	27,500	1,727,600	5a	37,500	994,900
2a	27,500	1,951,700	5b	37,500	911,600
2b	27,500	2,647,900	6a	40,000	426,900
3a	37,000	2,701,200	6b	40,000	648,000
3b	37,000	2,132,200	7a	40,000	352,700
4a	37,500	327,200	7b	40,000	363,600

The agreement between duplicate values is not remarkable, but it seems to be at least as good as that which is often obtained with necked bars, judging from the data given in Bulletin 124 by Moore and Kommers.

It is difficult to conceive of a machine part loaded under more ideal conditions than those which characterize the Farmer machine. Furthermore, the maximum fiber stress that exists in the region between the two loads is capable of a much more exact theoretical calculation than in the case of the necked bar. The only difficulty is the existence of localized stresses that cannot be readily calculated at the points of application of the loads. In order to prove their magnitude, it would be necessary to run a parallel series of fatigue tests on both straight and necked bars. I cannot find any record of such a series of tests in the reports by Moore and his associates, nor, in fact, any record whatever of tests on straight bars. A straight bar does not always break at or near the point of application of one of the loads. Sound bars occasionally break halfway between the loads. This fact, together with the general concordance of our data, gives us confidence that our fatigue measurements are of practical value.

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[7] H. F. Moore and J. B. Kommers, Oct. 24, 1921.

[8] Endurance of Steel under Repeated Stresses," *Chem. & Met. Eng.* (1921) **25**, 1081.

The formula of Moore and Seely, which was used in calculating the endurance limit, is

$$S = \frac{B}{(1-Q)N^{1/8}}$$

where S = applied fiber stress and N = number of revolutions. In the case of the Farmer machine, Q is equal to -1. Substituting in this equation the observed number of revolutions to failure, the fiber stress applied determines the constant B completely. The following values for B were obtained for the steels listed in Table 14:

HEAT NO.	VALUE OF *B*	HEAT NO.	VALUE OF *B*	HEAT NO.	VALUE OF *B*
120A	294,400	208A	362,500	204A	342,200
B	282,700	B	335,300	B	320,100
C	296,500	C	298,100	C	322,800
202A	363,200	206A	341,200	211A	351,900
B	295,000	B	336,400	B	343,400
C	295,600	C	303,200	C	380,400
205 A	336,700	203 A	375,700	217 A	378,100
B	323,300	B	337,900	B	344,900
C	310,300	C	374,600	C	348,700

	LADLE A	LADLE B	LADLE C
Average value of constant *B*	349,500	324,300	325,580

The endurance limit was then calculated from the same formula by substituting therein the appropriate value of B and putting N equal to 10,000,000. It was thought at the time that this value of 10,000,000 revolutions was low, if anything, but the more recent work of Moore and his associates indicates, on the basis of data obtained on necked bars, that the endurance limit could be more correctly defined as the stress a bar will stand for 5,000,000 revolutions. The same conclusions, however, would have followed from Table 14 if N had been taken at 5,000,000, or any other value, since they are already implied in the values of B given above. Moore and Seely gave 250,000 as a tentative value of B for structural steel and soft machinery steel and 350,000 for 0.45-per cent. carbon steel.

The old formula of Moore and Seely appears to be as satisfactory as any that has been proposed for bringing to a common basis of comparison fatigue data obtained at different loads. It is just as applicable today as it was when proposed, with the added provision that it should not be used for assumed values of N that lie beyond the knee on the S-N curves. It is the equation of the upper part of the S-N curve where S varies with N, and this variation is linear on a logarithmic scale of coordinates.

Doctor Gillett's expectation that steel No. 217 with its 0.72-per cent. carbon content would, under tests with necked-bar practice, show an endurance limit of about 50,000 lb. per sq. in. receives absolutely no support from the most recent work of Moore and Kommers. Bulletin 124 contains the results of tests on an annealed steel of 0.93-per cent. carbon content (No. 6). The observed endurance limit was 30,500 lb. per sq. in. in the Farmer machine, and 28,500 lb. per sq. in. by the "rise of temperature" test. This steel then falls quite in line with our own. The remaining steels of Bulletin 124 are either normalized (air cooled), or quenched and tempered, or subjected to cold work, except one of 0.20 per cent. C and 0.09 per cent. S content, which is also tested in the annealed state. This high-sulfur steel (No. 50) gave an endurance limit of 25,000 lb. per sq. in. when annealed at 1550° F.

No claims are made that zirconium is a "cure all." This element is an excellent scavenger and deoxidizer and our own conclusions regarding its effect on oxygen, nitrogen, and sulfur have been amply confirmed in commercial practice.

George F. Comstock,* Niagara Falls, N. Y. (written discussion).—It is with no intention of detracting unduly from the general value of the author's experiments that the following criticisms are offered of some of the methods used. While there can be no question as to the importance of the facts reported in the paper, the writer would like to point out several instances where a different interpretation of the results would seem more probable.

The method of dividing all the heats was unfortunate, for by pouring the various ladles from a heat serially, the steel could not be expected to be identical in each. The first ladle receives the steel that was next to the slag in the furnace and the last ladle receives the steel from the bottom of the furnace. It is well known among melters that there is apt to be a great difference in quality between these portions. With the carbide slag used on these little heats, the differences in oxygen, nitrogen, and sulfur content between the two ladles poured might just as well be due to the more efficient action of the furnace slag on the metal that entered the first ladle as to the different additions used in the ladles. Certain irregularities might also occur which would occasionally make the first ladle of steel inferior in quality to the others. In order to insure the same quality of steel in all ladles from a heat it is necessary to tap into the ladles simultaneously by means of a divided spout. This method has been used in commercial practice and is the only safe one for accurate experimental work of this kind.

The paper gives no information as to the presence or absence of zirconium nitride in the various alloys used for the addition of zirconium to the steel. A specimen of commercial zirconium-silicon alloy from the same manufacturer that supplied the author's material has been examined by the writer; and although the fracture is wholly silvery, a polished section shows abundant needles of a yellow constituent. In fact, under the microscope the alloy looks like a mixture of metallic silicon and zirconium nitride, the color of the latter not being seen in a fracture because the silicon is so soft and weak that the alloy always breaks through it. This is analogous to the black fracture produced by a little graphite in steel or iron. Our experience in the manufacture and use of zirconium nitride indicates that the process described in Mr. Beckett's patent on zirconium-silicon might easily result in considerable formation of the nitride; we have also found that this compound is capable of acting as a deoxidizer. It would, therefore, be interesting to know how much of the zirconium in the alloys used by the author was in the nitride form. The analyses in Table 1 show from 1 to 18 per cent. unaccounted for in the silicon-zirconium series; it may be that most of this is nitrogen.

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* Titanium Alloy Mfg. Co.

If the presence of nitride in the alloy used is confirmed, its presence in the steel treated with this alloy does not indicate any action on the nitrogen in the steel, as claimed by the author. The difference in nitrogen content in the steel would then have to be explained entirely by the difference in the quality of metal entering the various ladles, as noted above. In any event, the nitride crystals would not be expected in the low-carbon steels made without carbide slag in the furnace, because the nitride would be oxidized by these steels. The oxidation of the nitride is undoubtedly the true explanation of the phenomenon illustrated by Fig. 2, the so-called slag inclusion being merely the oxidized shell of the nitride crystal.

The author lays considerable stress on the elimination of banded structure and emulsified inclusions, which he considers related. It is possible that inclusions have an effect on the banded structure, but its chief cause is dendritic segregation in the ingot; this has been shown[9] to depend directly on the pouring temperature. The lack of banded structure in the zirconium-treated ingots probably indicates that they were poured at a lower temperature, rather than that the zirconium removed this structure. The author's Fig. 1, illustrating zirconium nitride, certainly shows a banded structure in the steel. No conclusions should ever be drawn from the supposed presence or elimination of inclusions emulsified so fine as to be incapable of being photographed at 500 diameters. Metallographists must realize that the methods of preparation of their specimens are not perfect, and with a material, like soft steel, that is so readily scratched, rusted, and tarnished, extreme caution is required in studying fine inclusions. Perhaps the effects noticed along this line were due to a slight hardening of the ferrite by zirconium or silicon which was retained in greater amount in the zirconium-treated steel, so that it was more easily polished and revealed fewer polishing defects at high magnifications. This explanation is supported by the Brinell tests on heats 120, 202, and 205.

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[9] Fred. G. Allison and Martin M. Rock: Studies of Macrostructure in Cast Steel. *Chem. & Met. Eng.* (1920) **23**, 383.

The most interesting facts brought out by the author's mechanical tests are the low yield point resulting from comparatively large zirconium additions, and an improved impact resistance brought about by zirconium in high-phosphorus steels, which without zirconium were much more brittle than ordinary steels. In attempting to explain these effects, the author fails to consider the presence of the nitride inclusions or crystals; yet is not the presence of these inclusions at least as plausible an explanation for both phenomena as the theories advanced in the paper? The yield-point effect seems to have been quite erratic, for in some heats the zirconium portion gave a yield point nearly the same as the regular portion, while in other heats it was only half or two-thirds as great in the zirconium portion as in the regular. This is as might be expected from an effect due to inclusions. It seems very probable that an excessive number of zirconium-nitride crystals in the steel would cause a more early and gradual slipping of the metallic particles during the progress of a tensile test by concentrating the stress between and at the corners of the inclusions. After the metal started to flow, this effect of the inclusions would be hidden, and there would not necessarily be any loss in ductility unless they were segregated. This action of zirconium is similar to that of aluminum in reducing the yield point of steel, as reported by R. A. Bull,[10] read before the Institution of British Foundrymen. As it is well known that the use of aluminum in steel involves the formation of alumina inclusions, it is evident that the reduction of the yield point in that case is due to inclusions, and the same effect might be expected from the presence of too many zirconium-nitride crystals.

In regard to the impact tests, the author's suggestion of a relation between notch toughness and the properties of the intercrystalline material is hard to accept, in view of the well known fact[11] that impact fractures in sound steel at normal temperatures occur through and not between the crystals. The low impact results obtained from high-phosphorus steels are due to more distinct or longer cleavage planes in the crystals than to weakness between the crystals. In the zirconium-treated steels, with the numerous nitride crystals present, would it not be reasonable to assume that these crystals interrupted or broke up the large cleavage planes to a certain extent, so that the impact fracture had to change its direction more often, and for that reason more energy was consumed in the sudden failure than would have been consumed without the inclusions? Brearley's work[12] on the testing of welds by impact might be mentioned in this connection, for he showed that an interruption in the continuity of the steel caused much higher results in the notched-bar impact test.

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[10] "Producing Steel Castings." *Foundry* (1923) **51,** 571.

[11] Walter Rosenhain and D. Hanson: Intercrystalline Fracture in Mild Steel. *Jnl.* Iron & Steel Inst. (1920) **102,** II, 36.

The assumption that iron sulfide occurs in all steels seems somewhat radical and should require proof. In steels where this compound is troublesome, it would undoubtedly be removed just as effectively by a slight increase in the manganese content as by an addition of zirconium. It is interesting, nevertheless, to know that zirconium has an effect on the rolling quality of high-sulfur steel, although its action as a deoxidizer may have more to do with this than the author indicates.

The results given on heat-treated steels are not conclusive, and it should be pointed out in connection with the curves in Figs. 10 and 11, that in four out of five cases the yield point and strength of the more ductile zirconium-treated samples were lower than those of the untreated. Some slight difference in chemical composition or manipulation during heat treatment may easily have accounted for the small variations in properties between these few zirconium-treated and regular samples.

In spite of the writer's disagreement with the author on these various points, most of which are matters of interpretation of the observations, he is not inappreciative of the value of the paper. The record of results obtained in the numerous experiments is interesting, and should be helpful to future investigators of this subject.

ALEX. L. FEILD.—The silicon-zirconium alloys utilized in the investigation have an exceedingly low nitrogen content. For example, silicon-zirconium alloy No. 2D (Table 1) showed, on analysis, less than 0.003 per cent. nitride nitrogen. The method of analysis employed involves decomposition of the alloy by the combined action of hydrochloric and hydrofluoric acids and is the best procedure so far devised. Microscopic examination of polished sections of the alloys fails to show the presence of zirconium nitride or any constituent remotely resembling it. Possibly Mr. Comstock has been misled as to the source or the type of zirconium alloy he has examined and to which he refers. Several types of zirconium alloys have been produced within recent years by the Electro Metallurgical Co.

The iron, titanium, and carbon contents of silicon-zirconium alloy No. 2C (Table 1) are not shown. They are 16.72, 0.26, and 0.55 per cent., respectively. Upon inserting these values in the table, the following are the totals of the percentages there given:

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[12] "The Welding of Steel in Relation to the Occurrence of Pipe, Blowholes, and Segregates in Ingots." *Jnl.* Iron & Steel Inst., (1921), **103,** I, 27.

Alloy	Total
No. 1	96.53
No. 2A	99.85
No. 2B	98.86
No. 2C	98.98
No. 2D	99.02

The paper states that there was no increase in the number of zirconium-nitride crystals in the steel beyond 0.15 per cent. added zirconium; therefore, these crystals could not have existed to any appreciable extent in the alloy itself. The nitrogen content of the steel before treatment is the determining factor, a fact which was established by increasing the amount of added zirconium up to 1 per cent. and above.

Mr. Comstock does not approve of the procedure of rapidly pouring several ladles in succession from each heat, because the metal from the top of the bath is apt to be lower in impurities than the metal at the bottom of the furnace, due to the refining action of the carbide slag. Such differences in composition as do exist are entirely negligible in comparison with the changes in composition brought about by zirconium treatment. Nevertheless, to avoid being misled by any such errors, the order in which the ladles were poured was arranged in such a way that these differences in bath composition operate in a direction that opposes the effects of zirconium. Data on the elimination of oxygen, nitrogen, and sulfur were obtained by comparing the second, or *B*, ladle, treated with ferrosilicon, with the third, or *C*, ladle, treated with a silicon-zirconium alloy. If there does exist any significant vertical gradient of chemical composition through the bath, then the *B* ladle (ferrosilicon-treated) should have been, prior to pouring, freer from oxygen, nitrogen, and sulfur than the *C* ladle (silicon-zirconium-treated) which followed it.

With respect to Fig. 2, where a zirconium-nitride crystal having the form of a perfect cube is embedded in a slag inclusion, Mr. Comstock concludes that the surrounding slag is "merely the oxidized shell of the nitride crystal." Such an explanation fails to account for the sharp corners of the embedded crystal, and that the enveloping slag is certainly not zirconium dioxide; and, it does not account for the fact that the phenomenon is a casual one and is observed under the microscope in the immediate vicinity of crystals free from adhering or enveloping slag, which is just as apt to be of the sulfide as of the siliceous type.

The several ingots from any given heat were poured at the same temperature, as judged by a skilled operator. The effect of zirconium on banded structure is not due to variations in the extent of dendritic crystallization in the original ingot, brought about by variations in pouring temperature. The contention that such is probably the case receives no support from the banded structure of the steel shown in Fig. 1. This particular photo-micrograph was selected solely to illustrate in a single section zirconium-nitride crystals exhibiting all four of the possible shapes—cube, triangle, rectangle, and hexagon. The particular specimen in question was taken from a bar of 0.70-per cent. carbon steel that had undergone 87 per cent. reduction by swaging, the final 62 per cent. reduction occurring in the cold. Naturally, a banded structure showing granular pearlite is to be expected in this case.

Mr. Comstock says that "no conclusions should ever be drawn from the supposed existence or elimination of inclusions emulsified so fine as to be incapable of being photographed at 500 diameters." Doubtless the statement to which he refers is "Photomicrographs illustrating this effect of zirconium are difficult to obtain, for the emulsified particles are barely visible at 500 diameters." These inclusions have been photographed and will be discussed in detail in a future publication. The effect in question is not due to any hardening effect of zirconium on the ferrite. Annealed carbon steels containing 0.60 per cent. zirconium, added as a silicon-zirconium alloy, possess no higher Brinell number than corresponding steels of equal silicon content but containing no zirconium.

The effect of zirconium on yield point is not due to the presence of zirconium-nitride crystals. The effect, for instance, is observed for the steels of heats 120 and 202 (Table 13) which contained no nitride crystals. The magnitude of the effect bears no relation to the number of crystals present; but would appear to depend on the extent of nitrogen elimination more than on any other factor. Mr. Comstock compares this action of zirconium on yield point to that of aluminum, and cites an article by R. A. Bull. That article gives no experimental data regarding the effect of aluminum on yield point. Our own work has shown that there is no parallel whatever between zirconium and aluminum, or the slightest suggestion of an analogy. The observed effect of zirconium here is due to nitrogen elimination. Additions of aluminum do not affect the nitrogen content.

Zirconium-nitride crystals occurred to the same extent in the low-phosphorus series as in the high-phosphorus series of steels. Hence, Mr. Comstock's explanation that the effect of zirconium on the Izod number of the latter is due to the presence of the nitride crystals does not appear to be reasonable. No effect on Izod number was observed in the low-phosphorus series.

The remarks of Rosenhain on fractures were made with reference to failures occurring in dead-soft boiler plate.

The well-known law of mass action requires that all steels contain at least a minute amount of iron sulfide. As zirconium has a greater affinity for sulfur than does manganese, zirconium treatment is much more efficacious than additions of ferromanganese in dealing with high-sulfur steels. Residual iron sulfide is much more quickly brought down to the point where it exerts no harmful effect during rolling or forging.

Deterioration of Malleable in the Hot-dip Galvanizing Process*

By W. R. Bean† Naugatuck, Conn.

(New York Meeting, February, 1923)

Probably few, if any, of the many serious problems confronting malleable foundries have been more difficult of solution than the question as to why malleable that is ductile, black in fracture, and normal in all respects before galvanizing should, by that process, be rendered brittle, brilliantly crystalline or "white" in fracture, and of low resistance to shock.

So far as we are aware, the technical literature contains no data or discussions on this subject, except two references[1,2] of a general character. It is apparently a question that every one concerned, producer and consumer alike, has been willing to evade as far as possible. Various theories as to the cause of the deterioration have been given but such checking of these theories as we have done indicates that there was no sound basis for most of them.

This work on the galvanizing of malleable was first undertaken in 1916, and has continued until the present time. We now feel that for all practical purposes we have reached a solution and are glad to be able to offer it in a form that can be readily applied to production processes.

It is impossible to give here a detailed account of the early work done on this problem. A great number of tests were instituted and an enormous amount of work was done, practically all of which yielded negative, or vague and inconsistent results. The investigation that forms the basis of this paper has yielded, so far as we know, the first and only definite information on the subject obtained by any one.

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* The term "malleable" is here used in the substantive sense instead of malleable iron, malleable cast iron, or any of the other terms applied at times to this product.

† With the coöperation of the staff of the Research Department, Eastern Malleable Iron Company.

[1] Bean, Highriter, and Davenport: Fractures and Microstructures of American Malleable Cast Iron, *Trans.* Am. F'drymen's Assoc. (1920) **29**, 327.

[2] H. A. Schwartz: American Malleable Cast Iron. *Iron Trade Rev.* (1921) **69**, 615.

As a matter of record we may say that the early work was divided largely into two classes: (1) Investigations of the tendency to deterioration of materials of different compositions; (2) investigations of the method of carrying out the galvanizing process. Work along both of these lines was, to a large extent, carried on simultaneously. Efforts to correlate composition and susceptibility resulted in the vague conclusion that susceptibility of malleable to deterioration seemed to be greater with silicon contents of 1.00 per cent. and above.

In so far as the method of carrying out the galvanizing process is concerned, the work tended to show that deterioration was more severe when castings were quenched in cold water immediately after removing from the zinc than if warmer water were used or if time elapsed between removal from zinc and quenching. In all of this work, however, the results were far from conclusive and seemed quite inconsistent with such knowledge as we had. It was impossible to predict whether or not a certain iron would withstand the process, and the process could not be modified to such an extent as to eliminate the trouble.

Outline of the Present Research

General Statement

After all this work had been found to be largely futile, it was considered advisable to make a fundamental investigation that would take into consideration every possible factor in the composition and annealing of the iron. This was to be carried out by preparing various sets of alloys, each set having all but one of the chemical elements fixed at a normal value, this one element being varied throughout the entire range that would be encountered in practice. These sets were then to be divided into groups and given various annealing treatments, specimens of all compositions and anneals, however, being subjected finally to one standard galvanizing treatment. In carrying out this plan, only two sets of alloys were prepared, the first being one in which the phosphorus content varies from the lowest possible to an excessive value; the second, a set in which both the silicon and phosphorus contents varied simultaneously, or rather three sets, each containing a different phosphorus content in which the silicon varied over a given range. It was found unnecessary to subject the specimens to more than one kind of anneal for the cause of the deterioration lay in the composition rather than in the annealing treatment.

Methods of Testing

For all of the earlier work on this problem, the tensile test was used as the criterion of quality and deterioration in galvanized material. Frequently, it was noticed that there was relatively slight difference in yield point, ultimate strength, or elongation between the ungalvanized malleable and the same material in the brittle condition after galvanizing. Repeated tests frequently yielded inconsistent results and, in general, it was evident that the tensile test was not sufficiently sensitive for the particular type of deterioration encountered here. Before proceeding with any further researches, it was obviously necessary to adopt a more sensitive form of test, and to that end a preliminary investigation was conducted on the Charpy tensile impact, Humfrey static notched-bar and Olsen single-blow impact (Izod) machines.[3] The results of the Charpy test proved to be vague and inconsistent. The Humfrey and Olsen tests were of greater value and, after due consideration, the Humfrey static notched-bar test was adopted and used throughout this research. The Humfrey test, although of rather recent origin,[4] is by no means obscure or unknown. It has distinct advantages over the single-blow impact machines[5] and as it measures the ability of a material to withstand slow bending it is an excellent means of detecting galvanizing brittleness in malleable. The test specimens are machined to size and notched with a standard, round-bottom, V notch. One end is rigidly gripped in a vise while a bending force is applied to the other end by a series of levers and springs, somewhat as indicated in Fig. 1.

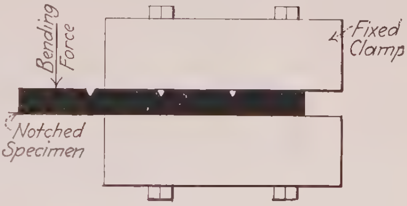

FIG. 1.

An ingenious device attached to the machine gave an autographic record of the test, in the form of a curve, from which may be read and computed: the maximum bending moment, in foot-pounds; the angle at which this occurred, in radians or degrees; the total bending angle, in radians or degrees; and the total energy absorbed in breaking the specimen. A mechanical integrator attached to the machine gives this last quantity directly, in foot-pounds. By an inspection of the curve produced by the autographic device, the ductile material may be distinguished from the brittle. The data relative to the bending angle are, of course, criteria of ductility, while the data relative to energy absorbed in breaking, *i.e.*, total amount of work done on the specimen, are criteria of the strength of the material. The standard specimen affords material enough for three notches so that three complete sets of values are obtained from one specimen.

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[3] Courtesy of Dr. F. C. Langenberg, Watertown Arsenal; Bureau of Standards; Tinius Olsen Testing Machine Co.

[4] Static Notched-bar Testing Machine. *Chem. & Met. Eng.* (1920) **23,** 1180.

[5] Recent Researches on Static Notched-bar Test. H. Holz. *Chem. & Met. Eng.* (1922) **26,** 941.

As a check on the Humfrey tests, one series of specimens in this research was tested by the Olsen single-blow impact (Izod) machine. In this test, the specimen (machined and notched) is clamped in a vise with one end extending as a cantilever, and broken by a pendulum hammer. The energy absorbed in breaking the specimen is measured by the amount of swing of the pendulum after the blow has been delivered, *i.e.*, in a manner similar to the Charpy measurement.

As has been stated, both the Humfrey and the Olsen were found more sensitive tests than the ordinary tensile test. Roughly, for a given iron, the deterioration produced by galvanizing and measured by these three methods is as follows:

	Per Cent.
Tensile, ultimate strength	6–8
" elongation	15–25
Humfrey static	50–60
Olsen impact	60–70

Preparation of Specimens

Previous work had shown that attempting to vary the composition of malleable by ladle additions to the regular furnace iron was both inconvenient and uncertain in results, so it was decided to make up melts on a laboratory scale, using as a base a mixture of American

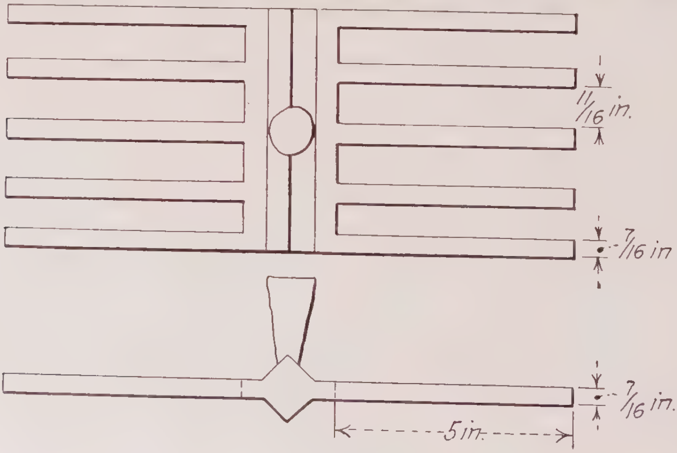

Fig. 2.

washed metal and steel scrap to which were added the necessary amounts of ferrosilicon, ferromanganese, ferrophosphorus and iron sulfide to bring the composition to the desired analysis. The melts were of approximately 18 lb. each; the melting was accomplished in graphite crucibles in a No. 3 gas-fired melter's furnace. Standard conditions were worked out and, with a little experience, iron of very uniform temperature could be turned out day after day, in a total melting time of 2¼ hours.

The Humfrey specimens were cast, ten specimens to a gate, as indicated in Fig. 2. As cast, the specimens were 7/16 in. square in cross-section and 5 in. long. Three such gates could be cast from one heat. Tensile bars and wedges were cast on some of the heats, but as this was not done on all heats, these results have been given no weight in this paper. The specimens were annealed in a regular annealing oven, under conditions of standard malleable practice. The pots containing the specimens were located approximately in the middle of the oven.

Treatment of Specimens

After annealing, the specimens of each composition were divided into three general groups, one of the groups being further subdivided, and each group or group subdivision was given a definite treatment. This gave rise to a system of treatment numbers, which is summarized in Table 1.

Table 1

Treatment No.	Treatment
1	Normal, tested as annealed.
2	Galvanized and quenched in water at room temperature.
2*a*	Galvanized and quenched in oil at room temperature.
2*b*	Galvanized and quenched in oil at 300° F.
2*c*	Galvanized and quenched in oil at 450° F.
2*d*	Galvanized and quenched in oil at 600° F.
3	Heated in furnace atmosphere to 850–900° F. and quenched in water at room temperature; no contact with zinc.

The points to be borne in mind are that treatments 1 and 2 are the principal treatments throughout the whole investigation, while treatment 3 and the lettered treatments under 2 are somewhat secondary in importance. The reasons for treatments 1 and 2 are of course obvious; the other treatments were included in an attempt to show that the galvanizing deterioration is a thermal effect due to the quenching operation rather than a direct chemical effect due to the pickling, zinc absorption, or any of the other reasons advanced at various times in the past. Treatment 3 in particular was intended to show this, for although the specimens were heated to the temperature encountered in the galvanizing process, there was no actual contact with zinc. As will be seen later, these treatments yielded data of great value in confirming our own theories and in leading us to a much clearer understanding of the fundamental causes of the deterioration.

The galvanizing treatments were carried out in a gas-fired furnace built especially for the purpose. Pyrometric control of the temperature of the zinc was maintained throughout the investigation and the bath at all times was well within the range, 850-900° F. Thermometers were used for measuring the temperatures of the various quenching baths.

After treatments the specimens were machined to size, ⅜ in. square in cross-section, notched and tested in the Humfrey static notched-bar testing machine. We wish, at this point, to acknowledge gratefully the coöperation and services of Dr. F. C. Langenberg, of the Watertown Arsenal, in arranging for and superintending the notching and testing of all the Humfrey bars in this investigation.

Experimental Data and Results

The P Series (Phosphorus)

The first series to be studied is the one in which silicon, manganese, sulfur, and total carbon were held approximately constant while the phosphorus was varied over the range 0.03 to 0.30 per cent. Table 2 gives the analyses of the heats obtained in this series. All of the heats in any one series are prefixed with characteristic letters; for example, all of the heats in this series are prefixed with the letter P and all those in the next series with SP, and so on.

Table 2.—*Series P; Chemical Composition*

Specimen No.	Silicon, Per Cent.	Manganese, Per Cent.	Sulfur, Per Cent.	Phosphorus, Per Cent.	Total Carbon, Per Cent.
P1	1.01	0.33	0.057	0.031	2.40
P2	0.91	0.29	0.036	0.051	2.39
P3	0.89	0.29	0.041	0.105	2.48
P4	0.81	0.31	0.043	0.123	2.45
P5	0.89	0.32	0.045	0.154	2.64
P6	0.96	0.23	0.053	0.174	2.44
P7	0.83	0.26	0.038	0.183	2.55
P8	0.88	0.26	0.055	0.228	2.55
P9	0.82	0.23	0.047	0.265	2.49
P10	0.86	0.23	0.075	0.297	2.60
Maximum	1.01	0.33	0.075		2.64
Minimum	0.81	0.23	0.036		2.39
Average	0.89	0.28	0.049		2.50

The fractures of the specimens after being broken in the Humphrey machine are shown in Fig. 3. The top row, treatment 1, contains the normal untreated specimens; the middle row, treatment 2, the galvanized water-quenched specimens; and the bottom row, treatment 3, the furnace-heated water-quenched specimens. Phosphorus increases from left to right. It will be noted that the white in the fracture after galvanizing (the middle row) increases with increasing phosphorus and is not

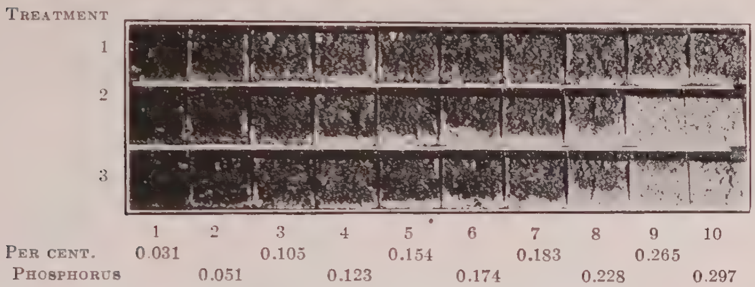

FIG. 3.—SECOND PHOSPHORUS SERIES.

TABLE 3.—*Series P; Humfrey Static Notched-bar Test*

Total Bending Angle (Ductility)

Specimen Number	Phosphorus, Per Cent.	Treatment 1, Normal as Annealed	Treatment 2, Galvanized and Water Quenched		Treatment 3, Furnace Heated and Water Quenched	
		Degrees	Degrees	Deterioration, Per Cent.	Degrees	Deterioration, Per Cent.
P1	0.031	17.2	17.2	0.0	20.1	−16.9
P2	0.051	22.9	22.9	0.0	22.9	0.0
P3	0.105	25.8	28.7	−11.2	27.5	−6.6
P4	0.123	25.8	22.9	11.2	22.9	11.2
P5	0.154	26.9	25.8	4.1	25.8	4.1
P6	0.174	25.8	22.9	11.2	22.9	11.2
P7	0.183	26.4	25.8	2.3	27.5	−4.2
P8	0.228	26.9	14.9	44.6	17.2	36.1
P9	0.265	26.9	6.3	76.6	10.3	61.7
P10	0.297	22.9	2.9	87.3	5.2	77.3

Integrator Readings (Strength)

		Treatment 1, Normal	Treatment 2, Galvanized and Water Quenched		Treatment 3, Furnace Heated and Water Quenche	
		Foot-pounds	Foot-pounds	Deterioration, Per Cent.	Foot-pounds	Deterioration, Per Cent.
P1	0.031	7.2	6.0	16.7	6.5	9.7
P2	0.051	9.9	9.0	9.1	9.2	7.1
P3	0.105	12.3	12.1	1.6	13.3	−8.1
P4	0.123	10.7	9.8	8.4	10.0	6.5
P5	0.154	12.2	12.0	1.6	12.1	0.8
P6	0.174	11.4	11.5	−0.9	10.7	6.1
P7	0.183	14.1	13.4	5.0	13.9	1.4
P8	0.228	12.7	9.3	26.8	8.3	34.6
P9	0.265	13.1	4.1	68.7	6.3	51.9
P10	0.297	9.1	2.1	76.9	3.0	67.0

evident to any serious extent until the phosphorus has reached approximately 0.15 per cent. The furnace-heated water-quenched specimens (the bottom row) show an exactly similar tendency, except that it is not quite so marked as in the galvanized specimens. The point to be stressed is that all specimens to the left of No. 6 or No. 7 are essentially normal in fracture.

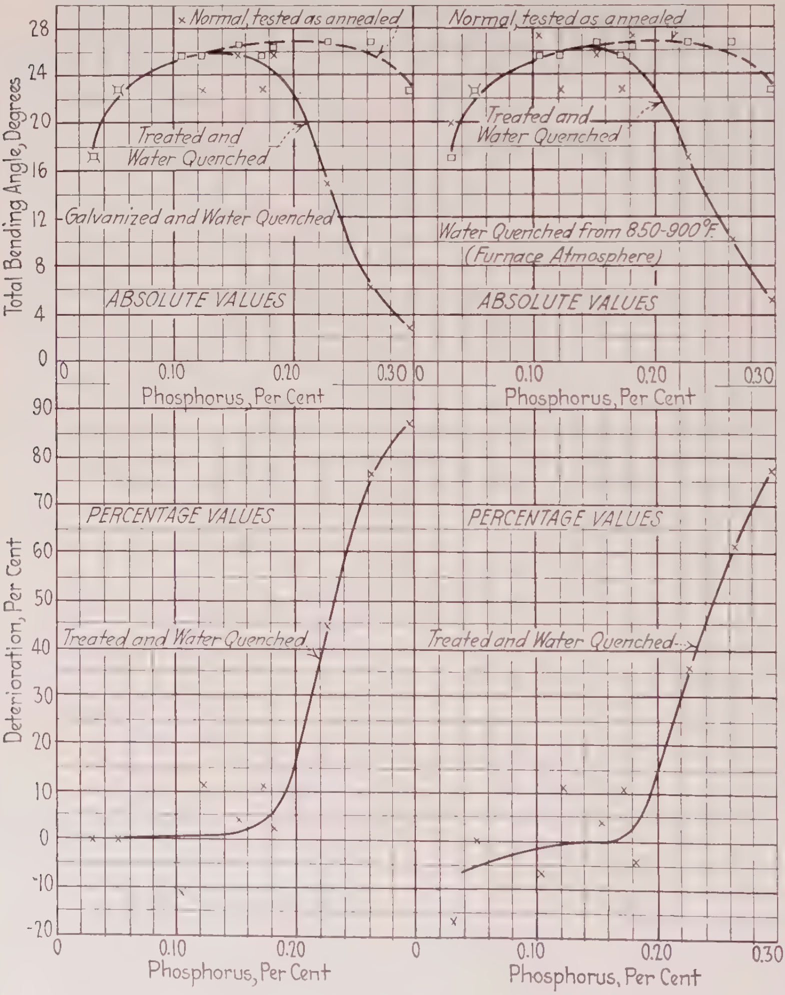

FIG. 4.—HUMFREY STATIC NOTCHED-BAR TEST, SERIES P; TOTAL BENDING ANGLE.

Table 3 contains the data on total bending angle (ductility) and integrator readings (strength) obtained in the Humfrey test. Throughout this paper, each value given in a table or plotted on a curve is the average of three tests; there was a high degree of uniformity of test values from the same specimen. Under treatment Nos. 2 and 3, the deterioration produced by the treatment is given in percentage values based on the values for the normal material (treatment No. 1) as 100 per cent. Of course a negative or minus deterioration is obviously no deterioration at all, but rather an improvement over the normal value.

Figs. 4 and 5 show the test values of Table 3 plotted against

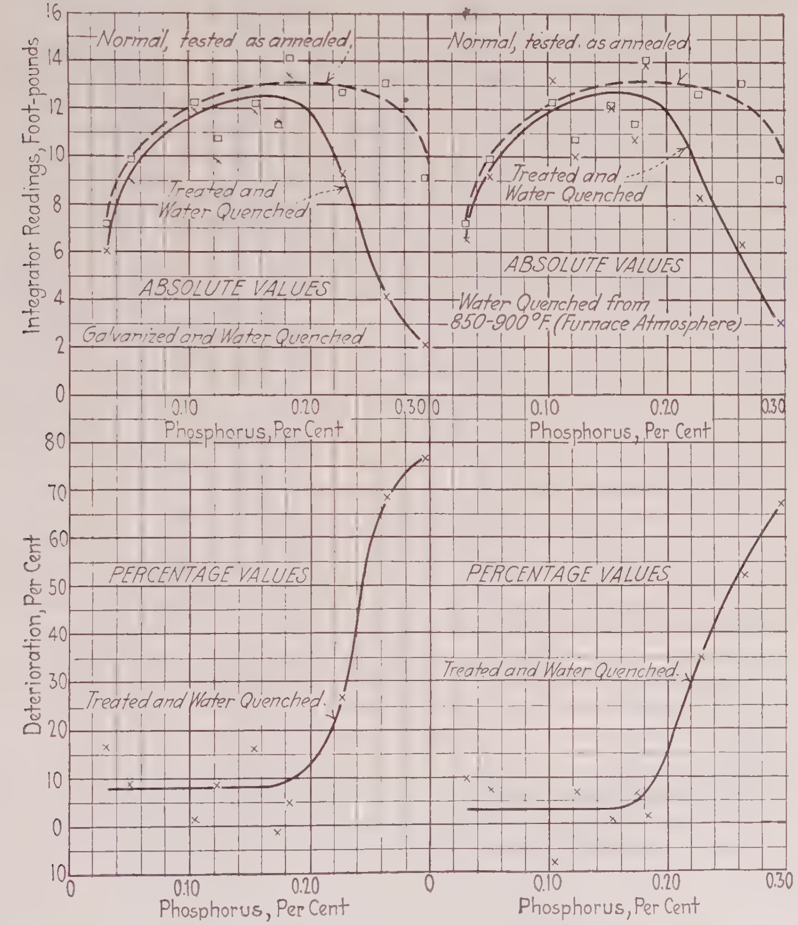

Fig. 5.—Humfrey static notched-bar test, series P; integrator data.

phosphorus content; Fig. 4 gives the ductility curves and Fig. 5 the strength curves. The dotted curve in each case plots the normal values of the material as annealed; the full line is the value after galvanizing or furnace heating and quenching. It will be seen that the two curves hold approximately together up to a certain phosphorus content, at which point the curve for the treated and quenched material falls off sharply.

It will also be noted that in this series and in the H, O, and B series (to be discussed later) the low-phosphorus specimens, 0.08 per cent. and below, were somewhat inferior in mechanical properties to those immediately above them in the phosphorus range. This may be due to the fact that shrinkage enters into the problem at these low-phosphorus contents and materially weakens the iron. In fact, large shrinkage cracks were observed in several of these very low-phosphorus specimens. Fortunately, however, it is not necessary to carry the phosphorus down to such excessively low values in order to avoid galvanizing troubles; therefore this phase of the matter has no actual significance in the practical application of the results of the investigation.

The curves on the lower halves of each figure are plotted from the percentage deterioration values; these percentage curves combine the data of the two upper curves. It must be borne in mind that such curves are a type of inverse curve and that a deterioration is actually represented by a rise. This method of plotting also tends to eliminate variations in test values which might be due to slight differences in chemical composition aside from phosphorus, shrinkage, etc. It will be seen that, just as in the case of the fractures, the curves for the galvanized and furnace-heated materials are exactly similar, the only difference being one of magnitude, and this very small.

To sum up, this series makes it evident that, all other elements being constant, above a certain value phosphorus is a vital factor in producing galvanizing brittleness. It is also evident, from the furnace-heated series, that the so-called galvanizing brittleness can be induced in the higher phosphorus specimens without any contact with zinc whatsoever.

The SP Series (Silicon and Phosphorus)

In the next series to be studied, the silicon and phosphorus were varied simultaneously. This was accomplished by taking three series, each of a constant phosphorus content, and varying the silicon over a given range in each. Manganese, sulfur, and total carbon were held as constant as possible throughout. The characteristic prefix of specimen numbers in this series is SP. Table 4 contains the analyses of the heats in this series and is divided into three sections corresponding to the three constant phosphorus contents. The unnumbered heat in the second section is P7 from the P series, the analysis of this heat permitting the use of its test values in both the P and the SP series.

The specimens after being tested in the Humfrey machine are shown in Figs. 6, 7, and 8. Each illustration corresponds to one of the constant phosphorus sections of the series, as indicated in Table 4. In each case, the silicon increases from left to right: while the average phosphorus advances from 0.104 per cent. in Fig. 6 to 0.189 per cent. in

TABLE 4.—*Series SP; Chemical Compositions*

Specimen Number	Silicon, Per Cent.	Manganese, Per Cent.	Sulfur, Per Cent.	Phosphorus, Per Cent.	Total Carbon, Per Cent.
SP1.	0.54	0.25[a]	0.051[a]	0.121[a]	2.51[a]
SP2.	0.58[a]	0.28	0.044	0.093	2.50
SP3.	0.63	0.25	0.045	0.111	2.45
SP4.	0.94	0.28	0.053	0.103	2.46
SP5.	1.09	0.25	0.049	0.101	2.53
SP6.	1.21	0.23	0.051	0.105	2.50
SP7.	1.28	0.27	0.049	0.109	2.36
Maximum.		0.28	0.053	0.111	2.53
Minimum.		0.23	0.044	0.093	2.36
Average.		0.26	0.047	0.104	2.47
SP8.	0.57	0.26	0.053	0.192	2.47
SP9.	0.66	0.23	0.048	0.204	2.54
SP10.	0.74	0.26	0.051	0.186	2.46
	0.83	0.26	0.038	0.183	2.55
SP11.	0.93	0.27	0.055	0.180	2.45
SP12.	1.08	0.29	0.050	0.184	2.49
SP13.	1.23	0.31	0.044	0.182	2.42
SP14.	1.47	0.27	0.042	0.198	2.49
Maximum.		0.31	0.055	0.204	2.55
Minimum.		0.23	0.038	0.180	2.42
Average.		0.27	0.048	0.189	2.48
SP15.	0.65	0.24	0.071	0.230	2.52
SP16.	0.84	0.33	0.052	0.239	2.44
SP17.	1.00	0.36	0.039	0.227	2.50
SP18.	1.36	0.31	0.053	0.259	2.48
Maximum.		0.36	0.071	0.259	2.52
Minimum.		0.24	0.039	0.227	2.44
Average.		0.31	0.054	0.238	2.49

[a] Figures not included in averages.

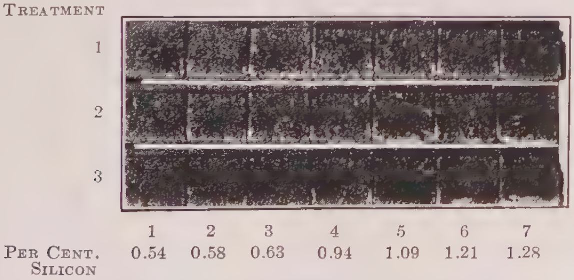

FIG. 6.—SILICON-PHOSPHORUS SERIES No. 1; AVERAGE PER CENT. PHOSPHORUS 0.104.

Fig. 7 to 0.238 per cent. in Fig. 8. It will be noted that the white in the fracture after galvanizing (treatment 2) increases as both the silicon and the phosphorus increase, but that in the lowest phosphorus section, all of the specimens showed normal black fractures over the whole silicon range. In the medium phosphorus range, only the lowest silicon irons show normal fracture after galvanizing; in the high phosphorus section,

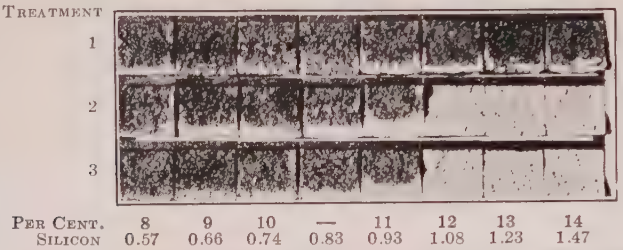

Fig. 7.—Silicon-phosphorus series No. 2; average per cent. phosphorus 0.189.

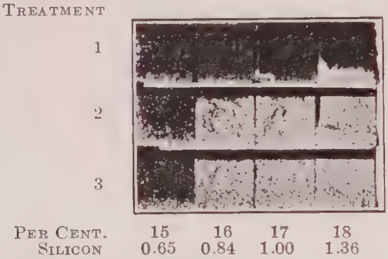

Fig. 8.—Silicon-phosphorus series No. 3; average per cent. phosphorus 0.238.

Table 5.—*Series SP; Humfrey Static Notched-bar Test*

Specimen Number	Silicon, Per Cent.	Average Per Cent. Phosphorus	Total Bending Angle (Ductility)			Integrator Readings (Strength)		
			Treatment 1 Normal as Annealed, Degrees	Treatment 2 Galvanized and Water Quenched, Degrees	Treatment 3 Furnace Heated and Water Quenched, Degrees	Treatment 1 Normal Annealed, Foot-pounds	Treatment 2 Galvanized and Quenched, Foot-pounds	Treatment 3 Furnace Heated and Quenched Water Foot-pounds
SP1	0.54	0.104	24.6	25.8	24.6	12.4	11.1	10.2
SP2	0.38		25.8	25.8	24.6	10.9	11.1	10.7
SP3	0.63		22.9	24.6	24.1	10.7	9.6	8.3
SP4	0.94		24.6	25.8	25.8	12.1	12.2	11.1
SP5	1.09		25.8	24.6	25.8	11.7	11.4	11.0
SP6	1.21		23.5	24.1	24.1	10.8	10.2	10.5
SP7	1.28		24.1	24.1	23.5	11.3	10.7	10.2
SP8	0.57	0.189	25.8	24.6	26.4	12.1	10.8	11.4
SP9	0.66		22.9	22.9	17.8	8.9	9.0	7.6
SP10	0.74		26.4	25.8	25.8	13.5	13.0	12.9
	0.83		26.4	25.8	27.5	14.1	13.4	13.9
SP11	0.93		28.7	20.6	17.2	15.1	12.3	12.2
SP12	1.08		27.5	4.0	4.6	13.9	2.5	2.8
SP13	1.23		26.4	4.6	4.0	14.1	3.3	2.6
SP14	1.47		26.4	4.6	3.4	13.7	2.5	2.2
SP15	0.65	0.238	21.2	20.2	17.8	8.3	6.9	6.8
SP16	0.84		26.4	5.7	6.3	13.8	4.5	4.1
SP17	1.00		25.8	5.7	4.6	12.7	4.2	3.6
SP18	1.36		14.3	2.9	2.3	9.2	1.7	1.5

practically all are adversely affected. An almost exact parallelism is evident in the furnace-heated and quenched specimens (treatment 3), bearing out the results of the P series (Fig. 3) in a remarkable manner.

Table 5 contains the data on total bending angle (ductility) and integrator readings (strength) for the specimens of this series as tested in the Humfrey machine.

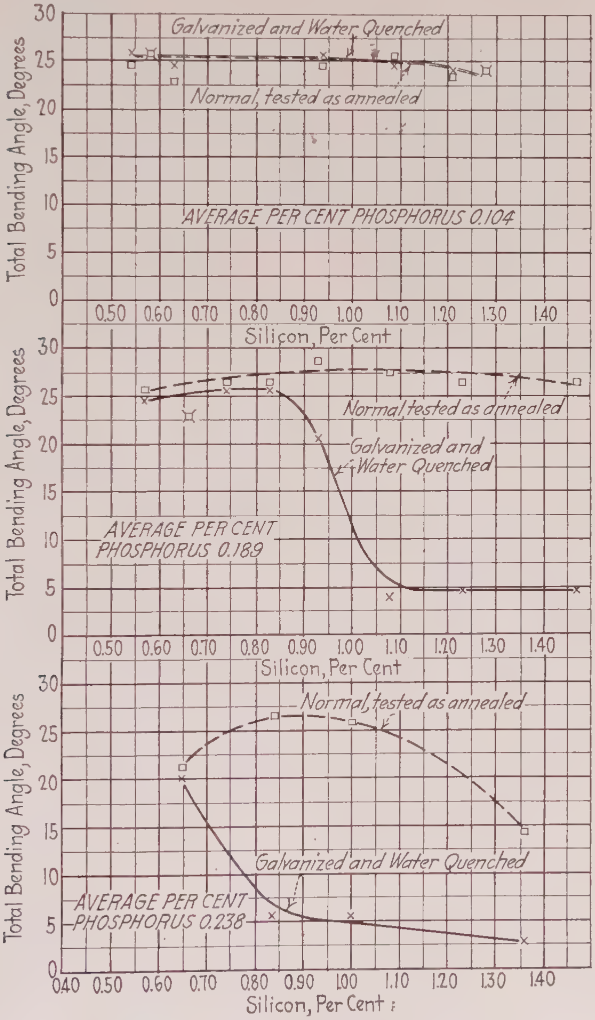

FIG. 9.—HUMFREY STATIC NOTCHED-BAR TEST, SERIES SP; TOTAL BENDING ANGLE.

The data in this table are given in curve form in Figs. 9, 10, and 11, Fig. 9 being the ductility curves and Fig. 10 the strength curves for the galvanized material, while Fig. 11 contains both the ductility and the strength curves for the furnace-heated and quenched material. The dotted curves are the normal values for the heats, tested as annealed (treatment 1). The full-line curves are the values for the treated material. The test data are plotted against the silicon content and a separate set of curves is given for each constant average phosphorus content, *i.e.*, for each section of Tables 4 and 5. The phosphorus content

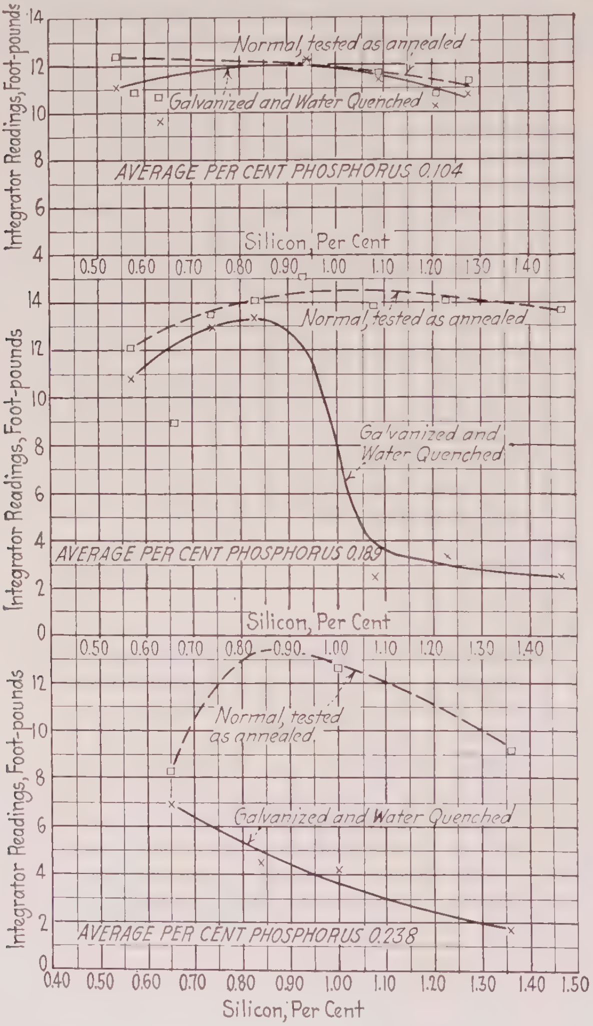

FIG. 10.—HUMFREY STATIC NOTCHED-BAR TEST, SERIES SP; INTEGRATOR DATA.

for each set of curves is marked plainly in the field. The data of this series could, of course, be properly plotted on a three-dimensional diagram, but the difficulty of interpreting such diagrams makes it inadvisable to include one in this paper.

The curves at the top of the figures show that there is little, and in some cases no, deterioration throughout the whole silicon range in this low-phosphorus material (0.104 per cent.). The integrator data would seem to be more sensitive in this respect than the total bending angle, although there is little difference in most cases. Points that are extremely low and off the curves, particularly the second and third points on some of the curves, are known to be due to unsoundness (shrinkage) in these specimens. For the next higher value of phosphorus (0.189 per cent.) the curves indicate that galvanizing has no deleterious effect until approximately 0.80 per cent. silicon is reached, above which value the curves for the galvanized material drop off sharply. For the

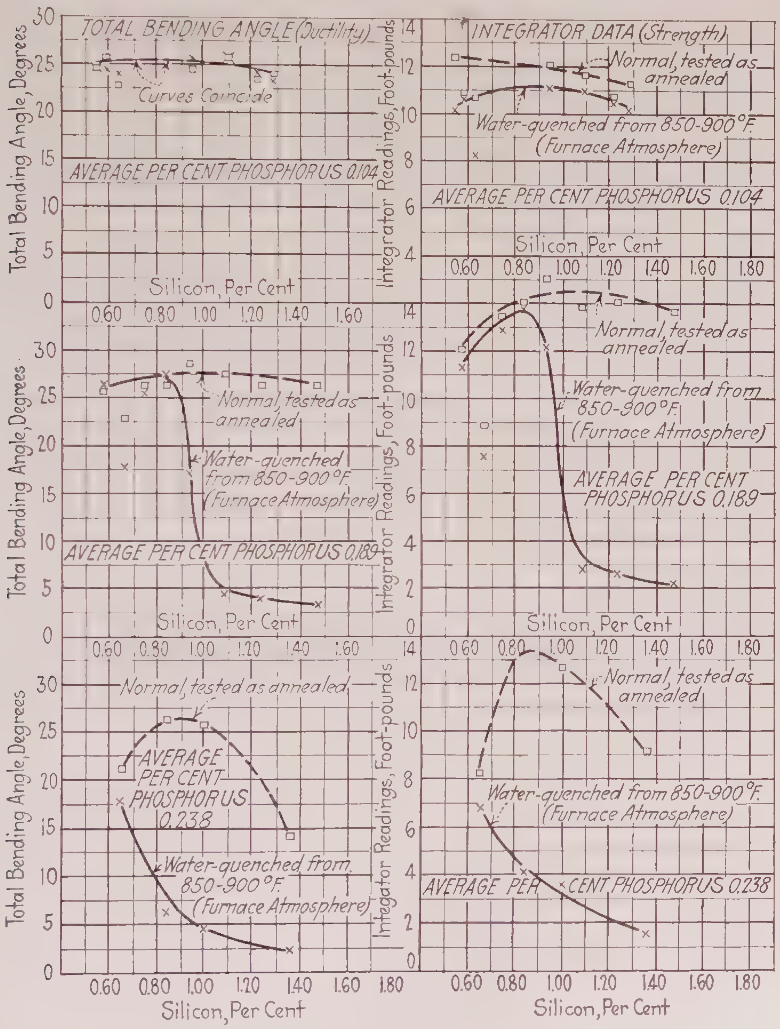

FIG. 11.—HUMFREY STATIC NOTCHED-BAR TEST, SERIES SP.

highest value of phosphorus (0.238 per cent.), the curves indicate that only the lowest silicon heat withstood the galvanizing process; and even this heat showed some deterioration. A comparison of the curves of Fig. 11 with those of Figs. 9 and 10 shows that the furnace-heated and quenched material acted in a manner exactly similar to the galvanized and quenched material and that, in turn, the test data for both of these treatments bear out the indications of the fractures. To sum up, then, this series makes it clear that besides the phosphorus, the silicon also plays an important part in producing the deterioration in galvanized malleable. The results on the furnace-heated specimens bear out those of series P, indicating that the deterioration is a thermal phenomenon.

The Working Diagram

The results of both the P and the SP series are incorporated in a single working curve given in Fig. 16. This curve plots the critical combination of phosphorus and silicon above which galvanizing deterioration may be expected and below which galvanizing can be accomplished with impunity. Reference will be made to the curve in the conclusions at the end of this paper; suffice it to say that the foundryman should not attempt to work too closely to this curve but should try to keep well to the low-phosphorus, low-silicon side.

The H Series (Humfrey Test)

The next series to be studied is similar to the first one discussed, that is, all elements except phosphorus were held constant while that element was varied over a given range. Table 6 gives the analyses of the heats in this series.

TABLE 6.—*Series H, O, and B; Chemical Compositions*

Specimen Number	Silicon, Per Cent.	Manganese, Per Cent.	Sulfur, Per Cent.	Phosphorus, Per Cent.	Total Carbon, Per Cent.
H, O, B 10	0.92	0.25	0.063	0.039	2.51
H, O, B 20	0.92	0.21	0.052	0.071	2.55
H, O, B 30	0.85	0.22	0.049	0.114	2.48
H, O, B 43	0.86	0.24	0.051	0.168	2.52
H, O, B 44	0.96	0.23	0.053	0.174	2.44
H, O, B 53	0.88	0.26	0.055	0.228	2.55
H, O, B 63	0.82	0.23	0.047	0.265	2.49
H, O, B 62	0.91	0.24	0.053	0.291	2.51
H, O, B 80	0.85	0.23	0.065	0.372	2.50
Maximum	0.96	0.26	0.065		2.55
Minimum	0.82	0.21	0.047		2.44
Average	0.89	0.23	0.054		2.51

As specimens from this series were used for three different studies, the series has no single characteristic prefix letter. The series was really divided into three parts: the H series, which was treated and tested in the usual manner in the Humfrey machine; the O series, which was treated and tested in the Olsen single-blow impact (Izod) machine; and the B series, which was used for the study of hot quenching baths, the specimens being tested in the Humfrey machine.

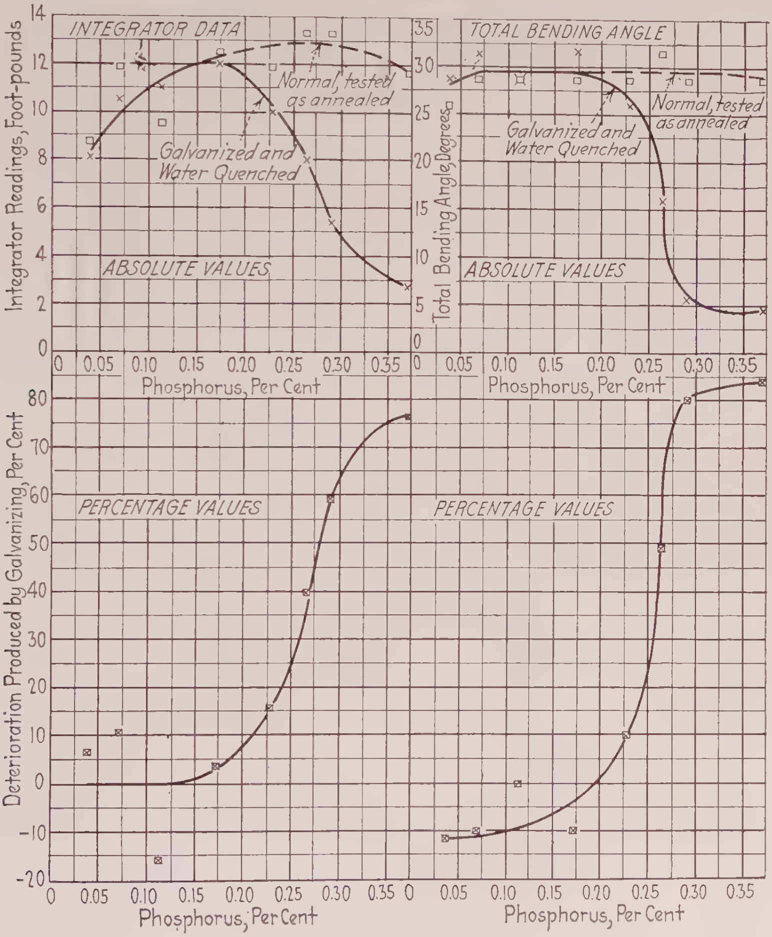

FIG. 12.—HUMFREY STATIC NOTCHED-BAR TEST, SERIES H.

Table 7 gives the data obtained in the Humfrey test on normal and galvanized specimens of this series. Under treatment 2, the deterioration produced by the galvanizing is given in percentage values based on the test values of the normal iron as 100 per cent.

In Fig. 12, test values have been plotted against the phosphorus content, the ductility curves being on the right, and the strength curves on the left. The dotted-line curves show the values for the normal material as annealed; the full-line, the values for the galvanized material. As in previous curves of this type, the test values for the normal and galvanized materials are practically identical up to a certain value of phosphorus, above which galvanized material deteriorates rapidly with increasing phosphorus. The lower curves on Fig. 12 give the same data in the inverse, or percentage, form. All these curves are exactly similar to those shown in Figs. 4 and 5; this furnishes an excellent

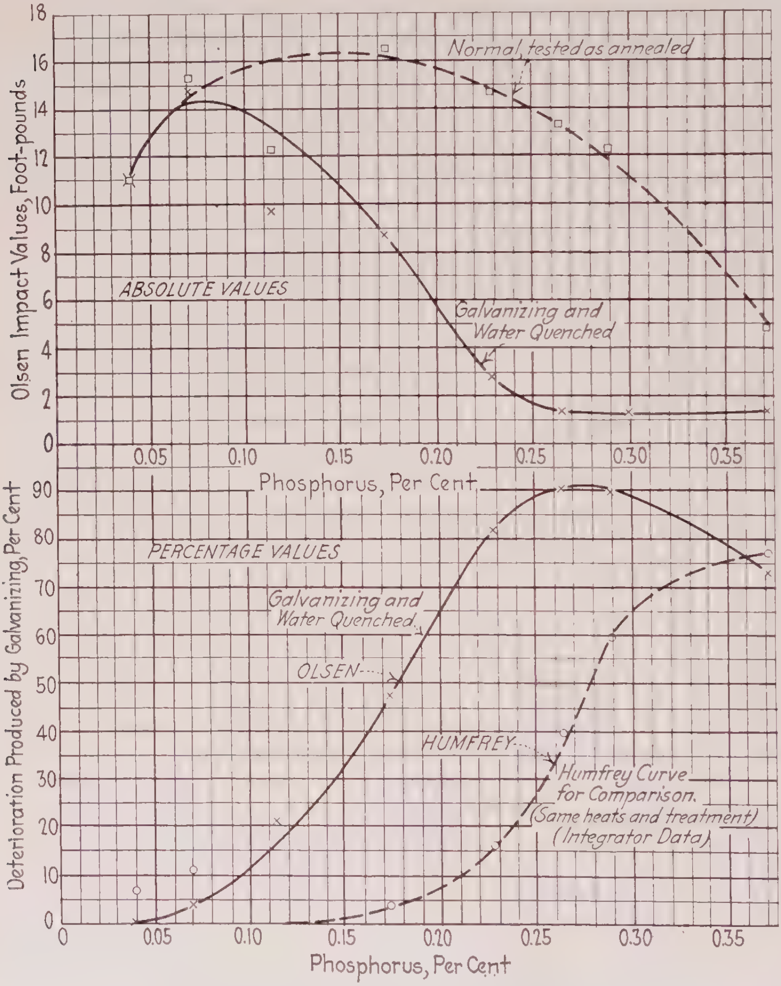

Fig. 13.—Olsen impact test, series O.

TABLE 7.—*Series H; Humfrey Static Notched-bar Test*

Specimen Number	Phosphorus, Per Cent.	Total Bending Angle			Integrator Readings		
		Treatment 1, Normal as Annealed, Degrees	Treatment 2		Treatment 1, Normal as Annealed, Foot-pounds	Treatment 2	
			Galvanized and Water-quenched, Degrees	Per Cent. Deterioration		Galvanized and Water-quenched, Foot-pounds	Per Cent. Deterioration
H 10	0.039	25.8	28.7	−11.2	8.7	8.1	6.9
H 20	0.071	28.7	31.5	− 9.8	11.8	10.5	11.0
H 30	0.114	28.7	28.7	0.0	9.5	11.0	−15.8
H 44	0.174	28.7	31.5	− 9.8	12.5	12.0	4.0
H 53	0.228	28.7	25.8	10.1	11.9	10.0	16.0
H 63	0.265	31.5	16.0	49.2	13.3	8.0	39.8
H 62	0.291	28.7	5.7	80.1	13.3	5.4	59.4
H 80	0.372	28.7	4.6	84.0	11.7	2.7	76.9

check on the work, as the data for the curves of Figs. 4 and 5 were obtained from an entirely different set of heats from those of Fig. 12 (see Tables 2 and 6).

The O Series (Olsen Test)

Table 8 gives the data obtained when specimens of these same heats were tested in the Olsen impact machine. In Fig. 13 are the test values plotted against the phosphorus content. The dotted line gives the values for the normal ungalvanized material; the full line gives the values for the galvanized specimens. The curves would seem to indicate that a slight embrittlement is caused by galvanizing, as soon as the phosphorus has exceeded 0.07 per cent. and this brittleness increases

TABLE 8.—*Series O; Olsen Single-blow Impact Test (Izod)*

Specimen Number	Phosphorus, Per Cent.	Treatment 1, Normal as Annealed, Foot-pounds	Treatment 2	
			Galvanized and Water-quenched, Foot-pounds	Per Cent. Deterioration
10	0.039	11.0	11.0	0.0
20	0.071	15.3	14.7	3.9
30	0.114	12.3	9.7	21.1
44	0.174	16.5	8.7	47.3
53	0.228	14.7	2.7	81.6
63	0.265	13.3	1.3	90.2
62	0.291	12.3	1.3	89.4
80	0.372	4.8	1.3	72.9

rapidly with higher phosphorus contents. The lower curve gives the data in percentage form, the dot-and-dash line being the lower left-hand curve of the H series (Fig. 12) reproduced for direct comparison between the Olsen impact and the Humfrey static tests. The indications are

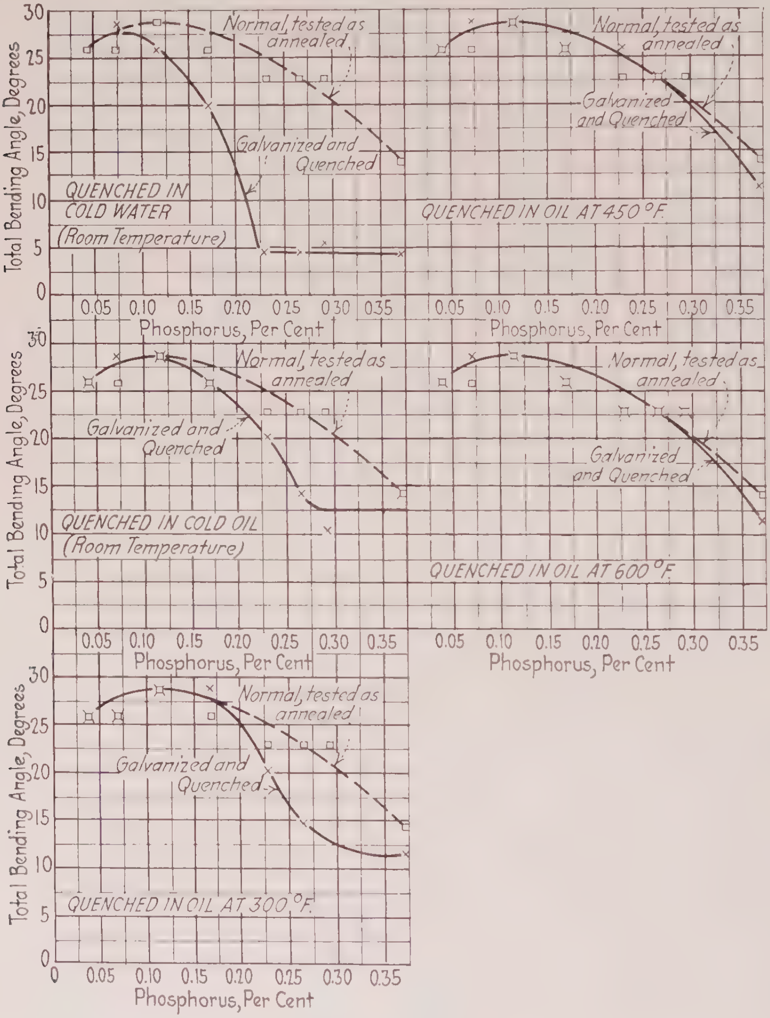

FIG. 14.—HUMFREY STATIC NOTCHED-BAR TEST, SERIES B; TOTAL BENDING ANGLE, ABSOLUTE VALUES.

that the Olsen impact is a more sensitive test; in fact, a probably supersensitive test so far as this type of deterioration is concerned. It is interesting to note that the Olsen curves have the same general shape as the corresponding Humfrey curves and hence serve as a check on the validity of the method of testing adopted.

The B Series (Oil-quenching Baths)

In this series, a number of specimens (the analyses of which are given in Table 6) were divided into six groups and treated, as in Table I under Treatments 1, 2, 2*a*, 2*b*, 2*c*, and 2*d*. These hot quenching-bath treatments were given in an attempt to show that the galvanizing deterioration could be eliminated by reducing the severity or suddenness of the quench; *i.e.*, by bringing the temperature of the quenching bath nearer to that of the zinc bath. The quenching media used in these experiments were lubricating oils of high flash point; the specimens after being quenched were allowed to cool in the air.

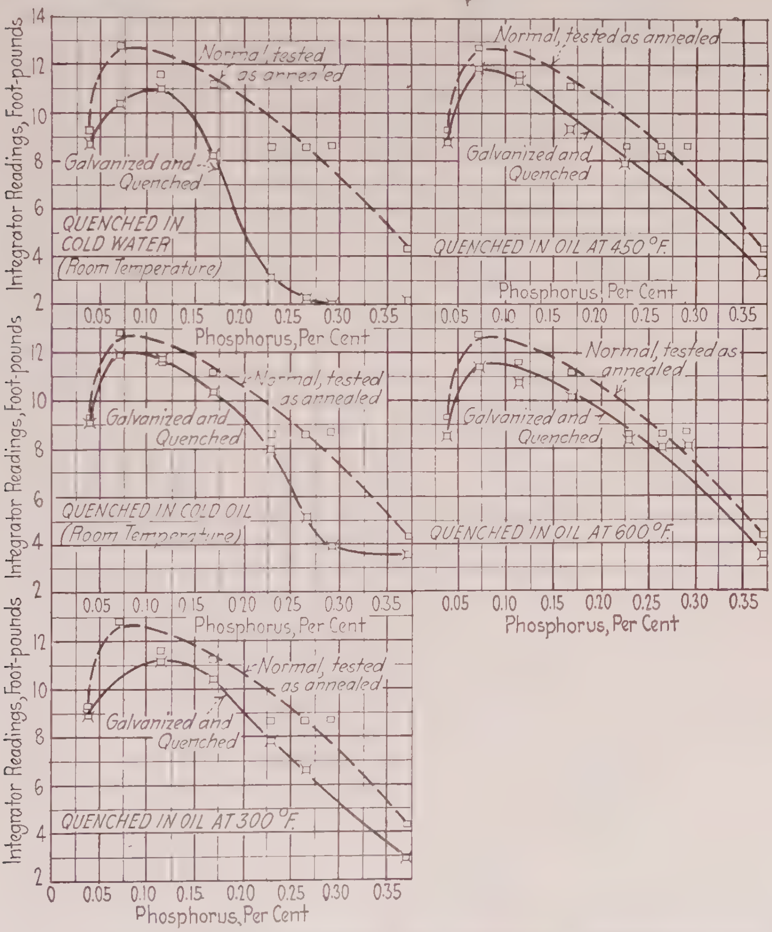

FIG. 15.—HUMFREY STATIC NOTCHED-BAR TEST, SERIES B; INTEGRATOR DATA, ABSOLUTE VALUES.

Table 9 gives the data obtained when these specimens were tested in the Humfrey machine. Fig. 14 gives the ductility data and Fig. 15 gives the strength data in curve form. The dotted curve in each case plots the normal untreated test values versus phosphorus content, while the full-line curve plots test values after galvanizing and quenching in one of the several media. The quenching medium and its temperature are plainly marked in the space occupied by each pair of curves. It will be noted that the curves are of the same general shape as all the others of this type in the report.

TABLE 9.—*Series B; Humfrey Static Notched-bar Test*

Total Bending Angle (Ductility)

Specimen Number	Phosphorus Per Cent.	Treatment 1, Normal as Annealed Degrees	Treatment 2, Galvanized and Water Quenched, Degrees	Treatment 2*a*, Galvanized and Quenched in Cold Oil, Degrees	Treatment 2*b*, Galvanized and Quenched in Oil at 300° F. Degrees	Treatment 2*c*, Galvanized and Quenched in Oil at 450° F. Degrees	Treatment 2*d*, Galvanized and Quenched in Oil at 600° F. Degrees
B 10	0.039	25.8	25.8	25.8	25.8	25.8	25.8
B 20	0.071	25.8	28.7	28.7		28.7	28.7
B 30	0.114	28.7	25.8	28.7	28.7	28.7	28.7
B 43	0.168	25.8	20.1	25.8	28.7	25.8	25.8
B 53	0.228	22.9	4.6	20.1	20.1	25.8	22.9
B 63	0.265	22.9	4.6	14.3	14.9	22.9	22.9
B 62	0.291	22.9	5.7	10.3			22.9
B 80	0.372	14.3	4.6	14.3	11.5	11.5	11.5

Integrator Readings (Strength)

		Treatment 1, Normal as Annealed, Foot-pounds	Treatment 2, Galvanized and Water Quenched, Foot-pounds	Treatment 2*a*, Galvanized and Quenched in Cold Oil, Foot-pounds	Treatment 2*b*, Galvanized and Quenched in Oil at 300° F., Foot-pounds	Treatment 2*c*, Galvanized and Quenched in Oil at 450° F., Foot-pounds	Treatment 2*d*, Galvanized and Quenched in Oil at 600° F., Foot-pounds
B 10	0.039	9.3	8.7	9.1	8.9	8.8	8.5
B 20	0.071	12.8	10.4	11.9		11.9	11.4
B 30	0.114	11.6	11.0	11.7	11.1	11.3	10.7
B 43	0.168	11.2	8.2	10.4	10.4	9.4	10.1
B 53	0.228	8.6	3.2	8.0	7.8	7.9	8.3
B 63	0.265	8.6	2.3	5.2	6.6	8.2	8.0
B 62	0.291	8.7	2.0	3.9			8.1
B 80	0.372	4.3	2.2	3.6	2.9	3.3	3.5

The galvanized and quenched material is practically as good as the normal material, tested as annealed, for the low values of phosphorus. As the temperature of the quenching bath is increased, the curve for the galvanized and quenched material approaches the normal curve over a greater portion of its length until, with the 600° F. quenching bath, the two curves are very close in the case of strength and practically coincident in the case of ductility, indicating that if the quenching could be entirely eliminated there would probably be no embrittlement in galvanizing. It is interesting to note the difference between the curves

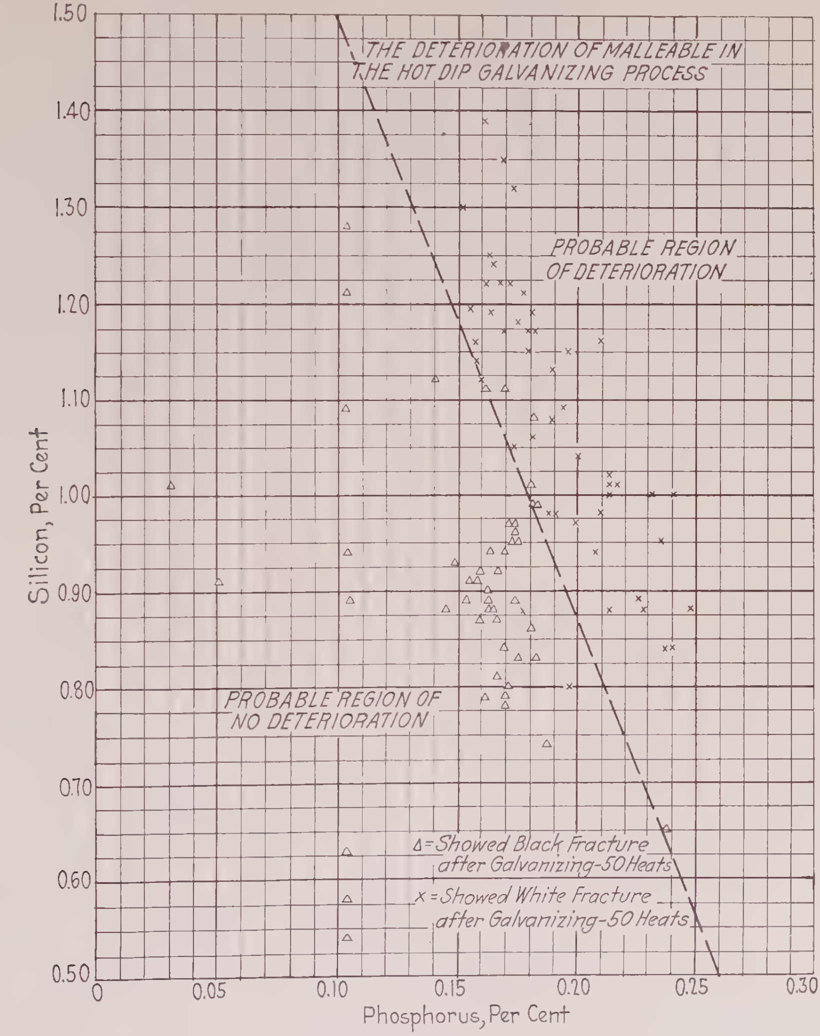

FIG. 16.

for the material galvanized and quenched in water, and that galvanized and quenched in cold oil. Both the quenching baths were at ordinary room temperature, yet there is an appreciably smaller deterioration in the oil-quenched specimens. The high-temperature quenching baths caused considerable damage to the zinc coating; for the practical application of this phase of the research, more suitable quenching baths would have to be devised and it is problematical if high quality zinc finish could be obtained, even with these, at the higher temperatures.

Practical Application of Results of Research

The results of this investigation have been tried out on a production scale and found to be in entire agreement with the indications of the laboratory work. The problem has been attacked by lowering the phosphorus in regular heats at a number of plants until the irons were well within the lower field of the working diagram, Fig. 16. Samples of iron from these heats, when given a commercial galvanizing treatment, have maintained all the properties of normal untreated material, *i.e.*, excellent bending properties and normal black fracture. A close study of the fluidity of the molten iron, as measured by the percentage of good castings, during the period in which the phosphorus was being lowered in the heats has shown that this property is not adversely affected, at least to any serious extent. It has also been evident that the ordinary tensile properties of the product have not been adversely affected by the lowered phosphorus: in a word, this lower phosphorus malleable is as good as, if not better than, the old higher phosphorus material and, in addition, can be sold as proof against galvanizing brittleness.

Path of Rupture in Normal Malleable and Brittle Galvanized Malleable

Data from a previous investigation on the purely metallographic side of this problem are included in this report. A study of the path of rupture in normal malleable and malleable that had been embrittled by the galvanizing process was made by inducing partial fracture in both types of iron and then examining the incipient fracture under the microscope. Figs. 17 and 18 show the path of rupture in the normal ungalvanized material at low and high magnification respectively. The fracture is transgranular, *i.e.*, across the ferrite grains, and is entirely in accord with the behavior of normal metals at ordinary temperatures.[6] It is interesting to note how the fracture travels from one temper carbon spot to another. Figs. 19 and 20 show the path of rupture in brittle, galvanized material at low and high magnification respectively. The fracture is intergranular, *i.e.*, around ferrite grains, or in the grain boundaries rather than across the grains, a condition always found to be associated with metals broken at high temperatures, or with abnormal material broken at ordinary temperatures.

References in the literature bearing on the plastic deformation of malleable furnish additional evidence on this phase of the work. A

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[6] H. S. Rawdon, *Chem. & Met. Eng.* (1920) **22,** 508.

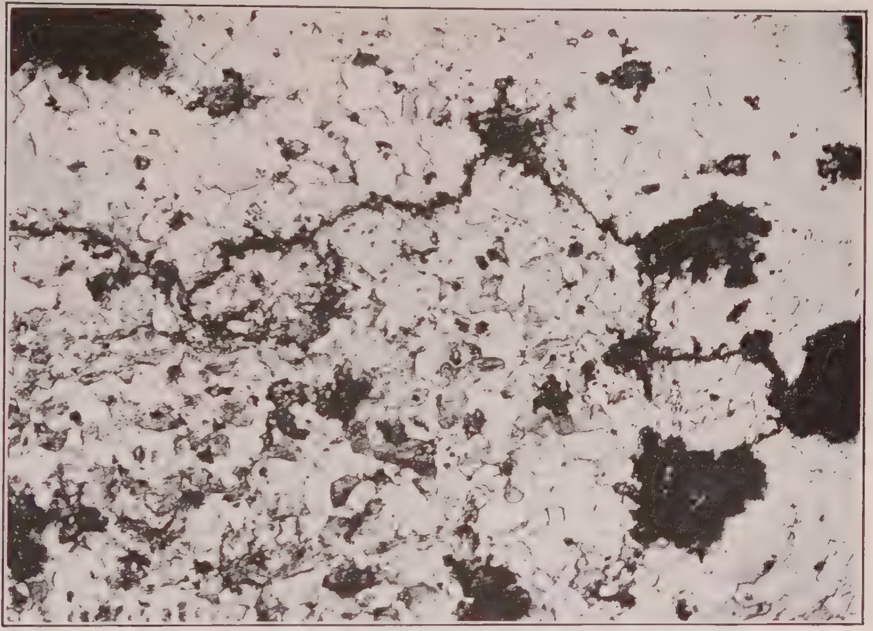

FIG. 17.—TRANSCRYSTALLINE RUPTURE IN NORMAL MALLEABLE. 100 ×.

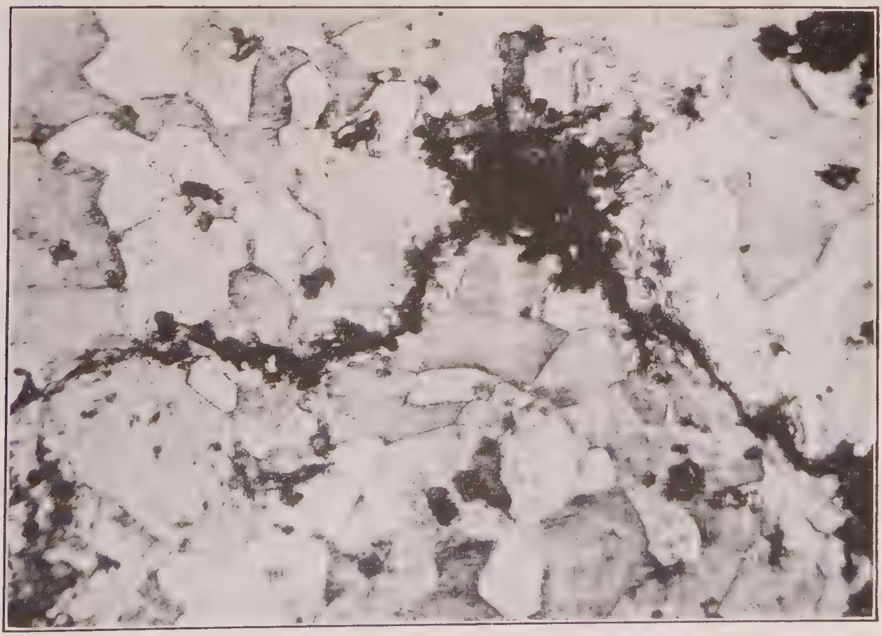

FIG. 18.—TRANSCRYSTALLINE RUPTURE IN NORMAL MALLEABLE. 300 ×.

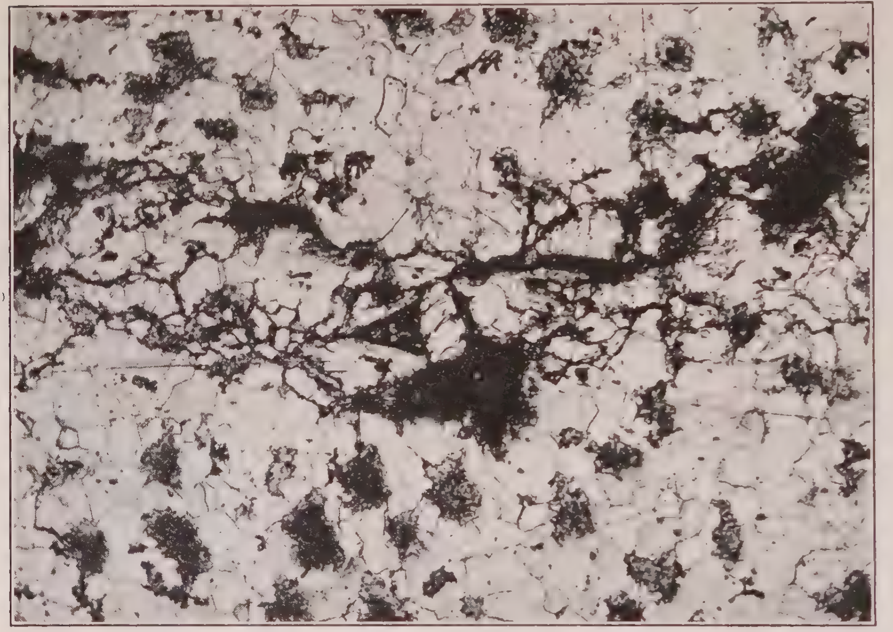

FIG. 19.—INTERCRYSTALLINE RUPTURE IN GALVANIZED MALLEABLE. 100 ×.

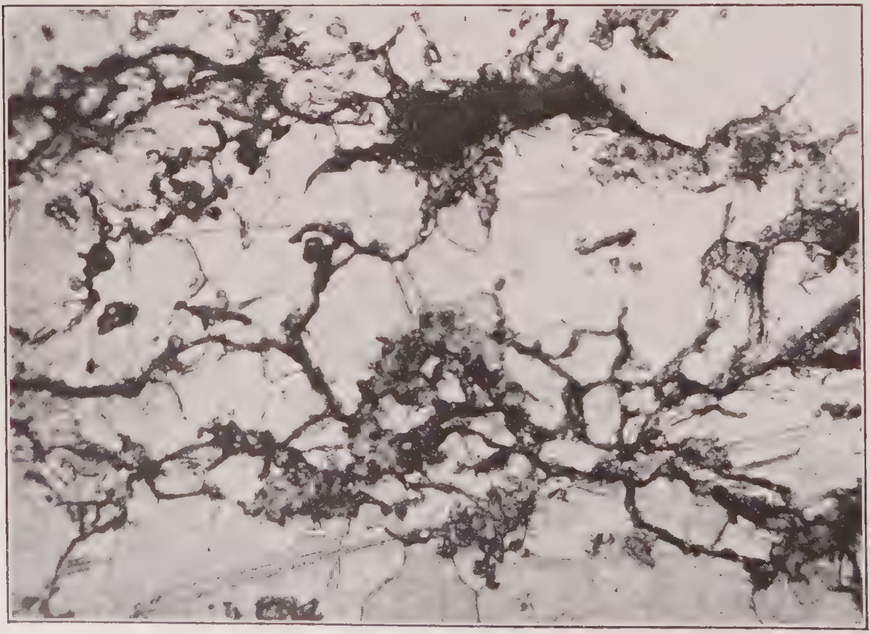

FIG. 20.—INTERCRYSTALLINE RUPTURE IN GALVANIZED MALLEABLE. 300 ×.

study[7] of the structures on the compression and tension sides of a bar of malleable distorted by transverse bending yielded evidence that on the tension side, deformation takes place in a transgranular manner, as in Figs. 17 and 18, while on the compression side deformation takes place in an intergranular manner, as in Figs. 19 and 20. As is well known, the compression side of bent malleable is frequently white in fracture. We are thus in possession of independent data confirming our conclusion that malleable which is brittle and white in fracture after galvanizing ruptures in an intergranular manner. Further work along these lines is in progress, in which we will attempt to determine just what is at the basis of this intergranular brittleness and how it can be correlated with the phosphorus, silicon and quenching factors.

General Summary and Conclusions

1. It has been shown that the deterioration of malleable in the hot-dip galvanizing process is intimately connected with the phosphorus and silicon contents of the iron. In general, low-phosphorus, low-silicon irons will withstand the process best and show practically no deterioration. With phosphorus contents of about 0.10 per cent., the silicon may go as high as 1.20 per cent. without serious trouble from galvanizing. High-phosphorus, high-silicon irons are practically certain to deteriorate and be embrittled by galvanizing.

2. A working curve has been given (Fig. 16), which differentiates between irons liable and not liable to deterioration, so far as silicon and phosphorus are concerned. This curve should serve as a ready guide to the foundryman in determining whether or not a given iron is fit for galvanizing. It should be borne in mind, however, that the curve is, at best, only an approximation, and that to be absolutely safe, the foundryman should keep to the left of it by at least 0.02 per cent. to 0.03 per cent. phosphorus. As time goes on, figures from actual practice will fix the position of this curve more definitely.

3. It has been shown that in those irons that are embrittled, the deterioration by the galvanizing is due to a thermal effect resulting from the quenching after the galvanizing operation. Below a certain combination of phosphorus and silicon contents, this effect is not evident or at least is negligible; above it, the effect becomes very marked (see Fig. 16). The exact mechanism of the deterioration, *i.e.*, whether it is a species of internal strain, or a segregation of the silicon and phosphorus in some way at the grain boundaries is not as yet known.

4. It is evident that galvanizing embrittlement may be avoided by manipulation of the phosphorus and silicon, by the use of high-tempera-

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[7] H. A. Schwartz: American Malleable Cast Iron. *Iron Trade Rev.* (1921) **68**, 902.

ture quenching baths (300° to 600° F.) after galvanizing, or by eliminating quenching entirely. The first is the most practical method and the easiest of accomplishment, as it can be carried out in the foundry and does not involve a modification of the galvanizing process itself.

5. Micrographic evidence has been submitted tending to demonstrate that fracture in the defective galvanized material is intergranular, an abnormal condition, while fracture in ungalvanized material is transgranular, a condition normal to nearly all metals at room temperatures. In other words, the brittleness of such malleables as deteriorate in galvanizing is due to intergranular weakness.

6. A test that involves a slow bending stress is an excellent measure of the quality of malleable and a sensitive detector of embrittlement or other form of deterioration. The single-blow cantilever impact test is sensitive to such deterioration in malleable but does not furnish as much information as the static bending test. Both tests are superior to the ordinary tensile test as a measure of the quality of malleable.

7. The means for eliminating galvanizing brittleness developed in this investigation have been found to be practicable from a production standpoint.

DISCUSSION

L. H. MARSHALL,* Mansfield, Ohio (written discussion).—This paper sheds a great deal of light on a defect that caused considerable worry to manufacturers of malleable iron; in fact, galvanized malleable iron was beginning to be regarded as unreliable because of its brittleness.

The writer has for some time been investigating this same problem. While our experiments did not lead to an extended study of the influence of composition, there are other points on which the author's results can be corroborated. The Izod impact test was used for this work and about 600 specimens were broken; this test has been found to give values that can be duplicated within quite narrow limits. Normal malleable iron showed an impact resistance of 5 to 8 ft.-lb., as measured by the Izod test, while the brittle metal broke at less than 1 ft.-lb. The distinction between good and brittle material, therefore, was usually quite marked.

The direct cause of the embrittlement appears to be the quenching from the galvanizing temperature, thus the metal may be heated in a zinc, lead, salt, oil, or air bath, yet, if the quenching temperature is the same in each case, the embrittling effect will be almost identical. In agreement with the author's results, it was found that, in general, the slower the rate of cooling from the galvanizing temperature the less the embrittlement. Specimens tested six months after galvanizing were as brittle as those, from the same metal, tested the day they were treated.

Samples from forty different heats of malleable iron from two foundries were analyzed and tested in impact before and after galvanizing;

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* Director of Tests, Ohio Brass Co.

Fig. 21, in which the phosphorus is plotted against the silicon content, shows the results. An arbitrary distinction was made between the good and poor metal, thus the iron which, after galvanizing, retained 50 per cent., or more, of its original resilience was considered good metal. The graph shows no relationship between the quality of the metal and the phosphorus and silicon content. The range of composition represented is not as wide as that studied by the author. The data do not contradict his conclusions, but indicate, rather, that the composition does not entirely define the results; some other factor also influences the embrittlement. In this connection, the thermal history of the metal is probably of importance.

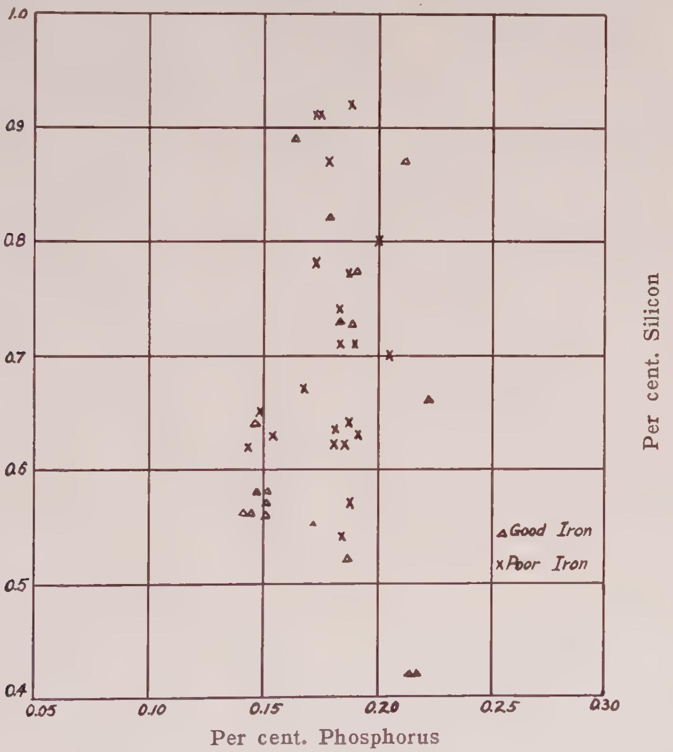

FIG. 21.—DETERIORATION IN GALVANIZING.

M. M. MARKUS, HILLSGRAVE, R. I.—Has the author, in his work on phosphorus and silicon limits on galvanized metal, found some method by which he can ascertain whether the castings have been spoiled in the galvanizer or whether they were brittle before they went to the galvanizer?

W. R. BEAN.—Out of hundreds, probably thousands, of specimens examined, those that showed abnormal fracture after galvanizing were known to be normal before galvanizing for in many cases the casting was broken before and again after it was galvanized; in other cases, specimens cast in the same mold were broken ungalvanized and galvanized. In fact, each test recorded in the paper is a duplicate of a specimen broken in the normal state.

M. M. Markus.—That answers part of the question, but have you devised some method of determining after a casting had been galvanized, whether the galvanizer had spoiled it or whether it was brittle before it went through the galvanizer?

W. R. Bean.—To this extent, we know positively that the ungalvanized material, from previous tests of the heats, was normal before it was galvanized.

H. C. Boynton, Trenton, N. J.—Has the author considered the fact that all malleable, in order to be galvanized, must be pickled; and in the pickling operation there is absorption of gases, presumably hydrogen, which produces an embrittling effect? Is it not possible that some of that brittleness referred to might have been caused by the absorption of hydrogen?

W. R. Bean.—The brittleness may be due to hydrogen to a certain extent, but to hydrogen taken up by that metal originally more than anything that was taken up from the pickling bath.

Before we started the study that forms the basis of this paper, we made an exhaustive study of the pickling and found no difference in physical properties or fractures in malleable specimens that were galvanized without being pickled and malleable specimens galvanized after pickling.

I am inclined to think that some of the inconsistencies that remain after revising the curve, as we have from studies just completed this week, may be due to absorption of oxygen, nitrogen, or hydrogen, because there are inconsistencies for which we have no explanation, although those inconsistencies are very limited. In the chart, in the probable region of no deterioration, two heats show deterioration. Moving the curve 0.04 per cent. phosphorus to the left throws those two heats well into the region of deterioration. We have endeavored to maintain a reasonable and adequate margin of safety, but in the 112 heats just tested, there are two exceptions. The reason for these exceptions, whether in sampling or in analysis, although they were carefully checked all the way through, is one of the things that perhaps is necessary to prove the rule.

H. C. Boynton.—In hot galvanizing high-carbon steel, ranging from 0.50 to 0.80 per cent. carbon, during or after the pickling operation, the pickled parts often snap or break suddenly, particularly if they are not baked quickly right after the pickling; it is customary to attribute this brittleness to the inclusion of hydrogen. One might think that the galvanizing operation itself would bake out the hydrogen, but it often does not; the pickled parts are still brittle after galvanizing; but after another baking or a lapse of time, say several months at ordinary temperatures, the metal will become normal and tough again.

W. R. Bean.—We were familiar with this experience in pickling wire and thin sheets and studied the matter thoroughly. We found absolutely no difference between the pickled and the unpickled in that respect; none of the specimens mentioned in the paper was pickled, or if so, the pickling was extremely light.

E. S. Davenport, Bloomfield, N. J.—I can corroborate what Mr. Bean has said because I was connected with that part of the work which dealt with the pickling. We made time tests to see if the effect would be any more pronounced immediately after the pickling, and were unable to find anything even there.

A. P. Ford, Bridgeport, Conn.—Do the conclusions in this paper apply to cupola as well as to the malleable furnace product referred to?

W. R. Bean.—We did not include any specimens of cupola malleable in these tests, so that I do not know. My opinion is that if you hold to the limits, assuming all other elements to be of normal composition, you would get essentially the same results, because the result is so definite in all five plants from which we have had specimens for testing, unless phosphorus is lower than it is generally run, silicon lower than it is frequently run. We have been unable to find any connection whatever between the embrittlement and manganese, sulfur, or carbon. We get high and low manganese, sulfur, and carbon in the good specimens and we get them in the bad. So that unless there is something hidden, such as the absorption of hydrogen, nitrogen, or oxygen, the cupola metal, with the same composition, should produce similar results.

Albert Sauveur, Cambridge, Mass.—Were some samples tested after pickling, but before galvanizing, and then compared with the properties of samples before pickling?

W. R. Bean.—They were.

Wm. J. Merten,* East Pittsburgh, Pa. (written discussion).—The deterioration diagram, if extended to the other ingredients of malleable iron, *i.e.*, manganese and sulfur, will probably come somewhat closer to the solution of the problem than the simple relationship of silicon and phosphorus.

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* Metallurgical Engineer, Material and Process Engineering Department, Westinghouse Elec. & Mfg. Co.

About a year ago, the Westinghouse Elec. & Mfg. Co. made a series of physical tests on thin-section castings and found that the fractures of nearly all of the galvanized malleable-iron castings were crystalline and brittle. An investigation to determine the cause convinced us that the brittleness of malleable iron caused by galvanizing is a considerably more complicated mechanism than a thermal effect resulting from quenching after galvanizing or a simple silicon-phosphorus relationship or ratio.

The chemical analysis of the iron in these castings, reported by J. L. Jones in charge of the chemical laboratory, was as follows:

	Per Cent.
Total carbon	2.00
Combined carbon	0.01
Manganese	0.28
Phosphorus	0.16
Sulfur	0.06
Silicon	0.758

This composition brings the iron well into the area of the "probable region of no deterioration;" still all castings were crystalline and brittle after galvanizing.

The following is an abstract from the report of this investigation: Tests: their purpose and results.

1. The most obvious question arising was that of quality of material both chemically and structurally as received from the foundry.

2. Is the sudden chilling after galvanizing responsible for the embrittling of good malleable iron?

3. Is it caused by incorrect or high heat of the zinc bath?

4. Is pickling the cause?

5. Is dipping in muriatic acid (HCl) harmful?

6. What effect has sal ammoniac fluxing?

The following tests were then conducted with the results indicated:

1. Chemical analysis and microscopical examination of castings as received: The analysis given above is representative and is that of a high-grade malleable iron furnished by the Eastern Malleable Iron Co. Fig. 22, by N. B. Pilling of the Research Engineering Department of the Westinghouse Elec. & Mfg. Co., shows the iron to be structurally perfect and giving a sooty fracture.

2. Check on temperature of zinc bath: The temperature of the zinc bath during the tests was approximately 800° F., the correct temperature for galvanizing.

3. Check on the purity of zinc metal used:

	Analysis
Lead	0.91
Aluminum	nil
Iron	0.04
Tin	nil
Copper	0.014
Cadmium	0.16
Zinc	98.876

This is western zinc metal of satisfactory quality for this work.

4. The M. I. castings of proper composition and of sooty fracture were then cut into sections and put through the different steps of the galvanizing process and a fracture test taken from a section of each casting after the completion of each step.

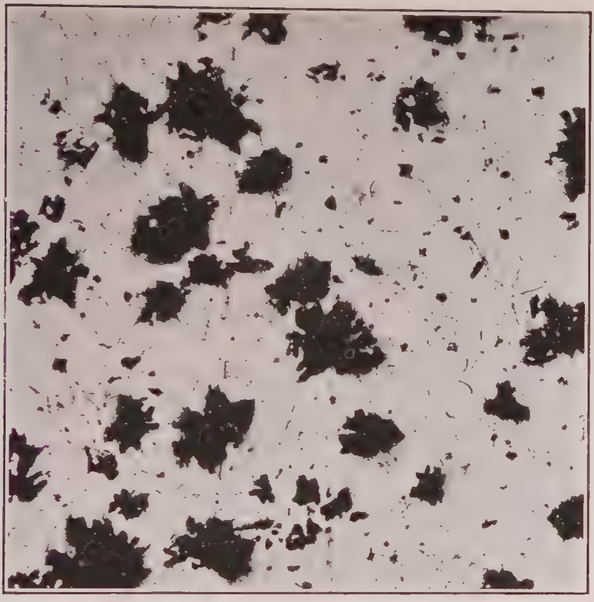

Fig. 22.—Perfectly malleableized iron. × 100.

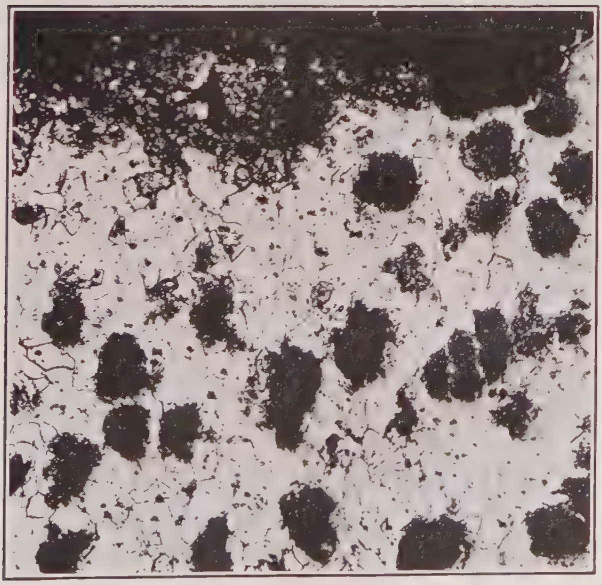

Fig. 23.—Transcrystalline break. × 100.

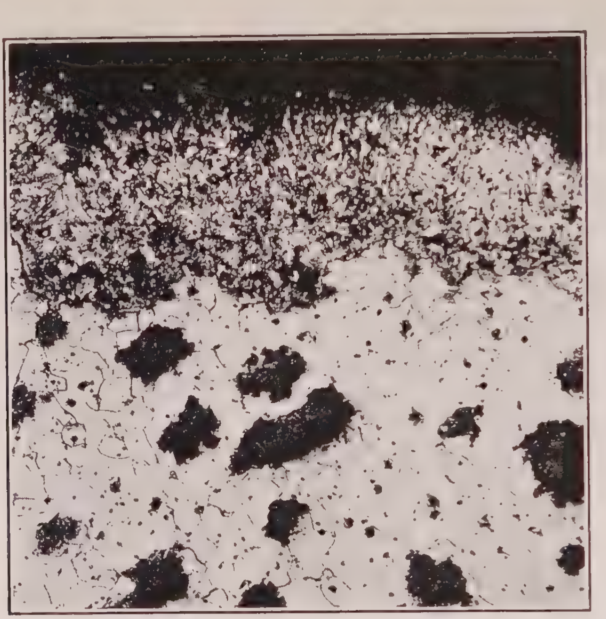

FIG. 24.—MALLEABLE CASTING SHOWING INTERCRYSTALLINE FRACTURE. × 100.

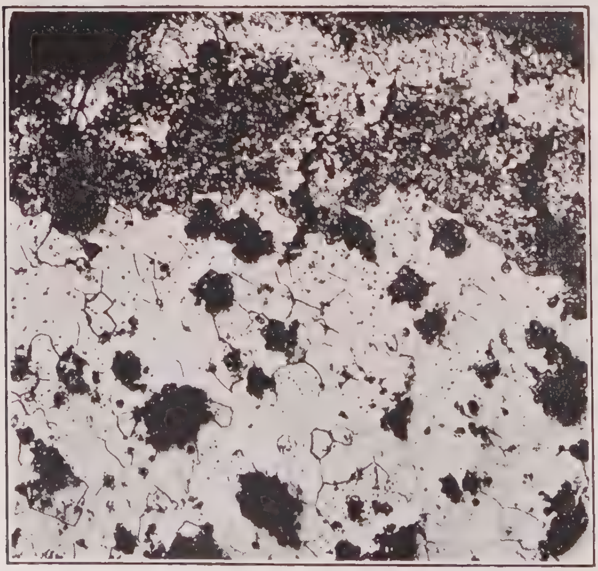

FIG. 25.—MALLEABLE CASTING SHOWING INTERCRYSTALLINE FRACTURE. × 100.

(*a*) The tests after pickling gave sooty fractures on all castings.

(*b*) Another set of tests, after immersion in muriatic acid (HCl), also gave sooty breaks on all castings.

(*c*) The third set of tests after fluxing and heating with ammonium chloride without dipping into zinc bath also gave sooty fractures.

(*d*) Some of the castings were dipped into the zinc bath and galvanized in the ordinary way and when taken out were quenched in boiling water. The fracture test on these castings gave a crystalline break and the iron was brittle.

(*e*) Another set of castings was cooled in air after galvanizing and all castings were brittle and gave a crystalline fracture.

(*f*) Extremely slow cooling in a salt bath or oil bath heated to 750° F. and 500° F., respectively, was of no avail and did not prevent the occurrence of the crystalline structure and a brittle fracture.

(*g*) Heating the casting prepared for galvanizing but not dipped into the zinc bath, the casting remained sooty and ductile after quenching from this heat.

(*h*) Heating the casting almost to the galvanizing heat prior to immersion in the zinc bath and then cooling slowly in air after galvanizing did not prevent the embrittling.

The fact that all castings showed a black, sooty fracture when heated to all temperatures up to and beyond that used for galvanizing and cooled again without coming into contact with liquid zinc, but showing crystalline fractures when galvanized irrespective of the rapidity and manner of cooling, certainly indicates that zinc is responsible for this phenomenon and further indicates that the method or rapidity of cooling after galvanizing is of little importance and has no influence upon the final results. N. B. Pilling, commenting on the fractures, checks with the results given in the paper, *i.e.*, that non-galvanized malleable iron presents a transcrystalline fracture and the brittle galvanized casting an intercrystalline fracture. Fig. 23 is a section of a non-galvanized malleable-iron casting; the part of the fracture that passes through the ferrite is transcrystalline. Figs. 24 and 25 are sections of embrittled, hot-dip, galvanized, malleable-iron castings and the part of the fracture passing through the ferrite is always intercrystalline and brittle whether quenched in cold or hot water or whether cooled quickly or very slowly.

The thermal effect resulting from quenching, so strongly emphasized in the paper, seems to be a fictitious factor in this respect and the critical values of silicon and phosphorus combinations of the diagram seem rather indefinite as shown by the embrittling of an iron well toward the left of the gradient.

In concluding, I wish to call attention to the chemical affinity of liquid zinc for the non-metallic constitutents of the iron, *i.e.*, phosphorus and sulfur present as metalloids (iron phosphide) and (iron and manganese sulfides). This affinity probably extends also to the iron silicides, which chemical affinity or possibly catalytic action of zinc can well be responsible for intergranular weakness and would result in easy slippage upon the crystal faces and consequently produce intercrystalline rupture, the fracture showing white and granular. This would suggest the solution of the problem by some method which averts the influence of zinc on these ingredients, or by keeping not only silicon and phosphorus but sulfur so low that the chemical activity of zinc on these metalloids cannot materially influence the physical properties of the iron.

W. R. BEAN (author's reply to discussion).—The discussions of Marshall and Merten in some cases confirm the data in the paper, but in others disagree. A noticeable point is that all of the work by Marshall confirms the conclusion that the embrittlement is due to a thermal effect, whereas the work by Merten indicates the opposite. A preponderance of evidence from subsequent study and tests confirms the original conclusions and indicates the deterioration to be thermal and to be dependent, in large measure, on phosphorus and silicon contents.

Marshall refers to the probability of the thermal history of the material being a factor of importance; this deserves consideration. There are indications that for a given composition the material which receives a heavy anneal may be expected to deteriorate less than that which receives a light anneal.

Study of the problem since the preparation of the paper, involving shock tests on duplicate specimens annealed under different annealing cycles, indicates that annealing may affect, at least slightly, the character of fracture produced. The 112 specimens of set No. 1 (annealed under the following cycle: time to 1450° F., 37 hr.; time 1450° F. or over, 118 hr.; maximum temperature reached 1750° F.) show slightly superior properties to the 112 specimens, of set No. 2 (annealed under the cycle: time to 1450° F., 63 hr.; time 1450° F. or over 94 hr.; maximum temperature reached 1555° F.). From these specimens, which were from 112 foundry heats, we have moved the working curve (Fig. 16) 0.04 per cent. phosphorus to the left, thus including a greater factor of safety, although there are two heats that are not consistent. When it is considered that the later results are from a shock-bending test whereas the previous chart was based largely on static bending tests, the agreement is close.

It is possible that there may be a maximum phosphorus content above which malleable is not proof against galvanizing embrittlement, regardless of silicon content and annealing treatment. In this connection we would refer to Fig. 6, which shows fractures of a group of tests in the silicon-phosphorus series in which the average phosphorus content is 0.104 per cent.; so far we have not found a single instance of deterioration where the phosphorus content was below 0.130 per cent. Nothing here stated should be constructed as eliminating silicon as a factor, for this element very definitely influences the result although it may not be of as primary importance as phosphorus.

In conclusion, the author wishes to express his appreciation to Dr. F. C. Langenberg, of Watertown Arsenal, for his coöperation in conducting the Humphrey tests on the specimens tested; and to former associates for their able assistance in carrying through the tests and compiling the data comprising this paper, particularly, Mr. H. W. Highriter, of Commercial Laboratories, New Haven, Conn., and Mr. E. S. Davenport, of the Westinghouse Lamp Co., Bloomfield, N. J.

Selecting Material for Formed and Drawn Parts

By L. N. Brown,* Birmingham, Mich.

(Canadian Meeting, August, 1923)

The use of sheet and strip metal has increased rapidly during the last few years and manufacturers have been called upon for an ever-increasing tonnage, better drawing quality, and better surface. The automobile industry is probably the most insistent in striving for a better product and a desire to reduce costs. Confronted by these demands, the mills have directed their efforts toward the delivery of material. Their personnel has been occupied largely with the problems incident to production, as any increase forced the use of new sources of raw material with their problems. These problems, combined with those met in the daily routine, have demanded immediate attention, with the result that the standardizing of heat treatments, quality, or temper, etc., have been neglected.

Considerable work in standardizing has been done by many companies, but no acceptable method has been developed. The following method, therefore, is given with the hope that some method that can be accepted by the manufacturers and users may be developed. The problem of selecting a material with such properties that the losses in processing are a minimum is interesting and difficult. A description of the properties required is likewise difficult, largely because of loose terms. Another handicap is the number of methods of testing which have not been standardized and concerning which little information is available.

In the selection of drawing or forming stock, it is advisable to eliminate as many variables as possible; consequently the method of testing the material is of the utmost importance. The method should be rapid and comparatively easy of application, as it is more important that a large percentage of the stock be tested with reasonable accuracy than that a small percentage be tested by a more accurate method. It is necessary to consider the process by which the material is made and samples must be selected with care and must be representative, as there is liable to be a great variation in any shipment. The results of the tests of material received during one year for one part are shown, graphically, in Fig. 1. A print of the part to be made was sent with the purchase orders in each case.

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* Inspection Division, Maxwell Motor Corpn.

Some plants have found it advantageous to use one class of material and, by processing, have been able to produce certain results. Another plant may work on less expensive material but produce the same result by varying the method of handling. A third plant may use more expensive material than either and produce comparable results with less hand work. The equipment, space, and facilities for handling are important factors and the ultimate choice may be governed by any one of about a dozen elements.

The method here described shows one plan of attack that has been satisfactory in so far as it gives a quick, fairly accurate check. The manipulation is simple, the first cost low, and the upkeep not excessive. The testing requires no special training and, after standards are set, may be performed by any one. The machine used is the Olsen ductility machine with pressure-gage attachment. In principle, a plunger is pressed into the stock, which is held between two dies. The depth of impression is shown by a dial micrometer and the pressure exerted is indicated by a pressure gage. The apparatus is subject to some criticism but seemed to offer the best possibilities at the time the work was started.

Inasmuch as that property vaguely called "hardness" is indicative, in a general way, of certain physical properties, it was thought that if a method for measuring the hardness with this machine could be devised, it would aid greatly in solving the problem. A large number of tests were made and, from the results, charts were developed that seemed to show a marked relation between the results obtained with the standard hardness-testing machines and the results obtained by the method described later.

On 16-gage material and under (where the ⅞-in. ball is used), the machine seemed to have considerable advantage for obvious reasons. The method of operation is a slight modification of one used by several firms that have done considerable testing with this machine. It consists of taking for a "hardness factor" the number of pounds per 0.001 in. at a depth of 0.250 in. This is obtained by dividing the reading on the pressure gage at a depth of 0.250 in. by the thickness of the stock, in thousandths of an inch. One of the reasons for using this figure instead of the direct reading obtained is that a correction is necessary for variation in gage in order that results may be comparable. To get com-

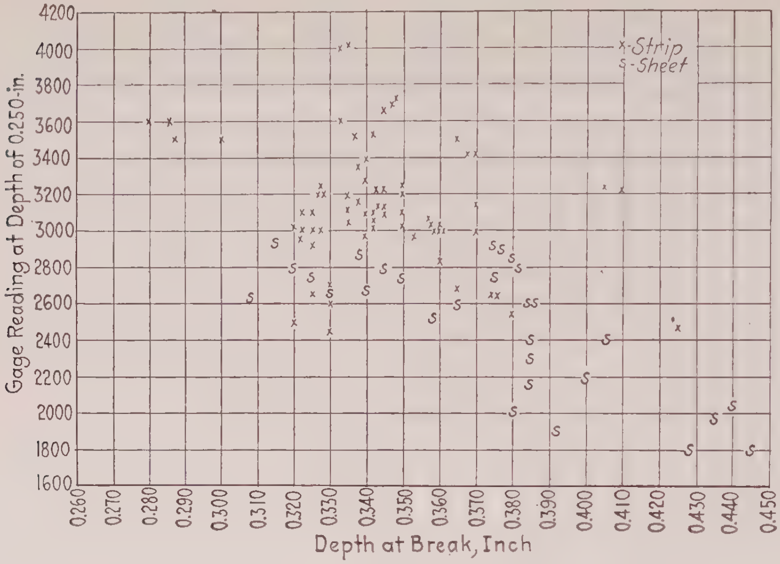

FIG. 1.—RESULTS OF TESTS OF 20-GAGE MATERIAL RECEIVED, FOR ONE PART, DURING ONE YEAR.

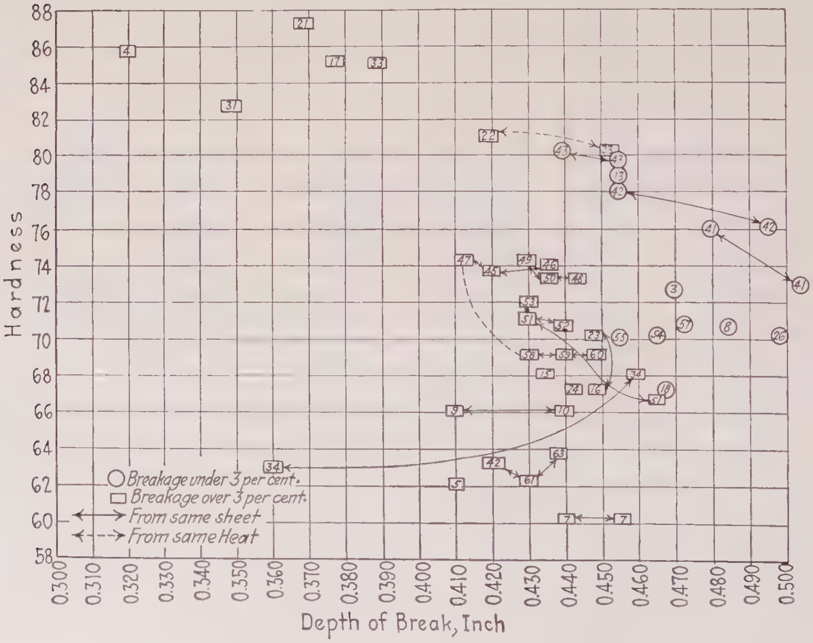

FIG. 2.—RESULTS OF TESTS OF 16-GAGE, BLUE-ANNEALED AND BOX-ANNEALED MATERIAL RECEIVED, FOR TWO PARTS, DURING ONE YEAR.

plete data, pressure readings were taken and recorded at 0.300 in., 0.350 in., 0.400 in. etc., as well as the depth and pressure reading at the point of rupture. The hardness factor and "depth" at rupture will be discussed later.

The suitability of a material usually depends on: (1) Percentage of breakage on drawn jobs or defectives on formed jobs; (2) freedom from graining or roughness; (3) freedom from cross wrinkles, stretcher strains, etc.; (4) variation in gage. The surface finish must be considered to a certain degree. The result desired is a finished piece free from breaks and objectionable surface defects caused by processing. With so-called hot-rolled or blue-annealed material, the breakage and gage variations are the important requisites, the others becoming important as material of higher finish is used.

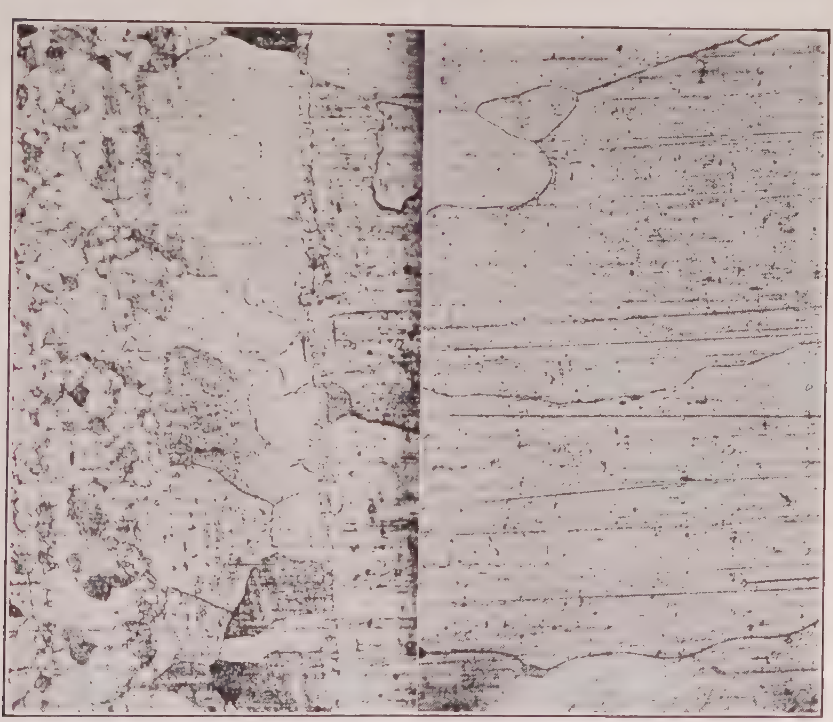

FIG. 3.—TEST STRIP NO. 34A. × 100. FIG. 4.—TEST STRIP NO. 34B. × 100.

To simplify the problem somewhat, the discussion will be limited to blue-annealed material, or stock of that character where the finish is comparatively unimportant. Just as the hardness values are indicative, to a considerable degree, of the other physical properties of heat-treated stock, the hardness factor is an indication of the other physical properties

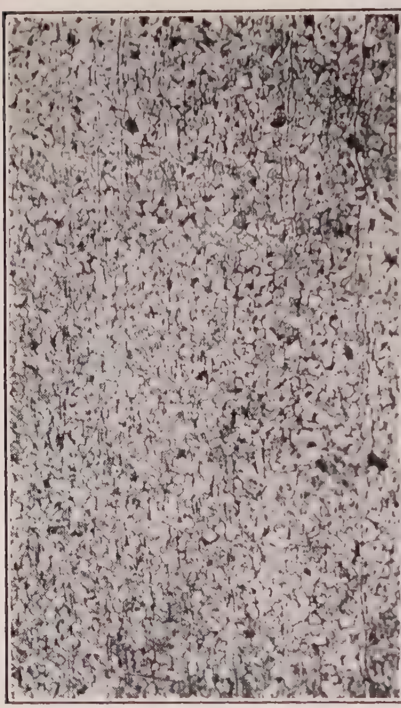

FIG. 5.—TEST STRIP NO. 61, 16-GAGE MATERIAL. × 100.

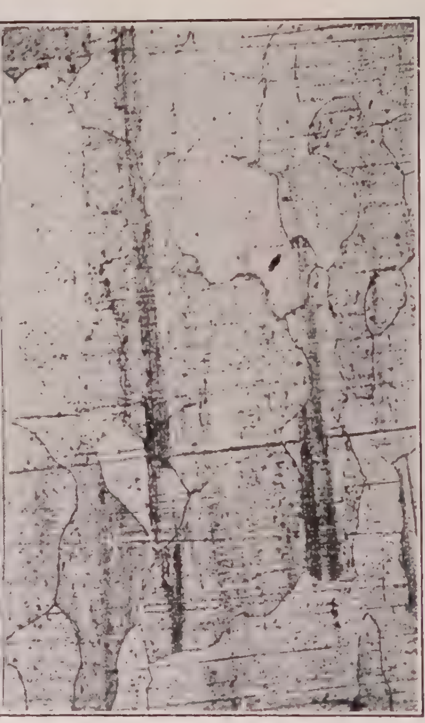

FIG. 6.—TEST STRIP NO. 5, 16-GAGE MATERIAL. × 100.

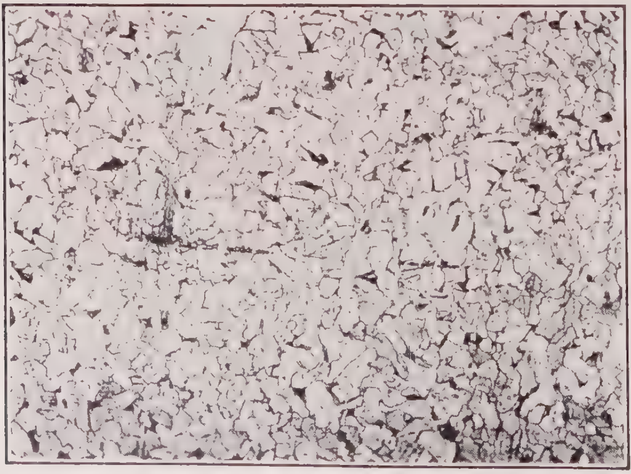

FIG. 7.—TEST STRIP NO. 48, SPECIAL DEEP DRAWING STOCK. × 100.

of the material under discussion. A percentage hardness inspection, small or total, presupposes many things such as correct analysis, homogenity, uniformity of handling, etc. Hardness is probably the best practical index we have, at present, when the history of the part in question is known—analysis, heat treatment, etc. As the history of this class of material is somewhat shrouded, the test must be carried to destruction after the hardness factor is derived, in order to obtain necessary information regarding workability.

The results of the tests of extra deep drawing material received during a year for two rather difficult parts are shown in Fig. 2. The numbers given are the numbers of the test pieces. Circles around a number indicate that the total breakage on the lot represented by that test piece was less than 3 per cent.; the numbers are placed inside rectangles when the loss is over 3 per cent. Lines connecting two numbers indicate that both samples are from the same sheet or strip. For example, the two tests marked "34" were from the same test piece and, in this case, were taken within 10 in. of each other; the breakage was 100 per cent. on a small lot. Tests 58, 59 and 60 were strips from a sheet of the same heat number as tests 45, 46, and 47, which were samples from a different sheet. These show uneven annealing. On the run of this lot of over 5000 pieces, the breakage was about 18 per cent. The breakage on the lot represented by tests 48, 49, and 50 was 4.6 per cent. on a run of 1100; this lot was bought in an emergency as deep drawing only.

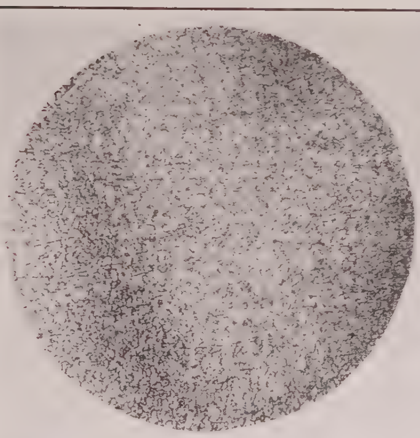

FIG. 8.—TEST STRIP NO. 275, 20-GAGE MATERIAL. × 100.

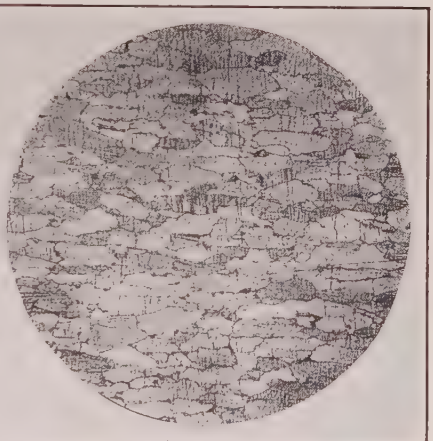

FIG. 9.—TEST STRIP NO. 163, WHICH ROUGHENED ON DRAW. × 100.

The reason so many bad samples are shown on the chart is that we were attempting to establish limits and carefully investigated all excessive breakage. The good samples represent a much greater tonnage. The limits necessary are easily discernible.

One of the peculiarities of certain grades of full finished stock is its tendency to "roughen" or "grain" when subjected to a comparatively deep draw. This is a bad condition as it necessitates a snagging, or polishing, operation. Dies are not designed to draw equally all over and the contrast of "rough" and "smooth" areas is very noticeable after japanning. It was found desirable to determine, roughly, if any depth of impression on the testing machine was comparable to the maximum effect of the presses under regular operation. Long strips were cut from one side of a sheet and impressions made with the testing machine at depths of 0.100, 0.125, 0.150 in., etc. A part was then drawn from the other half of the sheet. The character of the surface

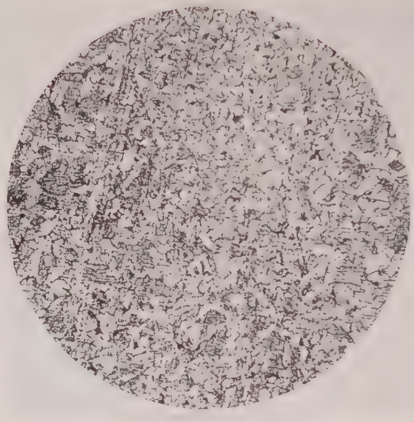

Fig. 10.—Test strip No. 160. × 100.

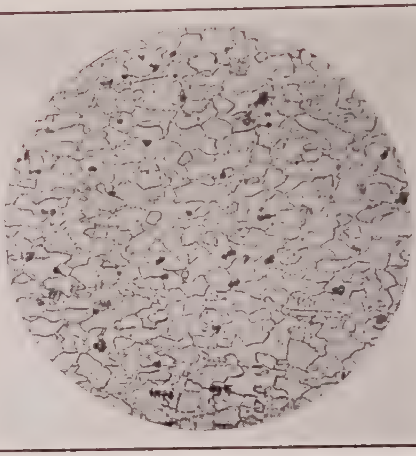

Fig. 11.—Test strip No. 220. × 100.

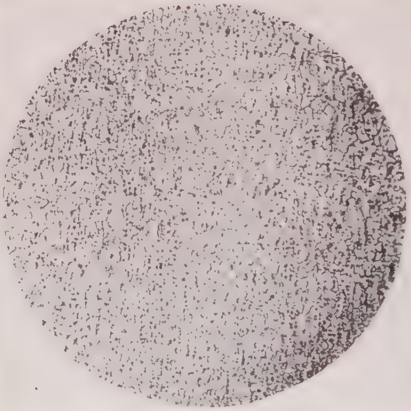

Fig. 12.—Test strip No. 237. × 100.

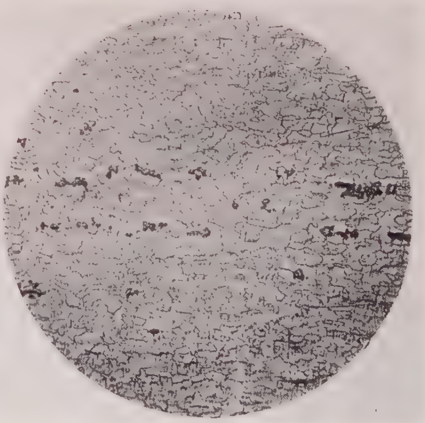

Fig. 13.—Test strip No. 222. × 100.

of the stamping at its roughest area was compared with the various impressions and the one determined whose surface was most nearly like it. This was repeated with several kinds of material. It could then be quite definitely stated that to be satisfactory a material should not roughen on test at a depth of, say, 0.050 in. greater than the depth selected, this giving a good factor of safety.

The phenomena known as stretcher strains, wrinkles, etc., can usually be discounted by observing the hardness factor carefully in connection with the total depth.

Fig. 14.—Test strip No. 181. × 100.

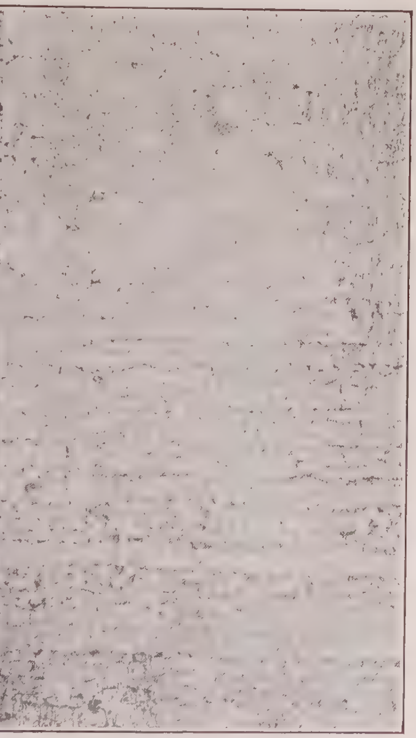

Fig. 15.—Test strip No. 159. × 100.

The permissible variation in gage has been worked out by many concerns using this material, but will not be discussed here.

At present, the best material is thought to be that which makes the best looking part and which requires the least hand work to finish. It is, therefore, frequently advisable to select material of slightly inferior finish but which will draw better than material with a higher finish. It is possible, however, to get both surface and temper for drawing or for forming.

By making a number of tests on the machine with carefully selected samples, parts of the same sheet or strip being run on the presses at a time when there is no breakage, and closely observing the finished part, limits can be established. These limits are comparable with the customary manner of setting those with a Brinell or Shore machine on heat-treated material. By this, we mean that by testing the material which works satisfactorily and that which does not work, we can decide the workability of a lot very closely without trying on the press. It is more difficult than setting limits with a Brinell machine at the present time as the factor of safety is not considered.

For testing material over 16 gage, the ½-in. ball and corresponding die should be used and the reading taken at a depth of 0.150 in. As the capacity of the machine is limited, the greatest part of the material causing trouble may be eliminated by using the hardness factor alone. In order to determine as definitely as is needed in the case of materials to be subjected to difficult draws, a punch and die arrangement may be installed on a tensile machine to run the test piece to rupture and the limits set accordingly, the procedure being the same as described above.

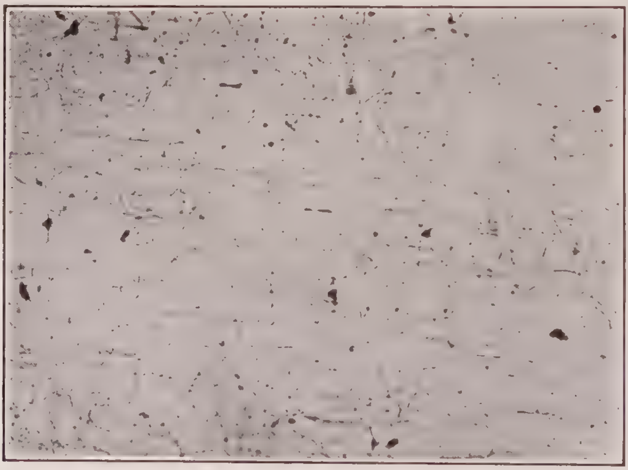

Fig. 16.—Test strip No. 691. × 100.

The effect of grain size will not be discussed, although it has a great influence on the character of the manufactured product. To get good results with microscopic analysis alone, expensive equipment is necessary, together with experienced operators and good supervision. As a method of checking other results, it is probably unequalled. The photomicrographs, Figs. 3 and 4, explain the cause of the peculiar results shown in Fig. 2. It is much more difficult to explain the failure of test piece No. 61 from the photomicrograph alone, Fig. 5. It is easy to explain the breakage of the parts made from No. 5, Fig. 6, which is also 16 gage, but somewhat more difficult to account for test piece No. 48, Fig. 7. It is likewise difficult to explain a breakage of over 80 per cent. on test piece No. 275 from the photomicrograph, Fig, 8, alone; this latter was 20-gage material.

The reason No. 163 roughened more than test piece No. 160 on the draw for the same job may easily be accounted for from Figs. 9 and 10. Nos. 220, 237, and 222, Figs. 11, 12, and 13, were for another job, one being rough, one good, and one breaking. However, it appears from testing light-gage material that the greater field for the use of the microscope is with the producers.

It is as essential that heavy-gage material be run to destruction as it is that light-gage should be, if no other inspection methods are employed; Figs. 14, 15, and 16 show what may be expected. Material No. 181 worked satisfactorily with no breakage while the breakage was very high with the other two stocks.

We have endeavored to explain the method we have found the most satisfactory; that is, for light-gage material a combination of hardness tests and determining the depth at rupture. For heavy-gage, hardness tests and either testing to destruction with a heavy-type machine or examination of structure with microscope. The information as to the limits found necessary is to be furnished to the sources of the material with the request that such material for those jobs be shipped as will work satisfactorily.

DISCUSSION

J. P. Roberts,* Detroit, Mich. (written discussion).—The selection of material for formed and drawn parts has always been of great importance to us as our requirements are much more exacting than those of almost any other industry. Our parts made from strip steel must be held to very close limits as regards both size and shape. The automobile builder requires that one or at most only a few similar parts be held to such close tolerances that they will function in exactly the same manner. In an adding machine, there are often twenty pieces that must be absolute duplicates of one another, regardless of whether the pieces were made yesterday or a year or more ago. In order to have the tools produce these similar parts day after day, the stock must be uniform in size and hardness, otherwise the numerous bends, forms, and draws would not be of the proper dimensions. Consequently we have always been anxious to employ a method of testing that would at once show whether or not the strip steel was as ordered and also could be used by the supplier in checking his production.

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* Burroughs Adding Machine Co.

We were among the first to use the Olsen ductility testing machine and did a great deal of experimenting before adopting the machine as the standard means of testing strip steel. We ran several thousand tests and the results obtained were formulated, developing a load factor, as mentioned by the author, for each of the three grades of low-carbon strip steel that we use. The formula to which this factor applies gives a straight-line curve and is $L = KT$ where L is the desired load, in pounds; T the thickness of the specimen, in thousandths of an inch; and K the factor for the grade being tested. The impression used was that made by the ⅞-in. ball and was 0.250 in. deep, the load reading being taken at that point. The factors for the three grades of steel were as follows: dead soft 65.5, quarter hard 86, half hard 101. For example, the desired load for a specimen of quarter hard stock 0.0625 in. thick was $62.5 \times 86 = 5375$ (5400 lb.).

A chart was made showing this value for each thickness of each grade purchased, making the checking of the shipment merely a matter of running the required number of tests and comparing the results with the prepared table.

After using this method of testing for some time, we discovered that the weighing device on the machine was so constructed that it might fail to give the correct load reading, through the loosening or breaking of the copper diaphragm that covers the oil container and carries the total load developed during the test. The only indication of a break in this part is an increasingly lower reading on a standard test strip. It was, therefore, necessary to have two complete weighing units, one being practically always at the manufacturers for repairs. In addition, we were unable to find two machines that would consistently give the same readings on the same piece of steel. We have had samples tested on three machines, one in the laboratory of the manufacturer, one at the steel mill, and our own, and have obtained considerably different readings on each machine.

I do not wish to create the impression that the test proposed by the author is unsatisfactory or that it will not give a true indication of the working qualities of sheet and strip material but rather wish to call attention to the weak points of the machine with which the test is made. A more rugged weighing device will eliminate the difficulties mentioned and I believe the manufacturers appreciate this fact as they have brought out attachments to be used on standard tensile testing machines. The test itself is the most satisfactory we know of and if the ductility machine can be improved to the point of real reliability there should be no question as to its usefulness to steelmaker and steel user.

X-Ray Examination of Irregular Metal Objects

By Ancel St. John, Ph.D., Long Island City, N. Y.

(Canadian Meeting, August, 1923)

Defects in the interior of metal objects are troublesome to both manufacturer and user. For the former, they frequently increase manufacturing cost through the rejection of material when defects appear after considerable work has been done. For the latter, they often cause damage to equipment, personal injury, or loss of life when a part, which has passed all inspections and has been put in service, gives way because of a hidden flaw. Methods of testing that locate such flaws without subjecting the specimen to possible, if not actual, destruction are few; x-ray examination, whether fluoroscopic or radiographic, is one.

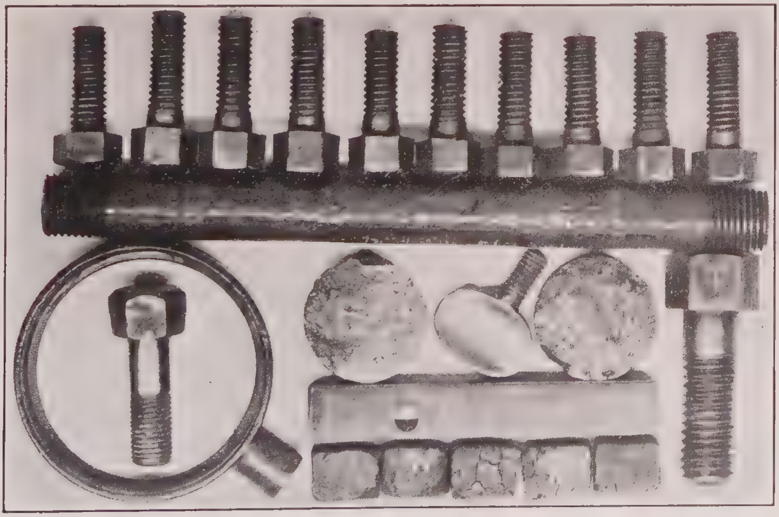

Fig. 1.—Group of irregular objects to be radiographed.

Hitherto, however, x-ray examination has been restricted to objects of nearly uniform thickness and regular outline; the inspection of such articles as those shown in Fig. 1 has been inconvenient and unsatisfactory. This figure represents a group of $\frac{5}{16}$-in. hexagonal-head machine bolts, a piece of iron pipe, a brass lens-holder from an optical bench, a 3/8-in. hexagonal-head machine bolt, two 1/4-in. rough sections of 1-in. bar stock, a brass thumbscrew, a piece of 1/4 by 1/2-in. brass bar, five pieces of cobalt, and half of a 1/2-in. cold-headed machine bolt which has been cut in two lengthwise.

The transparency to x-rays of all parts of the region exposed at one time must be of the same order of magnitude, otherwise the portions of the fluorescent screen or photographic negative corresponding to the most transparent parts will be materially over-illuminated when the illumination for the least transparent parts is correct; essential details will consequently be obscured. This is shown in Fig. 2, which is a radiograph of the group of objects of Fig. 1 exposed so as to bring out the defect in the head of the 1/2-in. bolt and some cracks in the cobalt. To overcome this difficulty, the author has developed a method of mounting specimens that makes possible the examination of such objects. Fig. 3 is a radiograph of the same group of objects by the new technique, exposed so as to bring out the defects referred to. Not only are the outlines of the objects clearly defined but many structural details, not apparent in Fig. 2, are easily discerned. The method is simple, rapid, and inexpensive and can readily be used on a commercial scale.

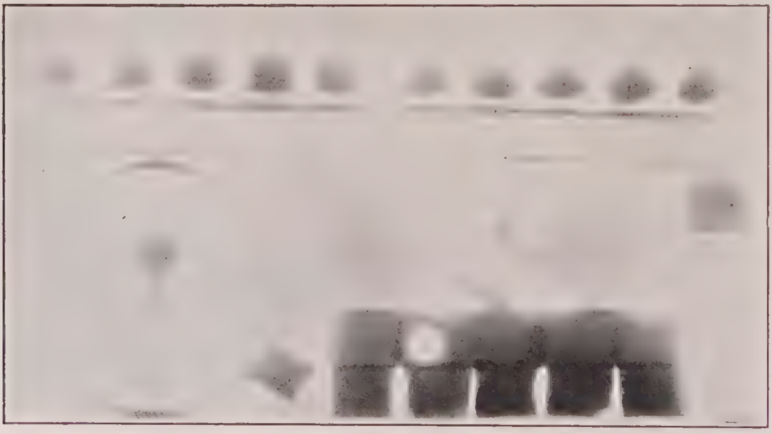

Fig. 2.—Radiograph when transparency to x-rays of all parts is not of same order of magnitude.

In this method, the specimens are surrounded with a medium of slightly different transparency to x-rays in a container with opaque walls and parallel transparent faces, as shown in Fig. 5. In this way the transparency of the entire field is made of the same order of magnitude. If the absorptive power of the surrounding medium is less than that of the specimen, its transparency will be greater and more x-rays will pass through it than through the specimen, which will thus appear as a darker area, as shown in Fig. 3. But if the absorbing power of the medium is greater than that of the specimen, less x-rays will pass through it than through the specimen, which will then appear as a lighter area, as in Fig. 4. Here the absorber used for Fig. 3 has been replaced by a more opaque medium with a transparency less than iron or cobalt but greater than brass.

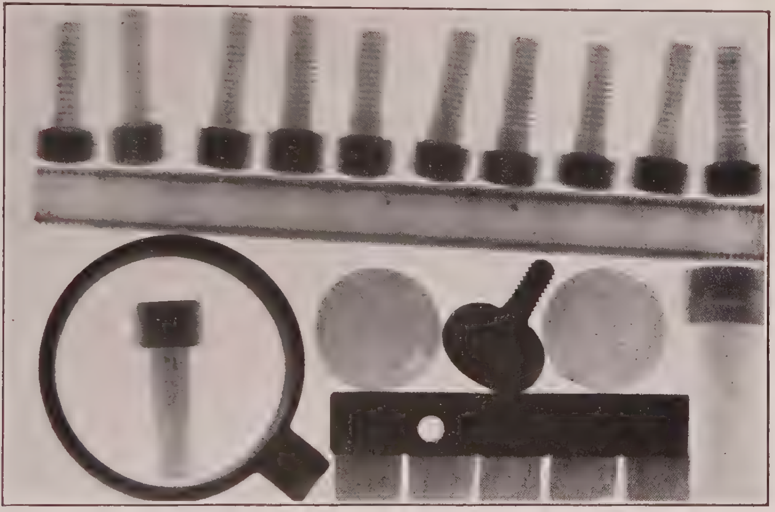
FIG. 3.—RADIOGRAPH MADE BY AUTHOR'S METHOD.

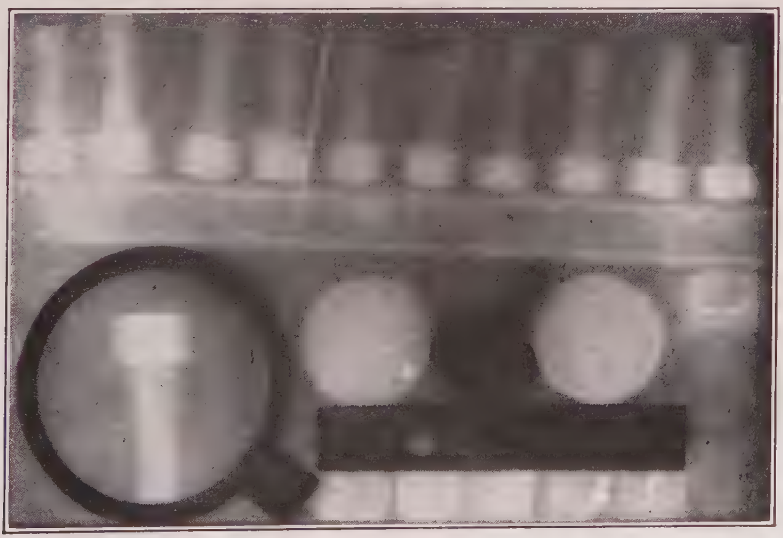
FIG. 4.—RADIOGRAPH WHEN ABSORBING POWER OF MEDIUM IS GREATER THAN THAT OF SPECIMEN.

In general, it is preferable to have the medium less absorbent than the specimen as the time required for a satisfactory exposure increases rapidly with the absorption. There are cases where the more absorbent medium is useful; for instance, it is thus possible to distinguish indentations on the surface from cavities in the interior; the former, being filled with absorber, would be darker, while the latter, being empty, would be lighter.

Before investigating the suitability of various materials as absorbers for use with tool metals a series of bars of tool stock was examined and a few containing interesting defects were selected as test specimens. Many absorbents were then studied. Granular substances give much the same appearance as spongy structures in the specimen and are useless. This is shown in Fig. 6, a portion of the radiograph of an intricate casting packed in a mixture of 100-mesh silicon and tungsten. The speckled area in the center represents a spongy structure in the casting, while the rather similar area toward the left-hand end is due to the granular structure of the absorber. Fine powders are better in this respect but it is next to impossible to pack them uniformly when dry; when mixed with water, oil or grease to form a paste, they tend to occlude air, forming cavities indistinguishable from blowholes in the specimen. This is shown in Fig. 7, representing two specimen bars in a paste of white lead and water. The best substance is a liquid.

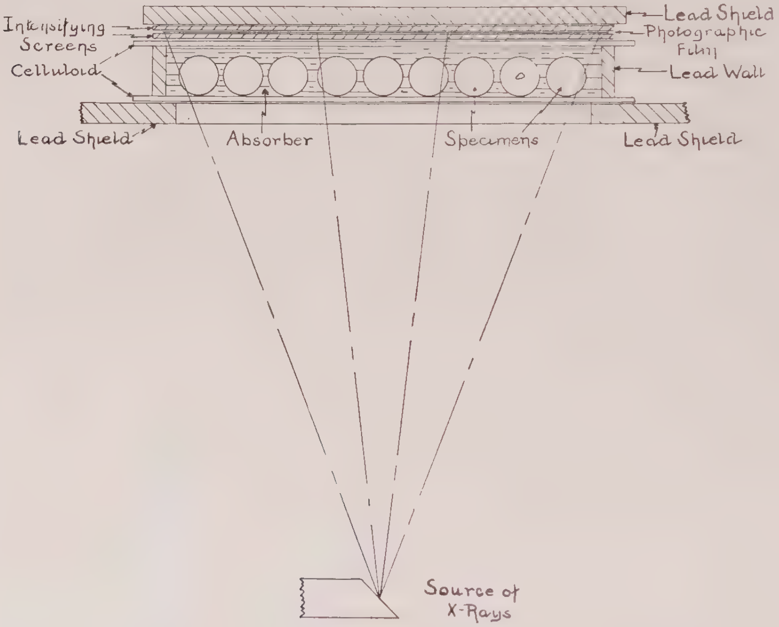

FIG. 5.—ARRANGEMENT OF APPARATUS FOR RADIOGRAPHING OBJECTS OF VARYING THICKNESS.

The choice of liquids is restricted to compounds of high density, containing a constituent of high atomic weight so that the x-ray absorption may be high. Convenience demands that the liquid should not react appreciably with the materials to be examined, should mix in all proportions with a common liquid, and should be non-volatile. Of the numerous liquids tested, methylene iodide, CH_2I_2, has proved most satisfactory. Its density is 3.34 gm. per cu. cm.; its principal constituent, iodine, has a very high x-ray absorption, so that it can be used with alloys containing 25 per cent. of tungsten; it mixes in all proportions with benzene, so that dilutions suitable for use with less absorbent alloys can be made; it is less volatile than water; and it reacts but slightly with most metals. Its objectionable features are a characteristic and persistent odor, a tendency to react with aluminum, and a rather high price. The actual cost in service, however, can be kept low by rinsing specimens and containers in benzene after exposure, saving the rinsings, and using them to replace benzene lost by evaporation from the dilute solutions. In this way one lot has been used day after day for nearly six months with a wastage of about 10 per cent. The reaction with aluminum is a decomposition accompanied by the evolution of heat and the liberation of iodine vapor. It takes place suddenly after the metal and liquid have been in contact for some time and is apparently encouraged by the presence of other metals in contact with the aluminum, by heating, and possibly by exposure to x-rays. It can be avoided by rinsing specimens and vessels at once after making an exposure.

FIG. 6.—RADIOGRAPH OF PART OF INTRICATE CASTING PACKED IN MIXTURE OF 100-MESH SILICON AND TUNGSTEN.

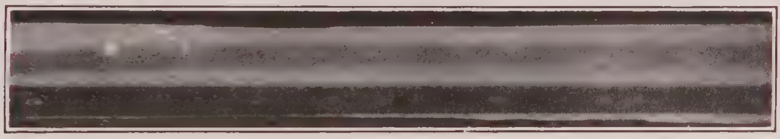

FIG. 7.—RADIOGRAPH OF TWO BARS IN PASTE OF WHITE LEAD AND WATER.

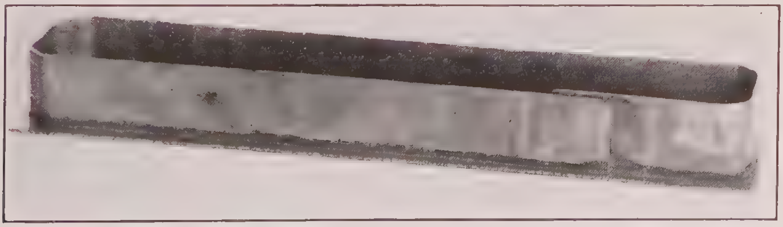

FIG. 8.—LEAD-WALL CONTAINER IN HORIZONTAL POSITION.

When methylene iodide was first used, the specimens were immersed in the liquid contained in an aluminum pan. After an experience with the reaction mentioned, a container with lead walls and celluloid faces fastened together with celluloid cement was devised; this is shown in Fig. 5. These containers are easily made to suit any thickness of material and can be adapted for use with the faces horizontal, as in Fig. 8, vertical, as in Fig. 9, or at an intermediate angle, say 45°, as in Fig. 10.

To gain some idea of the size of the smallest cavity that could be detected with certainty, a penetration meter was made; this consists of a set of aluminum strips 0.008 in. thick fastened together with the ends offset to form a series of shallow steps. In Fig. 11, *A* shows the penetration meter on a bar 1 in. wide and $\frac{3}{16}$ in. thick, containing no tungsten; *B* a $\frac{7}{32}$-in. round rod of the same stock; *C* a $\frac{7}{32}$-in. round rod containing 6 per cent. tungsten; *D* a $\frac{1}{4}$-in. square bar containing 3 per cent. tungsten; *E* a $\frac{5}{16}$-in. square bar containing 16 per cent. tungsten; and *F* a $\frac{5}{16}$-in. round rod containing 5 per cent. tungsten, all immersed in $\frac{1}{2}$ in. of a 70-per cent. solution of methylene iodide in benzene. A single step of the meter makes a distinct difference in the density of the radiograph; and as differences of the same character are observable in *B* it is evident that cavities only 0.008 in. thick can be detected. The outlines of bar *D* are distinguished with difficulty on account of the great similarity in transparency of this bar and the medium; this suggests the possibility of supporting small objects in frames of sheet metal, say copper, and diluting

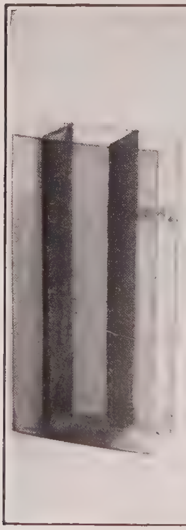

Fig. 9.—Lead-wall container in vertical position.

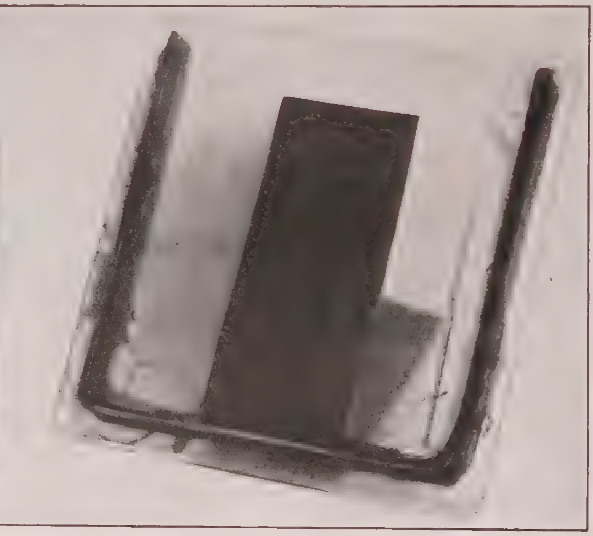

Fig. 10.—Lead-wall container set at intermediate angle.

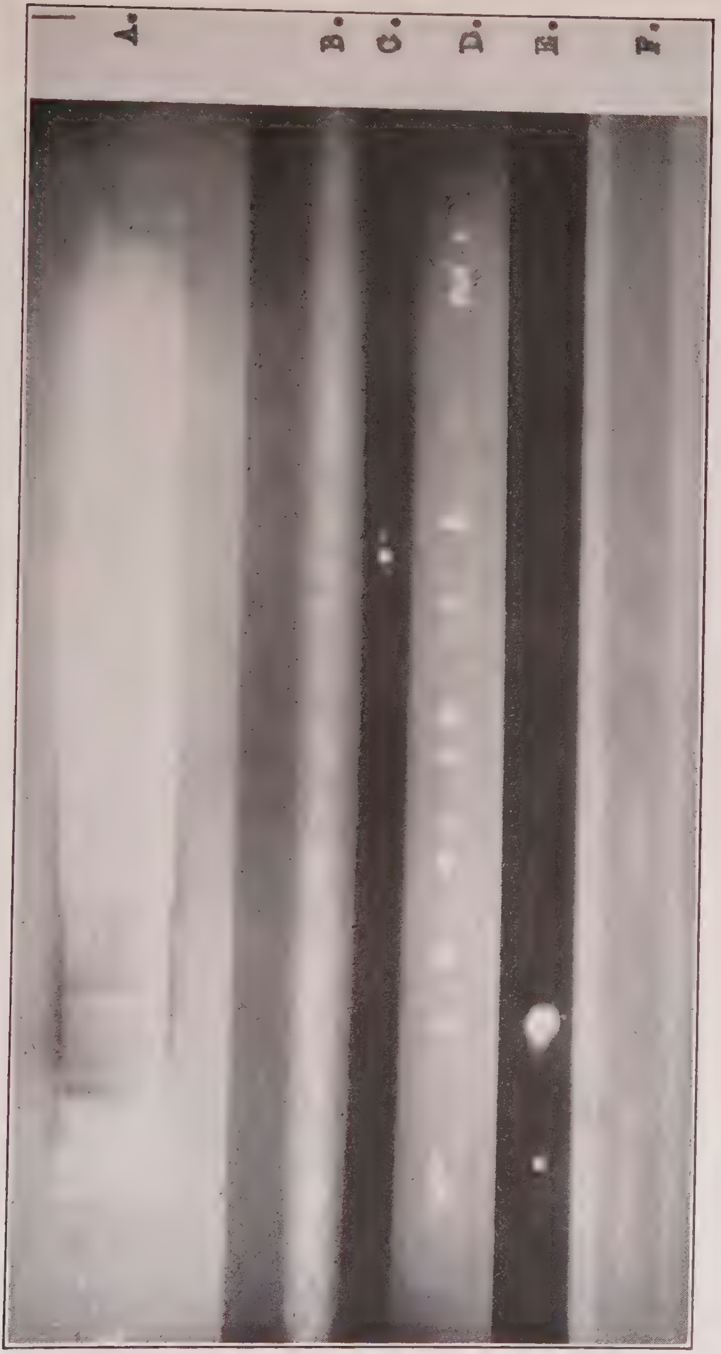

FIG. 11.—ILLUSTRATING USE OF PENETRATION METER.

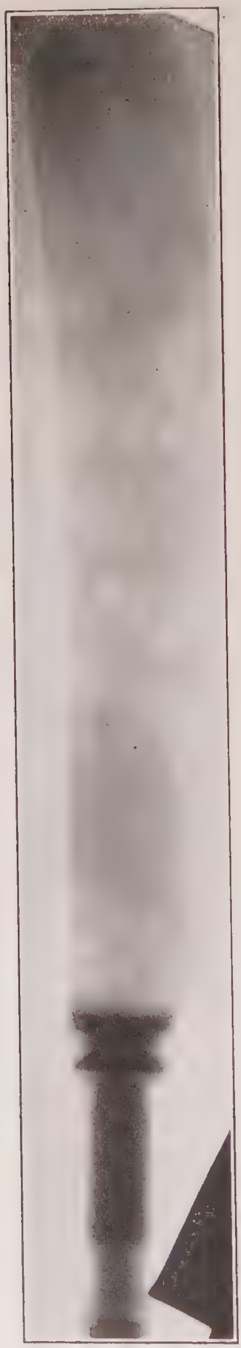

FIG. 12.—FORGING OF A TABLE KNIFE.

the liquid until its absorption is the same as that of the frames. These would then not be visible in a radiograph.

This figure also exhibits the three types of cavities observed during the investigation. The spots in *E*, the rounded portion of the large spot near the right-hand end of *D*, and the rounded part of the principal spot in *C* are blowholes. The jagged spots in *D*, the transverse spots in *F*, and the distinctly transverse spots in *B* are shrinkage cavities. The cloudy area near the left-hand end of *F*, the thread-like patches in *C*, and the faint patches in *B* are spongy structures like that in Fig. 6. None of these specimens had slag inclusions. Distorted remnants of these are, however, observable in Figs. 3 and 4. The straight filaments through the heads, and even through the entire length, of several of the 5⁄16-in. bolts are due to slag that has been drawn out during the rolling. The fact that these are not further distorted in the bolt heads shows that these bolts were cut from hexagonal bars and not cold-headed. In the latter case, they would have been spread out transversally, like the flaw in the fragment of the ½-in. bolt, which is known to have been cold-headed. The rest of the slag filament is probably in the other half of the bolt.

To show the variety of objects examined in this way, a few examples have been selected at random. Fig. 12 shows a forging for a table-knife; Fig. 13 a set of races for ball bearings; Fig. 14 shows 98 forged balls for use with such bearings, 90 of which have no detectable flaw, the remaining 8 being faulty. Fig. 15 shows two plug gages of tool metal containing considerable tungsten cast upon steel cores; one of these has no detectable flaw, the other has marked defects within 0.02 in. of the surface. These radiographs were all made on the rough forgings before any machine work was done. The faulty balls and the defective plug gage were, of course, rejected. Without inspection, the defect in the gage would have been encountered after considerable expensive machining had been done, that is, in the course of manufacture. The flaws in the balls would probably have escaped attention until they failed in service.

These two examples illustrate two cases in which routine *x*-ray inspection may be justified. In the first, it is a question of dollars and cents. If the manufacturing procedure cannot be developed so as to eliminate obnoxious defects, and if the cost of machining before the eventual rejection is large compared with the cost of *x*-ray examination, inspection before machining would be profitable. In the second case it may be a question of reputation, or even a question of human safety; in the latter case the cost should not figure. As a matter of fact, routine inspection should not be very expensive.

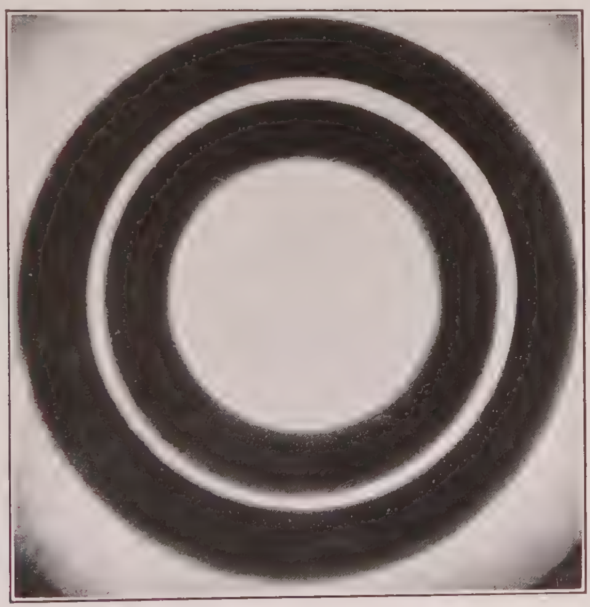

FIG. 13.—BALL-BEARING RACE.

The value of radiographic examination in the development of new products and new processes cannot be too greatly stressed. A single radiograph gives more information as to the character and extent of cavities and inclusions than a great number, one is tempted to say than any number, of sectionings and does so without damaging the specimen. It is thus possible to submit a specimen containing known defects to various mechanical tests and to correlate the results to the internal structure disclosed by the radiograph, or to section it at the places most favorable for direct examination of the defects. This use of x-rays has been discussed by Lester.[1]

For a long time, the use of x-ray examination in the metallurgical industry has been limited by the comparatively thin specimens that could be penetrated successfully and by the difficulties encountered in mounting specimens of varying thickness or irregular shape. But Lester shows that

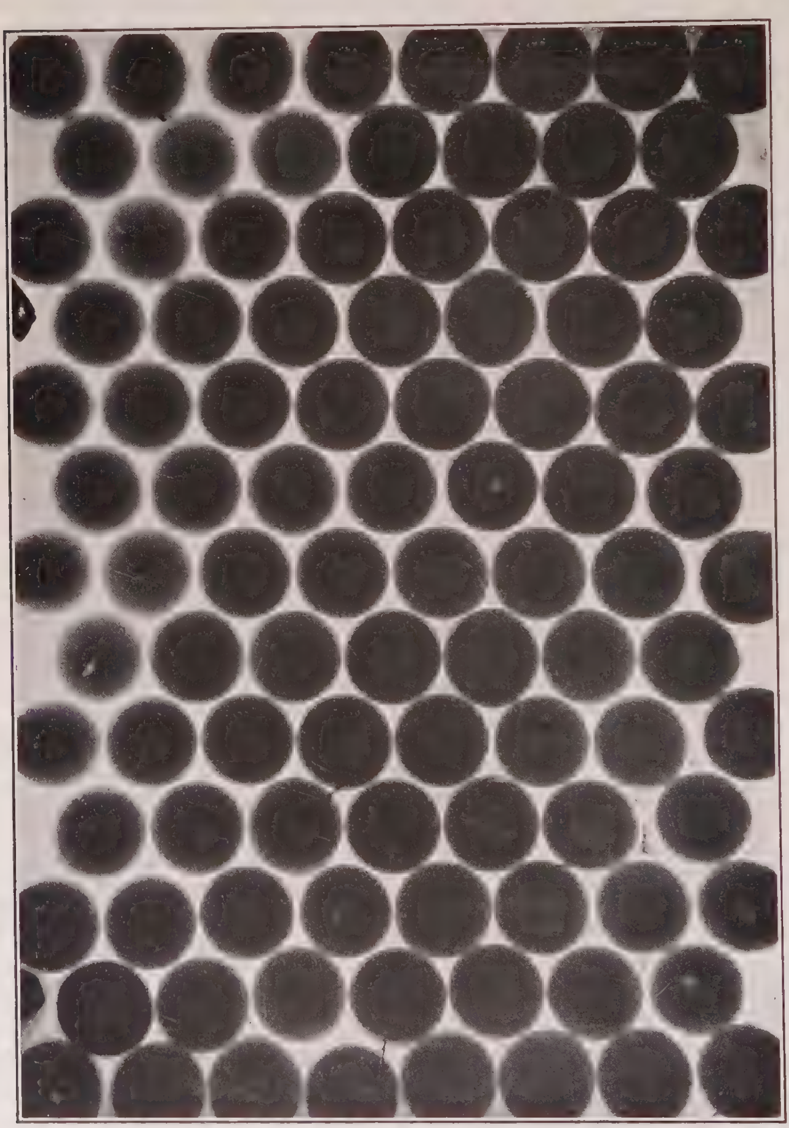

FIG. 14.—RADIOGRAPH OF FORGED BALLS FOR USE IN BEARINGS.

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[1] X-Ray Examination of Steel Castings. *Chem. & Met. Eng.* (1923) **28,** 261–267.

steel 3 in. thick can be radiographed in ½ hr. with apparatus now on the market. When the demand justifies the expense, the makers of tubes and generating plants will undoubtedly develop apparatus extending this limit considerably. The present paper has shown that good radiographs of objects of any external form can be secured with the new procedure and that the procedure itself is simple. The limitations hitherto imposed by shape have thus been removed. It is therefore permissible to say that any object that can be penetrated sufficiently by the x-rays can now be radiographed successfully.

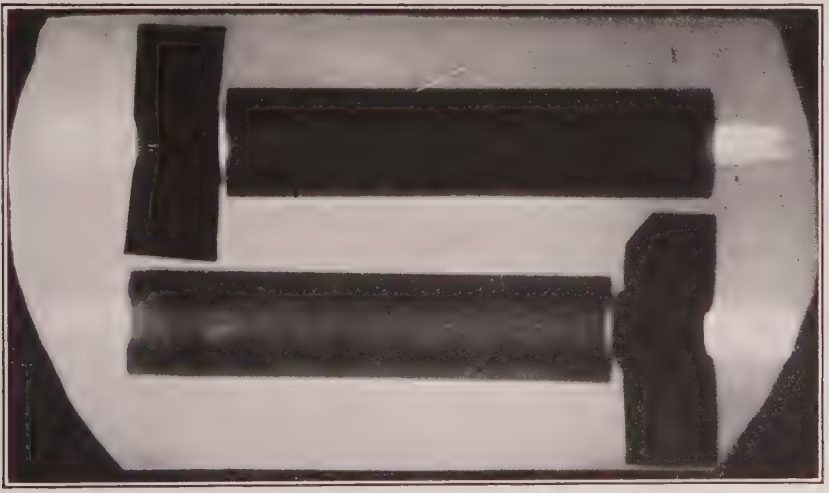

Fig. 15.—Plug gages.

In conclusion, the author wishes to acknowledge the assistance of Leroy C. Werking, who made most of the radiographs and contributed valuable suggestions during the search for the best absorbing medium, and the coöperation of W. P. Melville in improving the photographic technique.

DISCUSSION

H. H. Lester, Watertown, Mass. (written discussion).—All persons who have taken radiographs of metals know that it is not easy to get good radiographic results where there is considerable variation in the thickness of metal in the region of the picture. It is difficult to distinguish between surface markings and internal flaws. Surface pits are very often machined away in the finished product and, whether they are important or not, as they may be examined without the x-ray it is not desirable to have them appearing in the x-ray negative. The author's method eliminates this at least to a considerable extent.

As the title to the paper indicates, this method seems to be limited to small solid objects that can be immersed without injury in a solution that acts as an auxiliary absorber to even up the transparency. Of course it is evident that the exposure will be controlled by the depth of the solution that covers the specimen.

At the Watertown Arsenal *x*-ray laboratory, we have found that the sensitiveness of the *x*-ray method in determining small faults is a function of the metal thickness and becomes less as the thickness of material to be radiographed increases. For this reason, we would expect the detectability of a flaw to depend on the depth of the solution where the specimen is immersed in a fluid-absorbing material.

We have, for several months, been using a method similar to that described in this paper. Our specimens are too large to be immersed in a tank and often are of such a nature that immersion would not be possible even if the size permitted. We have used wax impregnated with lead salts and, sometimes, with bismuth. The wax is spread on the thin portion of the casting on the side toward the source of *x*-rays; we have used also thin sheets of metal to even up the transparency. We have found it necessary to place the auxiliary absorbing material on the side of the metal away from the film because the heavy metal particles suspended in the wax give off considerable scattered radiation, which escape from this auxiliary absorbing material through the fairly transparent wax. When the wax is spread on the side of the metal away from the film, this scattered radiation is absorbed by the metal and does not reach the film. Scattered radiation from lead will penetrate $\frac{1}{8}$ in. of steel when operating with 200 kv. on the bulb.

At first sight, it seems that the author's method is peculiarly liable to cause fogging of this sort. He uses a solution rather than a suspension; this may make a difference. Our experience has been, however, that fogging due to scattered radiation must be avoided and that auxiliary absorbing materials give a great deal of scattered radiation. Metal sheets can be used on the film side of the specimen because the effective scattered radiation in this case proceeds only from the surface of the sheet while, in a suspension and probably in a solution, the scattering centers may lie at a considerable distance from the surface.

As the author points out, when discussing paste absorbers, care must be taken to insure even distribution of the absorbing material in the suspending medium and also care must be observed that air bubbles are not included. In our work, the size of air bubbles has been less than the size of the flaws we were interested in. In cases where this was not so, we have not used the auxiliary absorber but have taken several exposures for the different thicknesses of metal involved.

ANCEL ST. JOHN.—In my practice, there is nothing but a thin fheet of celluloid, the enevlope in which the photographic film is placed, and the intensifying screen between the absorbing medium and the film, certainly not over $\frac{3}{32}$ in. of material, so that the effect of scattered radiation is greatly reduced. Probably, at the voltage at which I was working, the amount of scattered radiation from the iodine contained in the iodide is not sufficient to cause any trouble, because it did not. I got very clear, sharp negatives.

W. D. Coolidge,* Schenectady, N. Y. (written discussion).—The author has eliminated the worst limitation of metal radiography, that relating to the shape of the object. As to the long time required for making exposures of thick objects, this laboratory has recently developed an x-ray tube, with a water-cooled anode, which operates satisfactorily at 50 milliamperes and 250,000 volts (crest value). Under $\frac{1}{2}$ or 1 mm. of copper, this gives about fifteen times the x-ray intensity now obtainable with the high-voltage outfits used by the medical progression, operating at 5 milliamperes and 200,000 volts. The use of such a tube in metal radiography would greatly decrease the required time of exposure. The tube has been developed for experimental work in the medical field and has a source of x-rays ("focal spot") that is too large for radiographic work. However, a similar tube to handle this amount of energy, or even more, might be developed with a focal spot small enough for metal radiography.

Ancel St. John.—It has been asked to what extent this process is at present being used? I do not know. The Watertown Arsenal is using its outfit in development work, but not in commercial inspection. We are using our laboratory equipment in development work. We have used it to some extent on what might be called a semicommercial scale; that is where, for trial purposes, we wanted to be able to have tests made on material that was as nearly 100 per cent. perfect as we were able to select. We have examined a good many hundred balls and quite a number of races for ball bearings, and have decreased the proportion of rejections very materially. We did that by finding the defective ones and returning them to the manufacturer, who gradually improved his methods to remove the defects. The cost of doing this is now somewhat greater than would permit the very general commercial use except: (1) Where human life is dependent on the quality of the material used, and there is a reasonable expectation that, because of a low factor of safety or very light construction, unless the material is 100 per cent. perfect an accident might happen. (2) Where the material can be inspected in the raw or rough state before any manufacturing processes are carried out, in case those manufacturing processes are in themselves expensive. For instance, if the cost of inspection is ten cents a pound for a given type of material, and the cost of subsequent manufacturing is \$1, \$2, \$5, or \$10 a pound.

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* Research Laboratory, General Electric Co.

Radiographic examination is very satisfactory, but before we are through, if it is possible, we must have an *x*-ray eye, that is, we must have some sort of a detector involving the use of photo-electric or ionization effects and radio-amplifiers—some sort of a mechanism that will run steadily over the surface to be examined and, whenever it passes a flaw, will move a pointer, ring a bell, or drop paint on the material alongside of the flaw. The device must be entirely automatic and sufficiently sensitive to get the tolerable flaws. That means a long and tedious investigation, but we have the thing in mind and, I believe that the "makings" of this eye are at present available. Whether it can be made sufficiently sensitive so that the device can, with the amount of *x*-rays that come to it in ½ or 1 sec., make a sufficient record to do these things I have indicated (when the material being examined would require a ½ hr. exposure to the most powerful *x*-rays at present available) is a thing that I am unable to state.

WHEELER P. DAVEY,* Schenectady, N. Y. (written discussion).—The author's method of immersing irregularly shaped metals in a liquid of about the same opacity for *x*-ray represents a real advance in metal radiography and goes a long way toward removing one of the greatest objections to the *x*-ray examination of manufacturing materials. The one step to be taken before the radiography of metals is perfected, is using a Boucky diaphragm in connection with the author's method. The great mass of material that the *x*-rays traverse gives rise to considerable scattering of *x*-rays. This scattering may be likened to the effect of a cloud of steam upon a searchlight beam, so that the photographic plate receives not only the shadow image of the substance to be radiographed but also a foggy effect due to the cross-fire of scattered *x*-rays. This foggy effect may be largely eliminated by the Boucky diaphragm, using much the same apparatus and technique that the medical men now use when radiographing through the thicker portions of the human body. The combination of the two methods should make it possible to recognize defects too minute to be noticed when the materials are merely immersed in the author's solution. It is my opinion that this method represents the greatest single advance in the radiography of metals since the original work done in this laboratory.

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* Research Laboratories, General Electric Co.

Observations on the Occurrence of Iron and Silicon in Aluminum

By E. H. Dix, Jr., M.E., M.M.E., Dayton, Ohio

(New York Meeting, February, 1923)

All commercial aluminum contains small percentages of copper, iron, and silicon as unavoidable impurities. The purest metal obtainable commercially, special grade high purity ingot, contains a maximum of 0.50 per cent. impurities of which the usual minimum amount of copper is 0.01 per cent.; silicon, 0.12 per cent.; and iron, 0.20 per cent. This grade is too costly for most purposes. The two standard grades, Nos. 1 and 2, containing a minimum of 99.0 and 98.0 per cent. aluminum, respectively, are more generally used. Other grades of secondary aluminum often contain as high as 3 or 4 per cent. of impurities, which include 1 per cent., or more, of zinc. Specifications for high-grade aluminum-casting alloys generally allow a maximum of 1.7 per cent. for total impurities. The S. A. E. Specification No. 33, which is intended to cover alloys made from secondary aluminum ingot, gives the iron maximum as 1.5 per cent. and the maximum for silicon (including manganese and tin) as 0.75 per cent.

The effect of even relatively small amounts of impurities on the physical properties of ingot aluminum is shown by the following comparison of two lots of aluminum, which may be taken as representative of special grade and grade No. 2 ingot, respectively.

Analyses

	Special Grade, Per Cent.	Grade No. 2, Per Cent.
Copper	0.01	0.32
Silicon	0.12	0.32
Iron	0.42	0.82
Aluminum	99.45	98.54

PHYSICAL PROPERTIES WHEN SAND CAST

	SPECIAL GRADE	GRADE NO 2
Tensile strength, pounds per square inch	10,650	14,400
Elongation, per cent. in 2 in.	33.3	19.3
Brinell (500 kg.)	20.0	30.0
Scleroscope (mag. hammer)	3.0	5.0
Specific gravity	2.68	2.69

The metallographic structures, as exhibited in a section cut from small notched-bar ingots, are shown in Figs. 1 to 6. Fig. 1 shows the normal arrangement of the network of impurities of the special grade ingot, and Fig. 2 shows the formation of these impurities at higher magnification; Figs. 3 and 4 show the network arrangement in the grade No. 2 ingot. Fig. 3 is the average arrangement of impurities and Fig. 4, a segregated area near the pipe in the top of the ingot. Figs. 5 and 6 show at higher magnification the formation of impurities in the normal and segregated areas, respectively.

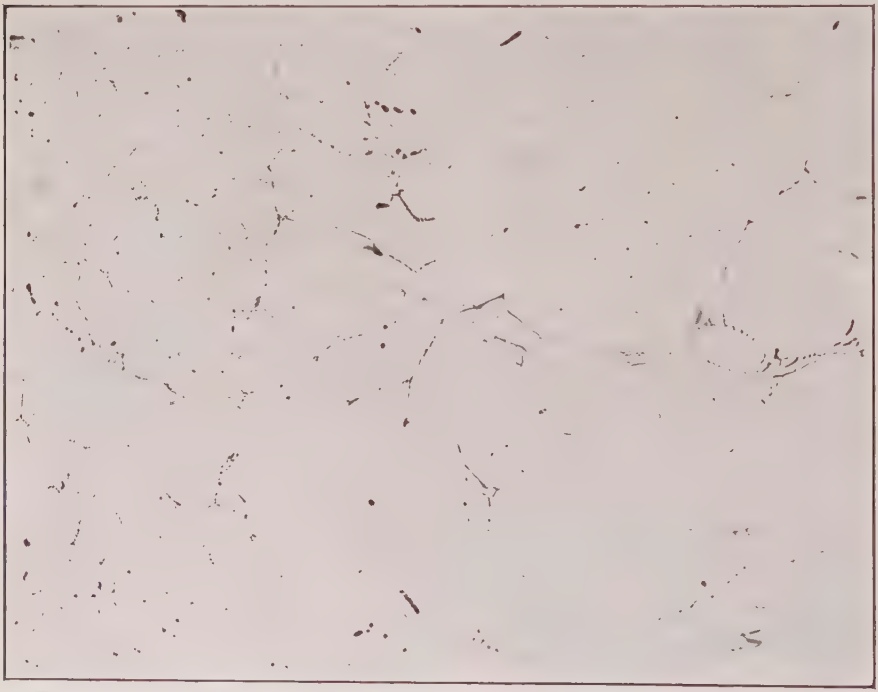

FIG. 1.—TRANSVERSE SECTION FROM SMALL NOTCHED-BAR INGOT; UNETCHED; ALLOY, CU 0.01 PER CENT., SI 0.12 PER CENT., FE 0.42 PER CENT. 100 ×.

FIG. 2.—UNETCHED; ALLOY, SAME SECTION AS FIG. 1. 500 ×.

Copper forms the compound $CuAl_2$ and, in the small amounts present in these two samples, is held in solid solution in the aluminum, and so does not appear as a separate constituent. The occurrence of the iron and silicon is, however, a matter requiring careful consideration. The metallography of aluminum ingot was considered by R. J. Anderson[1] in September, 1919. He accepted the view held by Gwyer[2] and Hanson and Archbutt[3] in regard to the iron constituent, namely, that iron forms the compound $FeAl_3$, which is practically insoluble in aluminum. He also mentioned that silicon occurs quite often as elementary silicon. At that time there was no suggestion of any reaction between the iron and silicon. However, in August, 1919, Merica, Waltenberg, and Freeman[4] noted the occurrence of a constituent similar to $FeAl_3$, but which they found was

FIG. 3.—TRANSVERSE SECTION FROM SMALL NOTCHED-BAR INGOT; UNETCHED; ALLOY, CU 0.32 PER CENT., SI 0.32 PER CENT., FE 0.82 PER CENT. 100 ×.

FIG. 4.—ALLOY, SAME SECTION AS FIG. 3; ETCHED 10 PER CENT. NaOH FOR 5 SEC. 100 ×.

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[1] R. J. Anderson: Metallography of Aluminum Ingot, *Chem. & Met. Eng.*, (1912) **21,** No. 5.

[2] A. Gwyer: Constitution of Binary Alloys of Aluminum with Iron, Copper, etc. *Zeit. Anorg. Chem.* (1908) 113.

[3] Hanson and Archbutt: Micrography of Aluminum and its Alloys. *Jnl.* Inst. of Metals (1919) **21,** 291.

[4] Merica, Waltenberg, and Freeman: Constitution and Metallography of Aluminum and its Light Alloys with Copper and with Magnesium. *Trans.*(1920) **64,** 3; also Bureau of Standards., *Sci. Paper* 337 (1919).

influenced by the percentage of silicon present. They, therefore, called this constituent X and suggested that it contained aluminum, iron, and silicon. This constituent X was described as being slightly darker than $FeAl_3$. In November of the same year, Wills[5], in a lecture before the Birmingham Metallurgical Society, described a steel-gray constituent in addition to $FeAl_3$, which he referred to as FeSi, a compound of iron and silicon, although he stated that the composition of this substance was not definitely known. The microstructure of the silicon-aluminum alloys has been considered by Fränkel[6] and Roberts.[7]

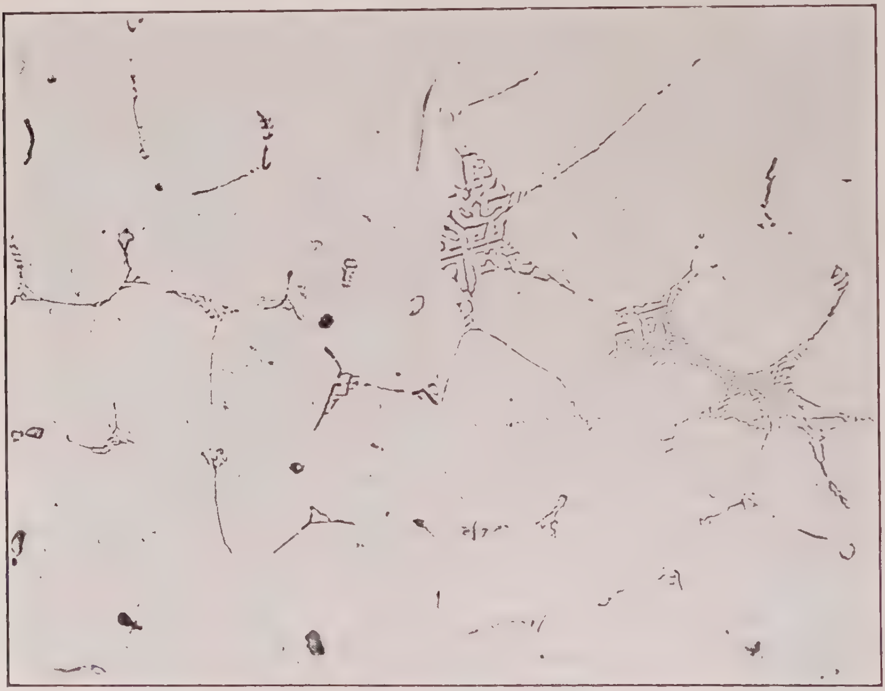

FIG. 5.—ALLOY, SAME SECTION AS FIG. 3; ETCHED 10 PER CENT. NaOH FOR 5 SEC. 500 ×.

FIG. 6.—ALLOY, SAME SECTION AS FIG. 4; ETCHED 10 PER CENT. NaOH FOR 5 SEC. 500 ×.

The Eleventh Report of the Alloys Research Committee[8] gives an account of the first thorough investigation of the iron-silicon relations. As a result of a systematic study of alloys containing up to 8 per cent. each of iron and silicon, it was possible to explain the mechanism of the formation of the X constituent, namely, that $FeAl_3$ first separates as a binary complex and then reacts with a portion of the liquid to form a new constituent, which they have referred to as constituent X. The relations between $FeAl_3$, the constituent X, and silicon are shown in a series of five diagrams representing 1, 2, 4, 6, and 8 per cent. iron, each with the silicon content of from 0 to 8 per cent. In each diagram, there is an extremely narrow field (small percentages of silicon) in which the $FeAl_3$ exists alone; a second wider field in which both $FeAl_3$ and the constituent X exist in equilibrium; and a third still wider field in which the constituent X exists alone. Beyond this field, the constituent X and silicon exist as a ternary eutectic with aluminum. Other constituents are also mentioned, which have a similar appearance and are difficult to distinguish from $FeAl_3$. Two photomicrographs are shown illustrating $FeAl_3$ and the constituent X. One shows a long needle with a dark core occurring in an alloy containing silicon 1 per cent., iron 8 per cent., balance aluminum, and it is stated that the core is $FeAl_3$ which is surrounded by the constituent X. The other photomicrograph shows an elongated particle of constituent X in an alloy of silicon 2 per cent., iron 8 per cent., balance aluminum.

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[5] L. J. Wills: Micrography of Aluminum Alloys. *The Met. Ind.* (British) (Nov. 21, 1919).

[6] W. Fränkel: Silicon-Aluminum Alloys. *Zeit. Anorg. Chem.* (1908) 154.

[7] C. E. Roberts: Silicon-Aluminum Alloys. *Trans.* Chem. Soc. (1914) 1383.

[8] Eleventh Report to the Alloys Research Committee of the Institute of Mechanical Engineers (August, 1921).

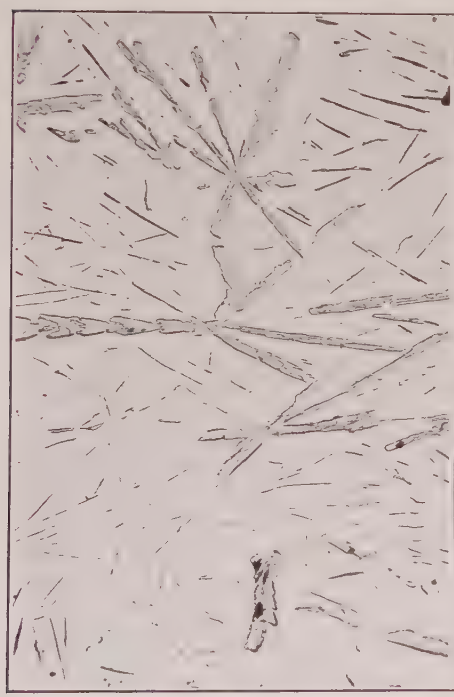

Fig. 7.—Alloy, Fe 5.38 per cent., Si 0.15 per cent.; as cast; unetched. 100 ×.

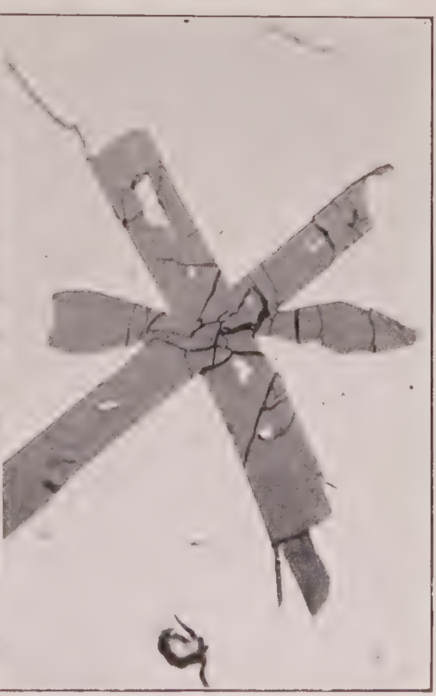

Fig. 8.—Alloy, same section as Fig. 7; as cast; etched 25 per cent. HNO_3, quenched for 30 sec. 1000 ×.

Owing to the similarity in appearance between $FeAl_3$ and the constituent X, great difficulty is experienced in distinguishing between the two except when they occur joining each other. In order to study the appearance of these two constituents, when occurring in relatively small quantities, and also to investigate their effect on the physical properties of commercially pure aluminum, some experiments on a series of alloys containing up to 5 per cent. each of iron and silicon have been conducted by the Engineering Division, Air Service, McCook Field, Dayton, Ohio. It is the purpose of this paper to illustrate, in considerable detail, some of the structures found in these alloys so as to aid in the more general recognition of the iron constituents.

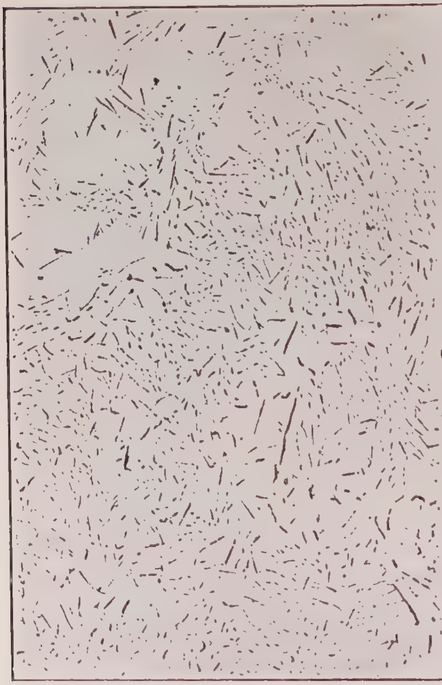

FIG. 9.—ALLOY, FE 2.0 PER CENT., SI 0.12 PER CENT.; ANNEALED; UNETCHED. 100 ×.

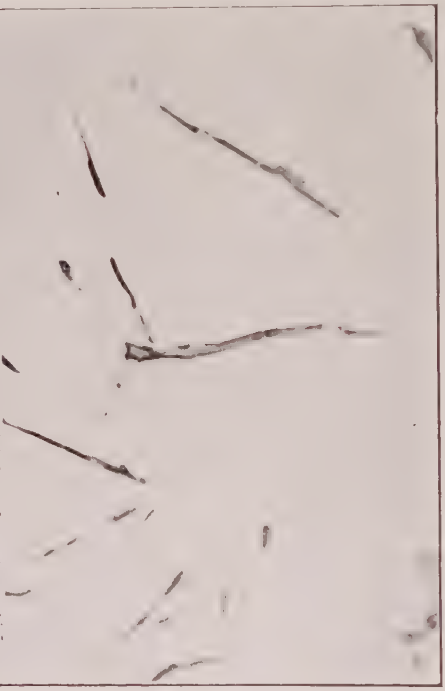

FIG. 10.—SAME SECTION AS FIG. 9; ETCHED 25 PER CENT. HNO_3 QUENCHED FOR 30 SEC. 1000 ×.

Two hardeners were made up containing approximately 5 per cent. of iron and silicon, respectively. These were analyzed and used for the preparation of all other melts, of which no analyses were made. Special care was exercised to see that no unintentional iron or silicon was introduced in the preparation of the alloys. The special-grade high-purity ingot, shown in Figs. 1 and 2, was used in making the hardeners and the remainder of the series; its analysis has been given. Wrought-iron horseshoe nails were used in the preparation of the iron hardener and commercially pure silicon (carbon 0.33 per cent., copper 0.02 per cent., iron 1.02 per cent.) for the silicon hardener. The analyses of these hardeners are:

	Iron, Per Cent.	Silicon, Per Cent.
Iron hardener.................	5.38	0.15
Silicon hardener..............	0.47	4.73

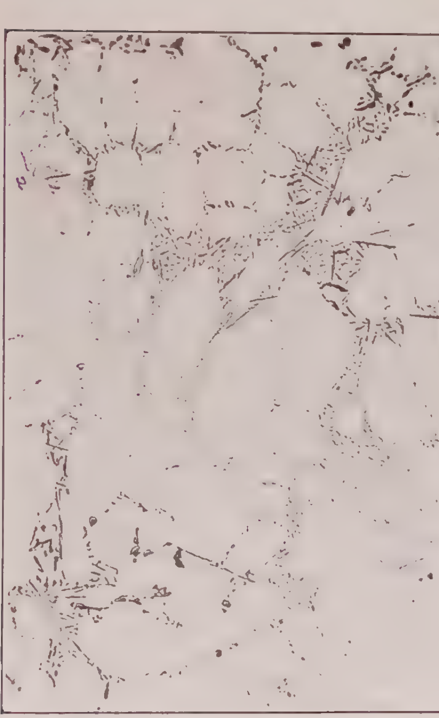

Fig. 11.—Alloy, Si 4.73 per cent., Cu 0.03 per cent., Fe 0.47 per cent.; as cast; unetched. 100 ×.

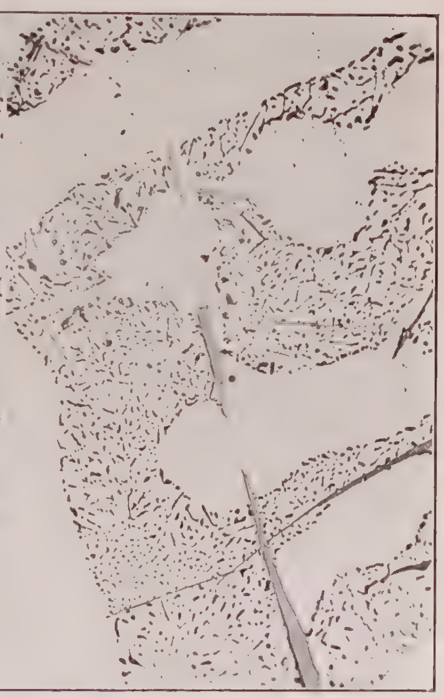

Fig. 12.—Same section as Fig. 11; unetched. 500 ×.

These analyses show that the amount of iron and silicon introduced unintentionally is practically nil.

Test bars were cast horizontally in green sand, three bars in a mold with common pouring head and risers (Air Service standard pattern TB1). Some of these bars were annealed for 24 hr. at 925° F., and then slowly cooled, through a period of one week, to 400° F.; from which they were allowed to cool down, with the furnace, to room temperature. Other bars were quenched in cold water after 24 hr. at 925° F.; these will be referred to as "annealed" and "quenched" specimens, respectively. Metallographic specimens were cut from the ¾-in. diameter ends of the "as cast," "annealed," and "quenched" bars.

Figs. 7 to 12 show the characteristic forms of $FeAl_3$. Fig. 7 is the structure of the "as cast" specimen of iron-aluminum hardener; large primary needles are shown distributed throughout a matrix consisting of the $FeAl_3$-aluminum eutectic. Fig. 8 shows the star formation of the primary crystals; the many cracks in the $FeAl_3$ particles show that it is extremely brittle. In Fig. 9 is shown again the curved-needle formation of the $FeAl_3$-aluminum eutectic in an alloy containing 2 per cent. iron. This alloy consists entirely of this eutectic formation and confirms previous data on the composition of the eutectic. There is 0.12 per cent. silicon present from the original ingot and even this small percentage was sufficient to form some particles of the X constituent. Fig. 10 shows two needles of $FeAl_3$ joined by a particle of X constituent. The X constituent is faint and shows boundaries, which are due to the action of the etching reagent.

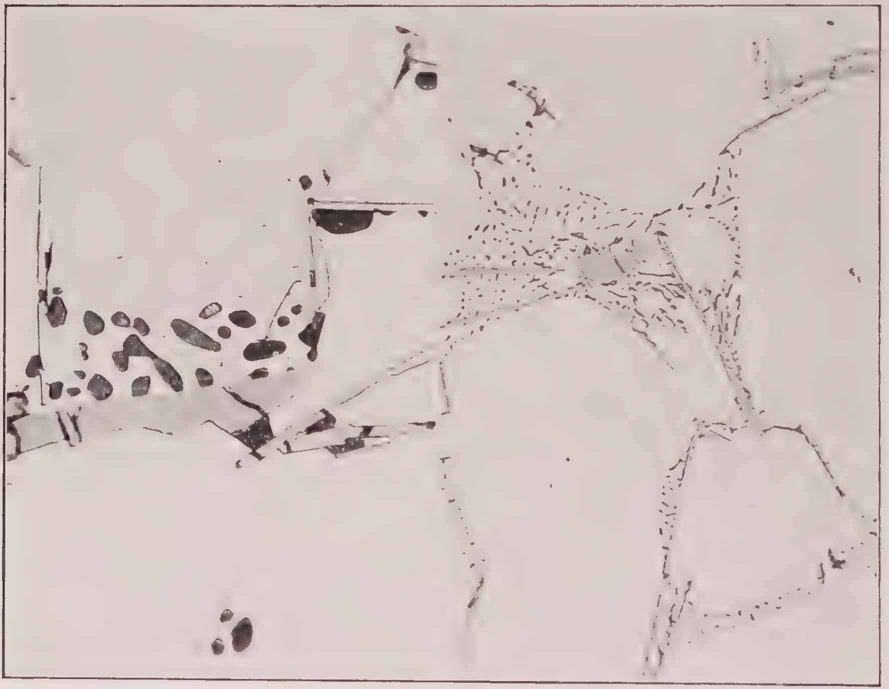

Fig. 13.—Alloy, Si 1 per cent. (Fe in original ingot 0.42 per cent.); etched 25 per cent. HNO_3 quenched for 30 sec. 1000 ×.

Fig. 14.—Alloy, Si 1.25 per cent. (Fe in original ingot 0.42 per cent.); as cast; unetched. 500 ×.

The structure of the "as cast" silicon-aluminum hardener is shown in Figs. 11 and 12; Fig. 11 shows the average structure at low magnification and Fig. 12 the fine silicon particles intermixed with fine and coarse needles of the constituent X. The formation of the constituent X is due to the presence of 0.47 per cent. iron in the original ingot; this is evidently the ternary eutectic of constituent *X*, silicon, and aluminum mentioned in the Eleventh Report.

Figs. 13 to 16 show these constituents in alloys containing different amounts of iron and silicon. Fig. 16 shows a rectangular blue-gray constituent (in half tone in the photomicrograph) in addition to the light-gray needles of the constituent *X* and the dark purple silicon particles; this has not been identified but has often been noted in the sand-cast silicon-aluminum alloys. It is interesting to note, in connection with the solubility of silicon in aluminum, that in the presence of 0.42 per cent. of iron as little as 0.50 per cent. of silicon shows a few particles of free silicon in the annealed specimens.

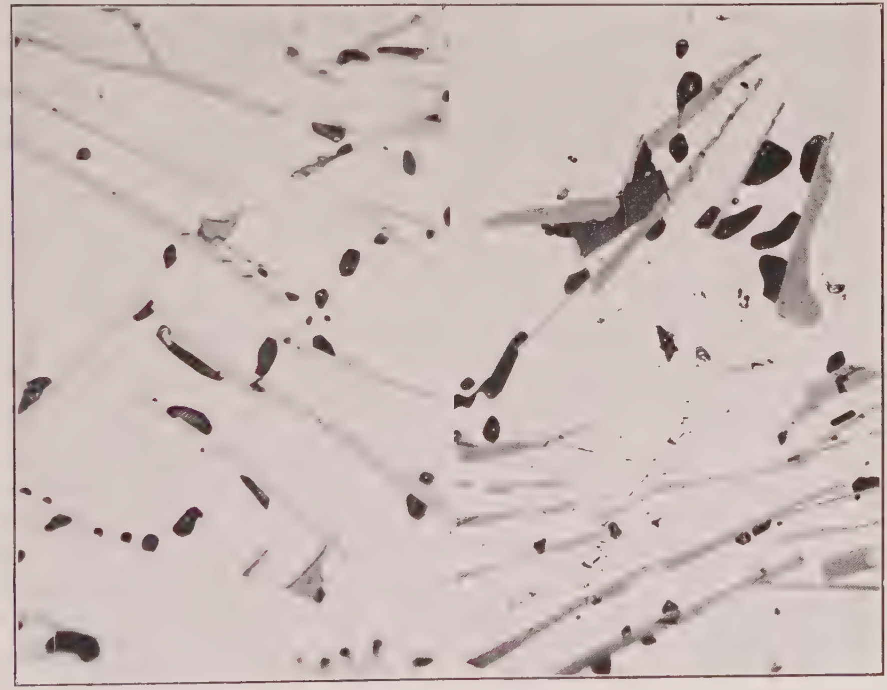

Fig. 15.—Fe 1.0 per cent., Si 2.0 per cent.; annealed; unetched. 1000 ×.

Fig. 16.—Same section as Fig. 15; unetched. 1000 ×.

Figs. 17 to 22 illustrate a structure, resembling "Chinese script," that is generally found where iron and silicon exist in nearly the same percentages. It is extremely difficult to determine in the unetched specimen, because of the fineness of this structure, whether this constituent consists of $FeAl_3$ or constituent *X*. Various etching reagents were tried, without success, until the nitric-acid quench suggested by Hanson[9] was prolonged from the 4 or 5 sec. necessary to etch the $CuAl_2$ to 30 sec. By this means it was found that the constituent X was attacked and turned a watery gray while the $FeAl_3$, although outlined, still showedthe characteristic purple tinge. The action of this etch is best shown in Fig. 10. Fig. 18 shows a "Chinese script" area attacked by the etch.

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[9] *Loc. cit.*

In the sand-cast test bars, there is usually a segregated area near the center of the specimen that sometimes also contains a small shrinkage pipe. It was noted in the alloys containing a nearly equal quantity of iron and silicon that this characteristic skeleton structure was usually segregated near the pipe and thus apparently had a low melting point. The alloy containing 0.63 per cent. each of iron and silicon was melted down and a small quantity of it allowed to freeze in an iron ladle with frequent agitation. When a very small amount of this liquid remained a small button was cast from it in green sand. On examination, this button was found to consist almost entirely of the "Chinese script" structure, a typical photomicrograph of this button is shown in Fig. 19. This also bears out the conclusion that the structure consists of constituent X, which is the low-melting-point constituent for this com-

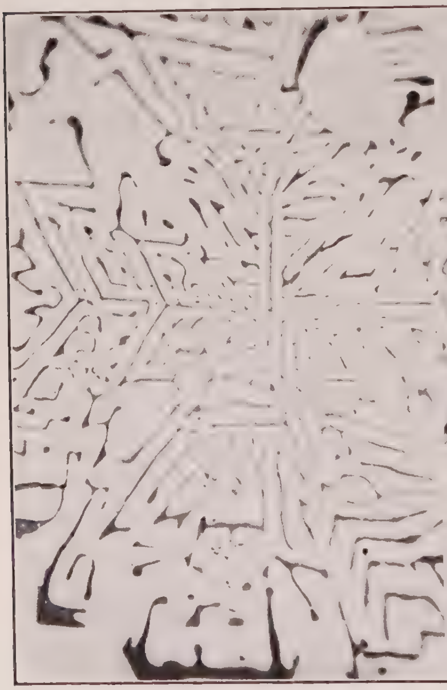

FIG. 17.—ALLOY, FE 0.63 PER CENT., SI 0.63 PER CENT., ANNEALED; UNETCHED. 1000 ×.

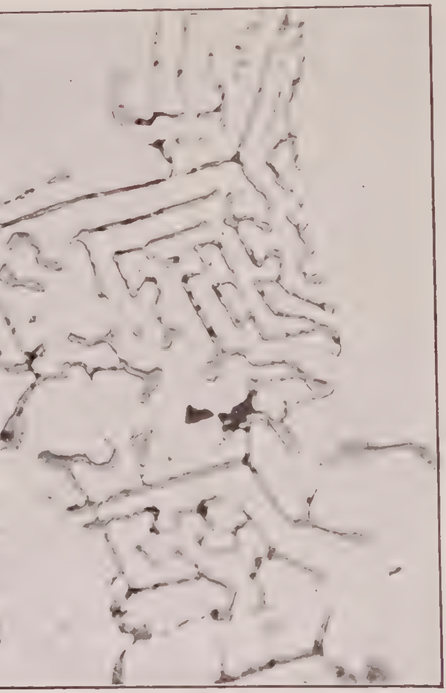

FIG. 18.—SAME SECTION AS FIG. 17; ETCHED 25 PER CENT. HNO_3 QUENCHED FOR 30 SEC. 1000 ×.

position. Fig. 20 shows the characteristic $FeAl_3$ eutectic formation with a small area of the skeleton structure.

It was at first assumed that all of the particles occurring in the curved needles, which are characteristic of the $FeAl_3$ eutectic, were $FeAl_3$; but it has since been shown, by the nitric-acid etch, that a number of these particles have been converted into the *X* constituent. With small percentages of silicon, they still remain in the characteristic formation of the $FeAl_3$ needles. However, Fig. 21 shows how the needles, after being converted to the constituent *X*, tend to spread out into the "Chinese script" formation. As the percentage of silicon is increased, this structure predominates until an excess of silicon is reached, when the ternary eutectic of constituent *X*, silicon, and aluminum is formed. Fig. 22 shows an area in which the "Chinese script" is closely associated with the ternary eutectic of constituent *X*, silicon, and aluminum.

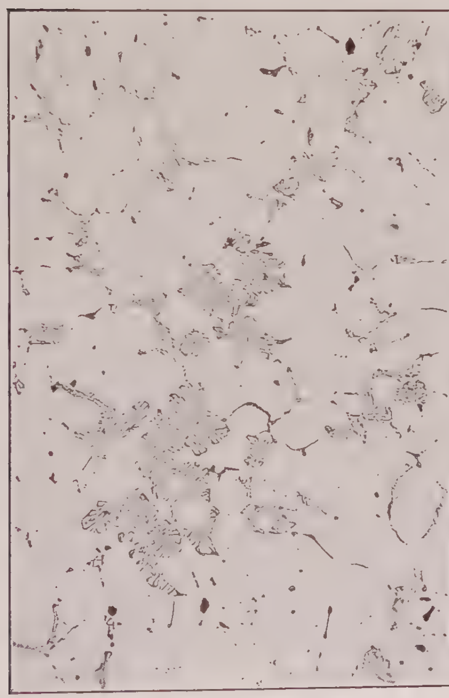

Fig. 19.—Alloy, low-melting-point constituent from alloy of Figs. 17 and 18; etched 10 per cent. NaOH for 5 sec. 100 ×.

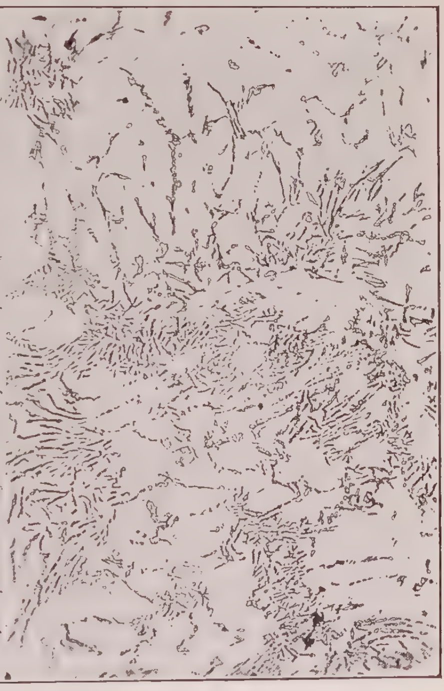

Fig. 20.—Same section as No. 17; etched 10 per cent. NaOH for 5 sec. 100 ×.

The fact that a constituent is formed, which is due to both iron and silicon, has given rise to several references in the literature to iron-silicides.

Jeffries[10] mentions in connection with silicon-aluminum alloys that "iron forms a silicide reported to be $FeSi_2$" and Edwards[11] also mentions iron silicide in connection with the silicon-aluminum alloys and, as has been previously stated, Wills mentions FeSi. Gulliver[12] states that the liquidus of the iron-silicon series shows one maximum and two minimums. The maximum at 33.6 per cent. silicon corresponds to the compound FeSi, and the minimums at 21.5 and 61 per cent. silicon, to eutectic mixtures of FeSi with Fe_2Si and with silicon, respectively. Fig. 23 shows the microstructure of an alloy containing $66\frac{2}{3}$ per cent. iron and $33\frac{1}{3}$ per cent. silicon, the proportions for the compound FeSi. This alloy consisted of the large crystals shown in the photomicrograph with a small amount of softer constituent in their interstices. This soft constituent, according to Gulliver, is Fe_2Si. An attempt was made to alloy this metal with aluminum but it was found difficult to get the iron silicide to

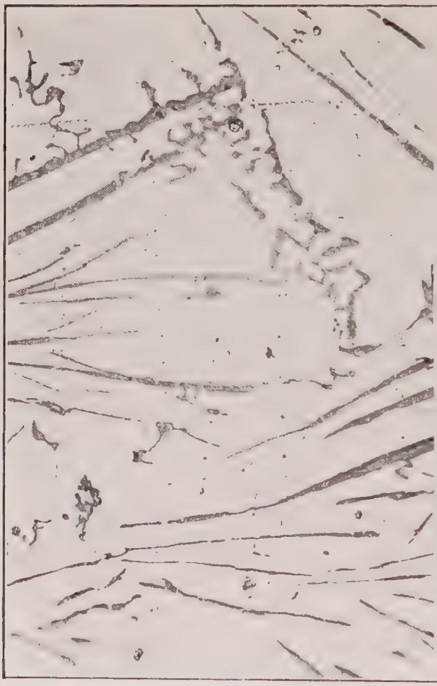

Fig. 21.—Same section as Fig. 17; unetched. 500 ×.

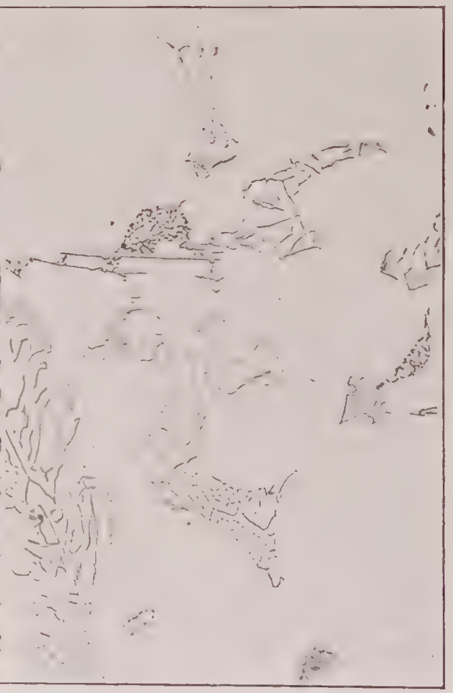

Fig. 22.—Same section as Fig. 19; etched 10 per cent. NaOH for 5 sec. 500 ×.

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[10] Zay Jeffries: Aluminum-Silicon Alloys. *Chem. & Met. Eng.*, (April, 26, 1922) **26,** 750.

[11] J. D. Edwards: Properties and Manufacture of Aluminum-Silicon Alloys. *Chem. & Met. Eng.* (Sept. 27, 1922) **27,** 654.

[12] Gulliver: "Metallic Alloys," 333.

dissolve, even at a temperature of 1800° F. After the melt was poured, some solid chunks of the original alloy were found in the bottom of the pot and one of these was polished; Fig. 24 shows the boundary between the FeSi and the aluminum which had alloyed with it; the dark crystals on the left are FeSi and the light constituent between them is Fe_2Si. Several other constituents are associated with the FeSi in minor quantities which cannot be identified. The second of the two zones to the right of the FeSi crystals in Fig. 24 joins a third zone, Fig. 25, around which there is a light fringe; the light area in Fig. 25 is the aluminum solution and the light fringe is apparently the constituent X. Some of the other phases of the three zones between the X constituent and FeSi are probably unstable and diffusion would take place on annealing, but the point to be made is that the X constituent is certainly not FeSi, and probably not any simple compound of iron and silicon, but rather a constituent containing a relatively small amount of silicon. The dark particles to the right of Fig. 25 are silicon set free from the FeSi. Fig. 26 shows the alloy that resulted from the addition of the FeSi to the aluminum. This analyzed as: 8.88 per cent. iron and 4.6 per cent. silicon. The needle shown is similar in form to that shown in the Eleventh Report, in which

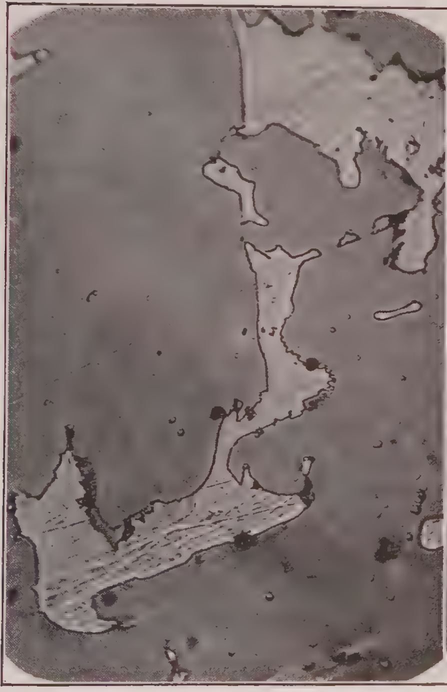

Fig. 23.—Alloy, Fe 66⅔ per cent., Si 33⅓ per cent., as cast; unetched. 500 ×.

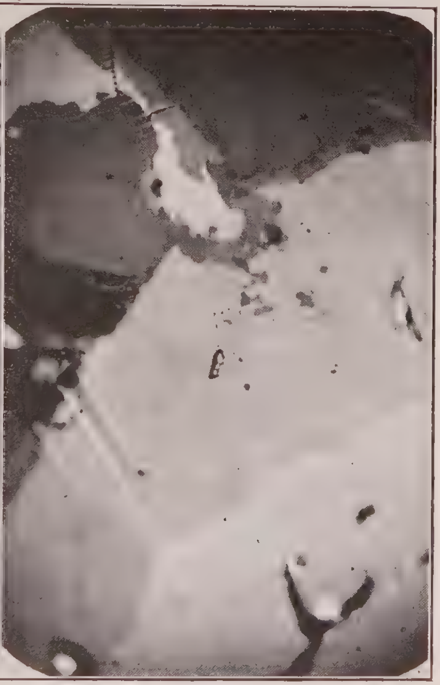

Fig. 24.—See description, p. 969; unetched. 1000 ×.

the dark core is said to be $FeAl_3$ and the boundary the X constituent. However, it will be noted that there are three constituents shown in this photomicrograph, one of which has been attacked by the etching reagent, which identified it as the same constituent found in the alloys containing small percentages of iron and silicon. According to the Eleventh Report, the dark core is $FeAl_3$; it may, therefore, be assumed that the other light gray constituent is unstable and would disappear on annealing.

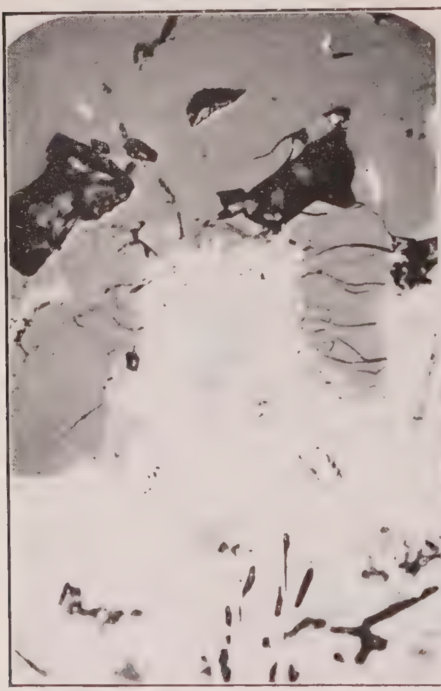
FIG. 25.—SAME SECTION AS FIG. 24; UNETCHED. 1000 ×.

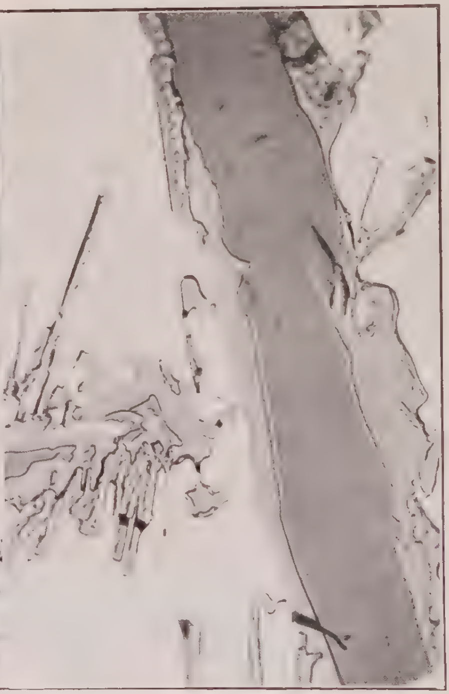
FIG. 26.—ALLOY, FE 8.88 PER CENT., SI 4.60 PER CENT. AS CAST; ETCHED 25 PER CENT. HNO_3 QUENCHED FOR 30 SEC. 1000 ×.

These last photomicrographs indicate the great complications that may be encountered in studying the metallographic structure of sand-cast aluminum alloys containing fairly high percentages of iron and silicon. In the commercial aluminum alloys, the iron-silicon relations are further complicated by the presence of other elements in solution and the formation of new compounds. It is felt, however, that a study of the structures illustrated in this paper will be of considerable aid in the identification of the iron constituents in other aluminum alloys.

Acknowledgment is made to J. H. Hester and Clifford McMahon for kind assistance in the polishing of specimens and photographic work.

DISCUSSION

Walter Rosenhain, London, Eng.—Any one who reads the Eleventh Report of the Alloys Research Committee of the Institute of Mechanical Engineers, will see that we did not feel satisfied that we had completely worked out that system. It is complicated and difficult to work out as the reactions are slow. $FeAl_3$ appears to react with the surrounding solution of silicon aluminum and a sheath surrounds the $FeAl_3$ needle. It takes a long time to get through that sheath and to get the reaction to penetrate to the core of the needles if they are as large as the author uses; but I would suggest chilling the $FeAl_3$ needle first of all thus shortening the time required to complete the reaction.

The existence of transition structures is probable. After all, $FeAl_3$ must be capable of dissolving aluminum and, similarly, the body X must be capable of dissolving iron and aluminum; therefore a zone of concentration is likely to be formed.

Here is a problem for the *x*-ray specialists; is it possible to determine the nature of a body like this, which it is difficult to isolate, and the composition of which it is difficult to determine? Is there any possibility of getting that out of a ray-spectrum?

E. H. Dix, Jr. (author's reply to discussion).—It is hard to reconcile Doctor Rosenhain's statement regarding the completeness of the work published in the Eleventh Report, presumably concerning the iron-silicon-aluminum alloys dealt with in the present paper, with the statement starting at the bottom of page 214 of the Report, which reads: "In that part of all sections representing alloys containing up to about 4 per cent. of silicon, the constitution is believed to have been satisfactorily determined, and is indicated by the different phase fields shown in the sections in Figs. 121 to 125." Ordinarily, one's opinion of the completeness of a work published in scientific literature would be expected to be based on the conclusion stated by the authors.

Although it is disappointing to have this doubt cast on the authenticity of diagrams on which the author based his paper, it is fortunate to have this criticism come at this time. The present paper must, therefore, be considered as a presentation of certain observations that cannot be properly interpreted until the constitutional relations are more completely worked out.

Regarding the use of chilling to give a finer structure than was obtained in sand-cast specimens used in this investigation, it must be remembered that the present paper was an attempt to apply the theoretical results given in the Eleventh Report to everyday sand-foundry practice.

A Study of Bearing Metals

By Christopher H. Bierbaum,* Buffalo, N. Y.

(New York Meeting, February, 1923)

The first significant fact observed in the study of bearing metals is that not a single pure homogeneous metal has given satisfactory service; all bearing metals are alloys made up of two or more phases; that is, they consist of hard and of relatively soft microscopic particles intimately mixed. The function of the hard particles, or *bearing crystals*, is to support the load and resist the wear when actual metallic contact exists between the bearing surfaces. The functions of the softer particles are to allow the harder particles to adjust themselves to the surface requirements of the journal and to wear down slightly below the surface of the harder, forming slight depressions on the apparently smooth wearing surface of the bearing, in which some of the lubricant is held when the bearing surfaces are brought into contact with each other. It is this characteristic of certain alloys to form these slight depressions, and thus provide the means for retaining a lubricant, that characterizes them as true bearing metals; in fact, a bearing metal has been defined as "an alloy that is capable of retaining a lubricant upon a bearing surface."[1] The extent to which the lubricant can be so held determines the most valuable characteristic of an alloy as a bearing metal.

An alloy which, perhaps, most characteristically represents a bearing metal is the composition of copper and tin that contains sufficient tin to produce the tin-copper eutectoid, or delta crystal. The depressions worn upon the surfaces of the softer crystals of the teeth of a motor-truck worm wheel, made of an 11-per cent. tin bronze, in many cases were from 5 to 6 microns in depth; all other conditions remaining constant, this depth of wear increases with an increase of working pressure. This copper-tin bronze contains intercrystalline shrinkage cavities, which also tend to retain the lubricant, and the relatively low fusing temperature of the delta crystal permits the same, under severe treatment, to fuse upon the bearing surface while it is being "run in." It is the dissimilarity of the component crystals, both in physical properties and in chemical composition, that makes bronze a bearing metal, and it is the similarity in physical properties and chemical composition of the component crystals that makes brass unsuited for this purpose.

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* Vice-president, Lumen Bearing Co.

[1] Report of Sub-committee on Bearing Metals, A. S. M. E. (1919).

Metals, as cast in the foundry for bearing purposes, are not in a state of equilibrium; they, therefore, must be studied in the condition in which they exist under service conditions. In fact, the value of controlling the chill effect on alloys, thereby rendering them especially serviceable for bearing purposes, is being appreciated more and more. In the copper-tin bronzes, where the delta crystal is desired, the chilling must be done with care; excessive chilling reduces the amount of this eutectoid formed, while cooling too slowly produces an undesirable coarse structure. Some other alloys cannot be chilled too suddenly, especially those in which it is important to reduce the grain size to a minimum. A bearing surface of bronze cast on a carbon chill, having grains of microcospic size, showed that, because the grains presented different angles of orientation, they wore at different rates, producing slight unevenness on the bearing surfaces, for which reason the grains, in a limited manner, caused a functioning of this bronze as a bearing metal.

So far ,all efforts to make accelerated wearing tests on bearing metals have failed because these tests are not made under service conditions, while tests made under service conditions are so prolonged that they lose their value. In almost all laboratory tests the softer, or more readily conforming, alloys are favored as they are "run in" in a shorter time. The test of a readily conforming bearing metal upon the hardened-steel arbor of a testing machine is meaningless, when considering practical requirements; a bearing alloy should be tested against the same metal with which it is to be used in service.

The only physical tests on a bearing alloy, in the aggregate state, that can have any possible practical value are those for determining its physical properties and its safety factor for withstanding the treatment to which it is to be subjected under service conditions and temperature; all other conditions relating to bearing metals are of a microscopic nature and depend on the physical properties of the individual crystals constituting this metal.

If a journal could be fitted so as to be absolutely smooth, true, and cylindrical, and additional polishing in service would be neither possible nor necessary, the journal could be made of a material having infinite hardness; but a journal of infinite hardness, not having a theoretically accurate surface, could not possibly give satisfactory wear with any bearing metal. In service, the bearing surfaces must polish each other;

therefore, the hardest particles in one member must bear a definite relation to the particles in the other; to insure best results, they must be of the same order of hardness. For a long bearing life, there should be a polishing effect between the two metals rather than an abrading tendency.

When studying the abrasion or polishing effected in a new bearing, we find that the worn-off particles of metal, carried away by the oil, gradually decrease in size. In a new bearing, made of proper material and correct design, though poorly fitted up, the first particles worn off may be large enough to be seen with the naked eye; but as the operation continues the particles become microscopic and then ultra-microscopic in size. The largest abraded particles resemble ordinary filings, but as they decrease in size they become spherical in form. This statement is made with due consideration of the fact that for a proportionate decrease in size of particles, the resolving power of the microscope also decreases. In the coarser abrasions, we may find entire crystals, which may be considered as only incidental in reducing the bearing to proper working conditions. In the operation of a long-life bearing, giving satisfactory service, the wearing is of a microscopic or ultra-microscopic order and represents the abrasion of correspondingly small particles of one member upon those of the other.

From the foregoing, it seems obvious that a study of bearings, without taking into consideration both members, could not be conducive of practical results. It is also obvious that a knowledge of the physical properties, especially of the abrasion hardness of the different microscopic constituent elements of a bearing alloy, as well as those of the journal, are imperative.

A natural suggestion for testing on so small a scale was to reduce present methods to a corresponding degree of refinement; this at the outset proved impractical. In fact, all methods for testing any one crystal at a time failed, if for no other reason than its difficulty of application. A fact sometimes observed upon a metallographic specimen, polished on soft broadcloth, is that a continuous sharp cut, crossing several crystals, is wider on some than on others; this fact suggested the development of the instrument herein described.

The following fundamental requirements were established, after somewhat prolonged experimentation: (1) The cutting point must be ground with an extreme precision and to definite specifications; its shape must be such that the vertical pressure exerted upon the surface tested will be greater at all times than the horizontal pull applied to the point. (2) The cutting point must be so mounted that it can respond to required successive elevations with the least amount of inertia to overcome, and it must exert a constant pressure while passing over the test surface. (3) The cutting point must be moved over the test surface at a uniform rate, slowly enough so that no additional indentation may occur by stopping on the softest crystal.

Various materials were experimented with when seeking a suitable cutting point: one that would be hard enough for all ordinary metals and their alloys, if possible including hardened tool steel; one having the necessary hardness and the microscopic homogeneity to permit it to be ground to an extremely accurate point. On account of its hardness, the diamond was the first material suggested, but its grinding has proved unsuccessful. Various other jewels have been tried, the sapphire, tourmaline, ruby, garnet, also various grades of special steels, and tungsten. The artificial leuco-sapphire (fused Al_2O_3) has proved extremely satisfactory, in that it has the necessary homogeneity, and is much harder than the crystals in the ordinary metals, including martensite in hardened steel, and tin oxide (fused SnO_2). The latter is the hardest crystal so far encountered in any of the alloys or their impurities; its hardness is more than twice that of martensite.

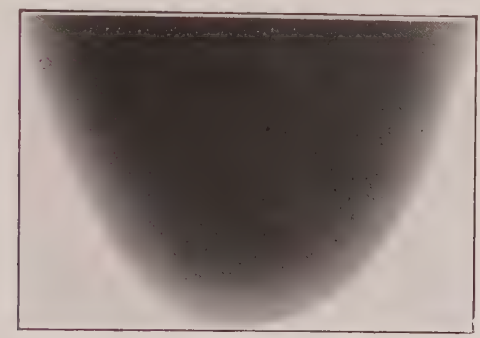


FIG. 1.—SHADOWGRAPH SHOWING POINT OF FINEST CAMBRIC NEEDLE, SHARP'S NO. 12. × 2000.

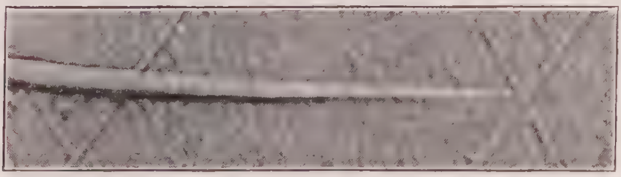


FIG. 2.—PHOTOMICROGRAPH OF SOFT-STEEL SURFACE SHOWING END OF MICROCUT × 2000.

The importance of a knowledge of the hardness of the individual crystals of the alloys was appreciated by the early investigators when applying the microscope to the study of metals. Their investigations, however, were crude, in that they applied the points of steel needles in their work. The crudeness of any work done with a needle is evident from the shadowgraph, of 2000 magnifications, of the point of the finest cambric needle, Sharp's No. 12, given in Fig. 1; in Fig. 2 is shown the end of a microcut made upon soft steel with a sapphire point ground to definite specifications. The point itself appears sharp under 2000 magnifications.

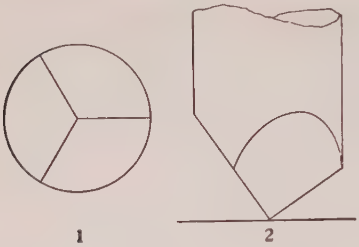


FIG. 3.—DETAILS OF JEWEL POINT.

After the selection of a suitable jewel, considerable difficulty was experienced in grinding a point that would be sharp and accurate to the extreme limits of the microscope. The grinding of a conical point was

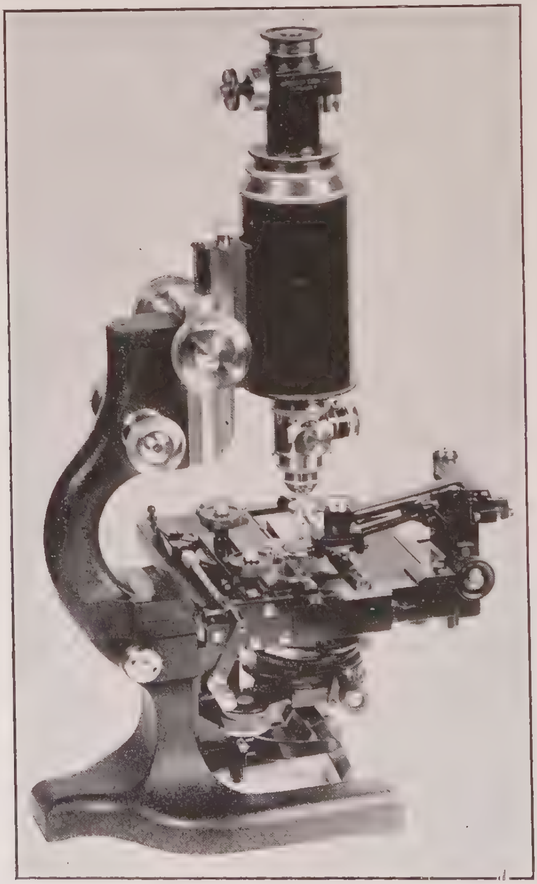


FIG. 4.—MICROSCOPE COMPLETELY EQUIPPED WITH MICROCHARACTER AND MICROMETER EYEPIECE.

early abandoned as being impossible; the point that was finally ground successfully was the corner of a cube or a solid right angle. Fig. 3 gives the details of this point showing (1) an end view and (2) a side view. The jewel was cylindrical in form 0.022 in. in diameter and 0.08 in. in length. It was cemented securely into a close-fitting hole in the end of the grinding stylus that made an angle of 54.73° with the grinding disk. Three successive facets were ground upon the end of the jewel after successively rotating it through 120°, thus making the point the corner of a cube. The jewel point is mounted so that the diagonal of the cube shall be normal to the surface to be tested and with one edge advancing directly in line of motion. After grinding, the jewel was placed in a jig and, after all adjustments for accuracy had been made, cemented into the end of a flexible spring, while both the jewel and spring were held in the jig in the field of the microscope; the cement was a heavy solution of specially refined shellac. After the cement was thoroughly hardened, the excessive length of the jewel was ground off to reduce it to a minimum weight.

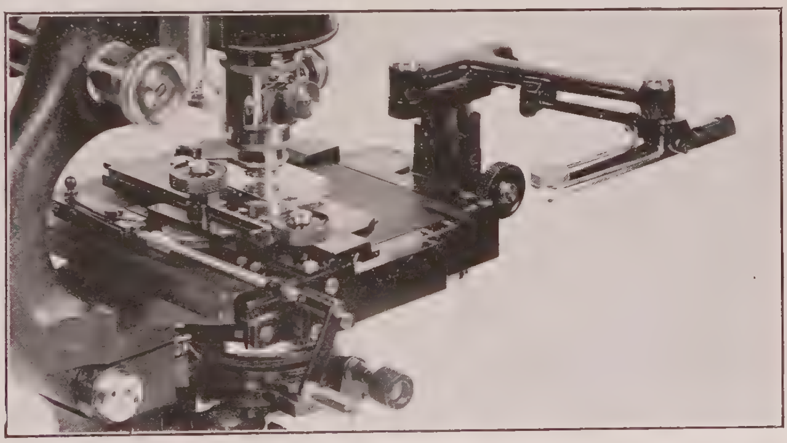


Fig. 5.—Microcharacter with jewel suspension bracket swung out.

Fig. 4 shows a microscope complete and ready for work, equipped with the instrument developed for research on bearing metals; the name chosen for the instrument is Microcharacter. The instrument may be described as a mechanical stage with two additions, the jewel bracket at the right and the micrometer feed at the left. The jewel bracket, as shown in Fig. 5, is provided with a vertical rack for raising and lowering the bracket that carries the jewel suspension. This jewel suspension consists of a balanced arm, pivoted on sapphire bearings, one end of which is provided with a spirit level and the other with a weight, which constitutes the load for the jewel point. The jewel is mounted in the small end of a tapered steel spring, the other end of which is fastened to the lower side of the suspension arm. The suspension bracket is also provided with two adjusting devices, by which the microcut can be located so as to appear in the field of vision. These devices are not absolutely necessary, but are desirable, in that they make possible the testing of a crystal that may be selected in the field of the microscope. A worm and wormwheel attached to the longitudinal movement of the mechanical stage constitutes the micrometer feed.

The specimen to be studied should not be more than ¼ in. thick, and from 1 to 2 sq. in. in area; it should be placed upon an ordinary microscopic slide, leveled, and held in plasticine. After the specimen is in place, the jewel point is brought down upon it and the jewel suspension, by aid of the spirit level, is brought into a level position; this insures that the full load of the weight is being exerted upon the jewel point. The specimen is then moved along under the jewel point by the micrometer worm and wheel actuated by turning the small handle. A drop of oil should be placed upon the test surface and in contact with the jewel before starting, for a somewhat smoother microcut is then made. After the microcut has been made, the jewel bracket is elevated, by turning the knurled head, and swung out of position, as shown in Fig. 5. The objective is brought into focus upon the microcut and, after it has been studied through a plain ocular, the micrometer eyepiece is substituted and the widths of the microcut are measured.

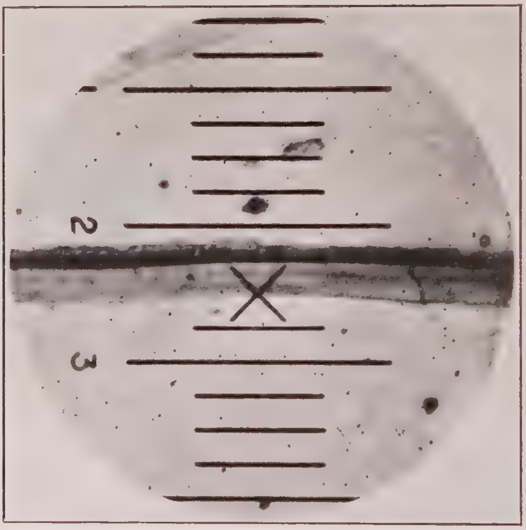
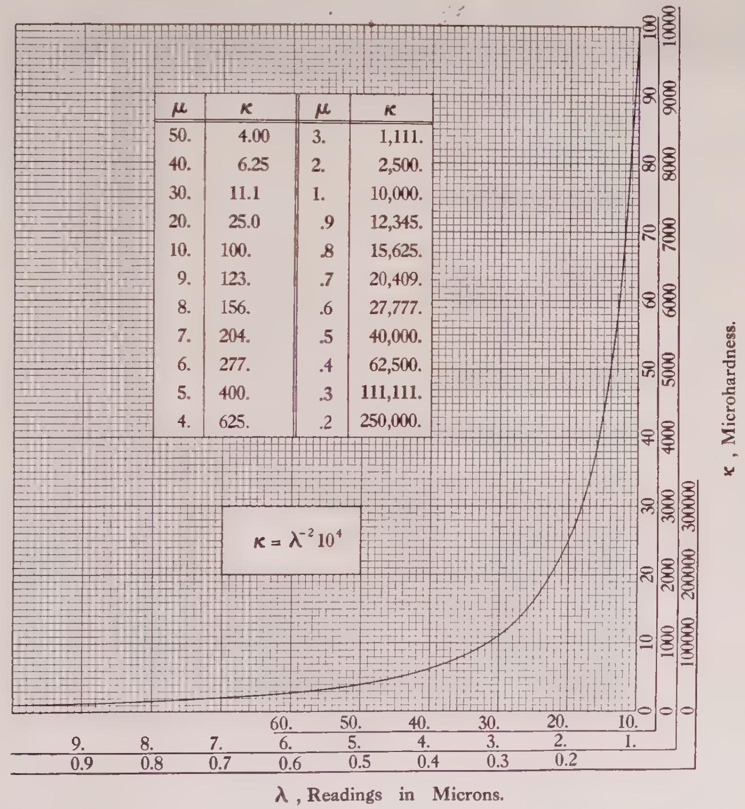


FIG. 6.—MICROCUT THROUGH MICROMETER EYEPIECE.

Fig. 6 shows a view of a microcut through the micrometer eyepiece; the focusing, of course, should always be done upon the original surface of the specimen. The X of the scale is moved to one edge, in which position a reading of the vernier is taken; after moving the X of the scale to the other edge the second reading is taken, both edges should be approached in the same direction. The difference between the vernier readings gives the fractional parts, whereas the full units can be read directly. It is necessary to calibrate the scale of the micrometer eyepiece by comparing it with the scale of a stage micrometer, for which the working set-up of the microscope should be used. The micrometer eyepiece scale is then calculated, in terms of microns, and the reading of the width of the microcut multiplied by this factor reduces the reading to terms of microns. A 2-mm. apochromatic objective with 160-mm. tube length and a 20× micrometer eyepiece with a ¼-mm. scale, gives substantially 2.5 microns per unit of scale division. A convenient weight upon the jewel point seems to be 3 gm., as with this load readings can be made upon the softest lead as well as upon the hardest steel without any change in the microscope set-up.



μ	κ	μ	κ
50.	4.00	3.	1,111.
40.	6.25	2.	2,500.
30.	11.1	1.	10,000.
20.	25.0	.9	12,345.
10.	100.	.8	15,625.
9.	123.	.7	20,409.
8.	156.	.6	27,777.
7.	204.	.5	40,000.
6.	277.	.4	62,500.
5.	400.	.3	111,111.
4.	625.	.2	250,000.

$$\kappa = \lambda^{-2} 10^4$$

FIG. 7.—FORMULA, TABLE, AND DIAGRAM OF SCALE OF MICROHARDNESS.

The micron, 0.001 mm., seems to be a convenient unit for our purpose. As the various cross-sections of microcuts in the different metals and alloys are symmetrical and their areas are proportional to the squares of their respective widths, these squares should be used for comparison.

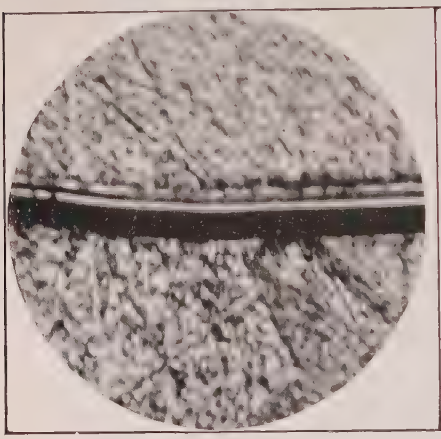


FIG 8.

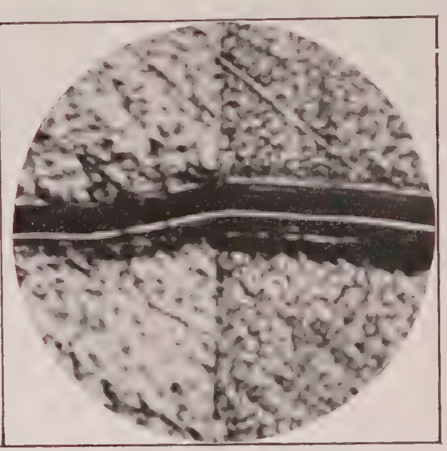


FIG. 9.

FIG. 8.—MICROCUT IN TWO ADJACENT GRAINS ON PURE COPPER HAVING SAME APPARENT HARDNESS. × 1000.

FIG. 9.—MICROCUT ON SAME CRYSTALS AS IN FIG. 8 SHOWING MAXIMUM AND MINIMUM MICROHARDNESS OF CHEMICALLY PURE COPPER. × 1000.

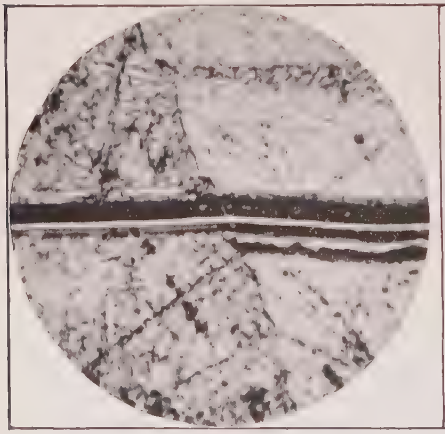


FIG. 10.

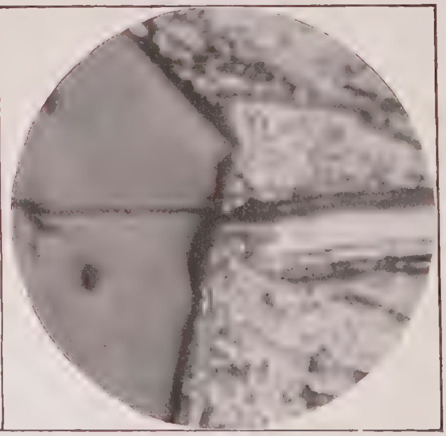


FIG. 11.

FIG. 10.—MICROCUT SHOWING GREATEST RANGE OF MICROHARDNESS OF CHEMICALLY PURE ZINC. × 1000.

FIG. 11.—MICROCUT UPON FUSED TIN OXIDE AND BRONZE. × 1000.

The same should appear as a factor in the mathematical expression for the value which may be designated as microhardness. For the reason that the greater widths represent the softer material the reciprocal of these squares should be used. Let κ represent the microhardness and λ the width of microcuts; then after multiplying by a convenient constant, to avoid unwieldy numbers, $\kappa = \lambda^{-2}10^4$. In Fig. 7 this formula has been plotted to two rectangular coördinates covering an ample range of microhardness.

The microhardness of grains in most metals varies in different directions; that is, if a grain on a polished metallographic specimen is cut in different directions the results vary within certain limits, and these limits vary for different metals. This property is illustrated by Figs. 8, 9 and 10. Fig. 8 shows a uniform width of microcut upon a surface of chemically pure copper. The two grains, of necessity, have a different angle of crystalline orientation but they presented approximately the same angle to the surface and to the direction of the microcut, so the two crystals present an equal degree of microhardness. Fig. 9 shows the same two grains as in Fig. 8, but the microcut is in a different direction, with the result that maximum and minimum microhardness are shown.

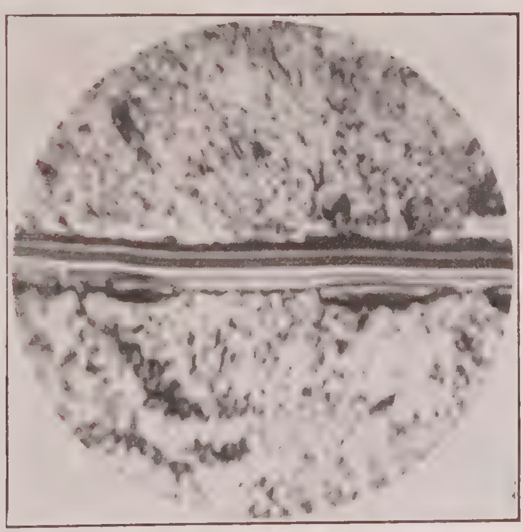


Fig. 12.—Microcut upon slightly etched hardened tool steel. × 2000.

Fig. 10 shows the greatest difference of microhardness, so far encountered, for chemically pure metal; it shows a microcut of maximum and minimum widths for electrolytic zinc.

Fig. 11 shows a crystal of fused tin oxide embedded in bronze. The narrow width of microcut indicates the extreme hardness of the crystal, which is found only in bronze that has been heated above the fusing temperature of tin oxide. It has long been known that "burned" bronze was destructive to bearings. The actual hardness of the crystal, however, does not seem to have been determined before this as being more than twice that of hardened tool steel; it is by far the hardest crystal found in any of the common alloys or their impurities.

Fig. 12 shows a microcut in hardened tool steel; the martensite formation is distinctly seen.

Heat tinting or very light etching may be used for the identification of crystals; the etching, however, should always be very light in order not to affect the results. This method of testing offers strictly comparable results, for a single microcut upon a substance of, say, unit microhardness would disturb as large a volume of material as 100 microcuts on a substance having a microhardness of 100. Should conditions require the hardness of the diamond, there is no doubt that it can be ground to this same degree of refinement and results obtained uniform with the present. While it is desirable that the instrument fit the microscope stage it is not absolutely necessary that it should; a plain stand may be improvised, and after the microcut is made the specimen can be transferred to the microscope where all measurements can be made in the usual way.

By the use of this instrument, it has been clearly shown that the addition of zinc, in excess of 2 per cent., to a copper-tin bearing bronze increases the hardness of those crystals which, without the zinc, would have been the softest. This addition of zinc, therefore, tends to reduce the range of hardness between the hardest and softest crystals and theoretically reduces the bearing value of such an alloy. This offers a most striking corroboration between theory and practice, for it has long been known that the hardening of bearing bronzes by the addition of zinc increases the rate at which they wear off in service, and that this destructive wearing effect increases with an increase of zinc content.

Name of Substance	Width of Microcut, Average	Micro-hardness, Average
Lead, c.p., Picher	37.7	7.03
Tin, c.p., electrolytic	29.3	11.7
Copper, c.p., Raritan	11.8	78.4
Antimony, commercial	9.1	121.0
SnSb crystal as found in babbitt	6.9	208.0
Nickel, c.p., International Nickel Co	6.4	244.0
PCu_3 crystal as found in phosphor bronze	6.1	267.0
Iron, Swedish, softest crystal	4.8	420.0
Cobalt, c.p., International Nickel Co	4.0	625.0
$SnCu_3$, or delta crystal in copper-tin bronze	3.6	750.0
$SnCu_4$, hardest crystal in high-Cu hardened babbitt	3.2	1006.0
Hardened steel—Johansson test blocks	2.2	2229.0
SnO_2 (fused) as found in burned bronze	1.4	5390.0

An equally interesting fact was brought out in the study of the distinctive effect that a certain high-percentage copper babbitt had on a low-carbon journal; the results showed that the $SnCu_4$ crystal was much harder than any crystal in the steel, which was unexpected and accounted for the result. While this instrument was developed for one particular purpose, it should prove of greater interest as it offers means for investigation and research that have not been available heretofore.

DISCUSSION

H. S. Rawdon, Washington, D. C. (written discussion).—The early investigations of hardness were confined almost exclusively to the study of minerals.

Seebeck (1833) and Franz (1850)[2] improved Moh's method by the introduction of a sharp point, usually steel, bearing a load that could be varied, the hardness being expressed in terms of the load necessary to produce a scratch just perceptible to the eye.

Grailich and Pekárek[3] extended and continued this work and have published results obtained by similar methods. The use of a lens as an aid in detecting the visibility of the scratches produced by a loaded needle has been credited to Exner.[4] He confined his work along this line almost entirely to the study of the variation in the width of the scratches within the face of a single crystal according to the direction of the scratch.

To Turner[5] belongs the credit of extending this method of hardness testing to the study of metals and alloys, a diamond point being used for the purpose. Martens[6] improved Turner's method and developed his well-known sclerometer. This utilized a 90° conical diamond point at one end of a weighted lever arm, under which the polished face of the specimen was moved by means of a traversing table. Martens pointed out, however, the imperfections and limitations of the diamond for this purpose. The hardness number used was the load, in grams, necessary to produce a cut having an average width of 10μ, that is 0.01 mm., and the term 'scoring hardness' was used to designate hardness determined in this way.

The Martens sclerometer was improved considerably by Pöschl[7] and adapted to the stage of the microscope. The Martens method, however, has remained essentially the basis of all investigations of this kind up to the present. The further marked improvements and refinements described and illustrated by the author are apparent to all and need no further comment.

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[2] R. Franz: Uber die Härte der Mineralien. Pogg. *Ann.*, (1850) **80,** 37.

[3] Gralich and Pekárek: Sitzungeber. d. Wien Akad., Naturw. Math. (1854) kl.

[4] Exner: Untersuchengen uber die Härte an Kristallflachen, Wien. (1873).

[5] Thomas Turner: The Hardness of Metals, *Proc.* Birmingham Philosophical Society (1886) pt. 5.

[6] A. Martens: Mitt. k. techn. Versuchs-Anst. (1890), **8,** 236.

[7] V. Pöschl: Die Harte der festen Korper, Dresden (1909)

It might be inferred from the author's statements concerning the use of steel points that any attempt to study variations in hardness by such means is useless; the experience of the Bureau of Standards in this respect, however, is to the contrary. The study of the erosion of steel rifle barrels may be cited as an illustration. This investigation was carried out at a time when results were needed with the least delay possible, so that but little time could be devoted to the apparatus. It was found that an ordinary safety-razor blade, mounted vertically on a horizontal axis and weighted at one end from above, formed a useful device for demonstrating the localized hardened zone which was produced along the bore of the rifle in use. The specimen was moved under the corner of the weighted blade by means of a traveling microscope stage. The photomicrographs in Fig. 13 show the character of the cuts made in this way and how definitely they reveal variations in hardness in the structural constituents of a material. The alloy used was one of tin and antimony containing the characteristic cubic crystals of the compound SbSn; the razor blade was loaded with a 3-gm. weight in making the test. A cut made by means of the microcharacter has been included for purposes of comparison. The writer does not wish to minimize in any way the usefulness of the latter instrument, but does wish to emphasize the useful results in this field that often may be obtained by very simple means. Such results are, in general, largely qualitative, although for any

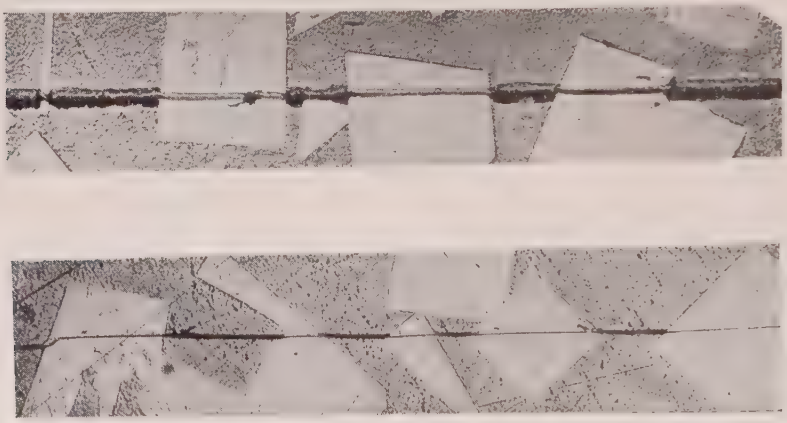

Fig. 13.—Antimony-tin alloy showing difference in hardness of microconstituents as revealed by variations in width of cut made by a sharp-pointed instrument. × 100.

Top cut made by microcharacter; bottom cut made by the corner of a safety-razor blade.

particular case, as in the one illustrated, the relative hardness of the different constituents can be expressed numerically if desired, preferably in terms of width of cut for a given load.

Anisotropy, or variation of properties with direction, is a fundamental characteristic of the crystalline state. Most investigators have considered that hardness was one of these properties, but as a result of a rather extensive investigation of the hardness of mineral crystals, Pöschl[8] concluded that this variation in hardness according to direction within a crystal is more apparent than real, basing his conclusion upon the fact that other investigators had considered only the width of the cut and had neglected entirely the distorted zone adjacent to the cut on each side. These two effects represent the work done by the cutting point and should be considered together. It is interesting to note that the author's results support the general view held concerning the variations in hardness. In most metal crystals, the distorted zone is very narrow, and variations in the width of this zone of strained metal of a magnitude sufficient to account for the pronounced variations of the width of the cut are not to be found.

E. H. Dix, Jr., Dayton, Ohio (written discussion).—Can the variation in hardness of a cored solid solution in, say, the ordinary 90-10 copper-tin bearing bronze, be measured by the microcharacter?

Most laboratories working on the metallography of metallic alloys are equipped with the inverted type of metallurgical microscope; as the microcharacter cannot be applied to this type of microscope, it seems desirable to mount it on a separate base. The specimen is then mounted on a microscope slide and the cut made as described by the author. Several parallel cuts can be made and the specimen examined to determine what constituents have been in the path of the cuts. If the instrument is operated in this manner, the two adjustments on the jewel suspension bracket are unnecessary. The author has stressed the accuracy with which the jewel point has to be set, but it is just as necessary that the motion of the stage in making the cut be absolutely parallel to the axis of the spring bar holding the jewel. At best, it is only possible to obtain a rough approximation by means of the two adjustments on the jewel suspension bracket. If, however, this bracket were made rigid, with a ground fit so that when swung in place the axis of the jewel suspension arm would be exactly parallel to the motion of the stage, much greater accuracy would be possible.

The reason for the two adjustments on the jewel suspension is to enable the cut to be made through any portion of the field as viewed through the microscope, but I have not been able to do this successfully. Certainly where only the inverted-type microscope is to be had it would be much more convenient to have the jewel suspension bracket rigid.

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[8] *Loc. cit.*

In the operation of the instrument, the beginner has great difficulty in obtaining the proper focus in order to get the exact width of the cut, as a slight variation in focus makes a relatively large error in this measurement. In using this instrument on aluminum alloys, as the path of the cut is intercepted by one of the hard constituents present in the aluminum matrix the jewel point attempts to dodge around the hard particle; thus the determination of the hardness of the different constituents encountered in the aluminum alloys is difficult. This difficulty, however, will doubtless be overcome by making a sufficient number of cuts and the microcharacter may prove of great value in determining the relative hardness of the various constituents and may aid in distinguishing between the several iron constituents which are known to be present in all aluminum alloys. This particular problem is being investigated by Z. G. Deutsch at the University of Cincinnati on specimens furnished by the Engineering Division of the Air Service.

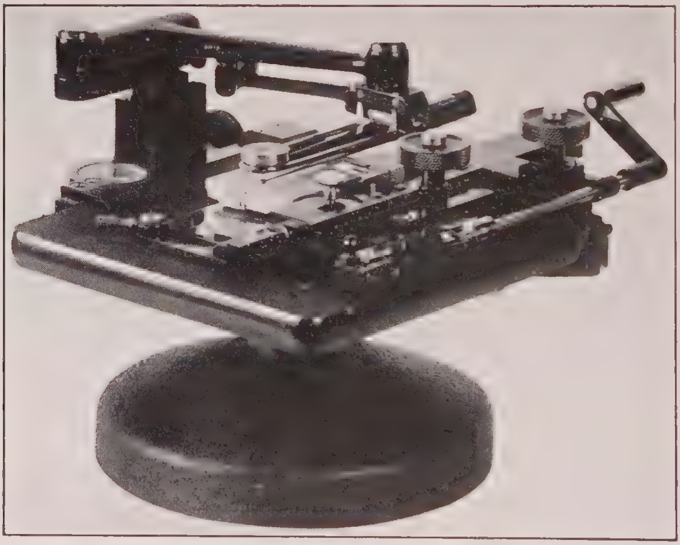

Fig. 14.—Mount for microcharacter when microcut need not pass through optical axis of microscope.

William Campbell, New York, N. Y. (written discussion).—The description of, and the results obtained by the microcharacter are of great interest and the figures obtained ought to be more applicable to bearing metals than those of the Brinell hardness machine or the sclerometer or the Shore scleroscope. However, one or two statements in the paper do not conform to the modern ideas of crystal structure and constitution; for instance, "the tin-copper eutectoid, or delta crystal." There are two copper-tin eutectoids and probably three. In this paper, the first is evidently the one referred to. This consists of a mechanical mixture of the soft copper-tin solid solution and of a hard solution called delta, which, by some, is considered to be the compound Cu_4Sn. In bearing bronzes, the delta does not occur as crystals but tends to form a network, or matrix, to the dendrites of the alpha solid solution.

In the same paragraph it is stated: "the relatively low fusing temperature of the delta crystal permits the same, under severe treatment, to fuse upon the bearing surface while it is being run in." The delta crystal has no definite melting point, even in the pure state, because before melting it transforms into a solid solution. In bronzes, the eutectoid containing particles of delta, on heating above 500° C., changes over to the solid-solution beta. On heating the mixture of alpha and beta up to the solidus, or fusing point, the beta tends to break down into alpha and a liquid richer in tin than the original beta.

In the table, the hardness is given of two compounds: Sn_3Cu, or delta crystal in copper-tin bronze, and Sn_4Cu, the hardest crystal in high-copper hardened babbitt. Sn_3Cu does not occur in copper-tin bronze; the delta, if a compound, is probably Sn_4Cu. In babbitt metal hardened with copper, two copper-tin compounds or solid solutions are encountered in the case of the tin-base babbitts: up to about 7.5 per cent. copper, the hard constituent epsilon, sometimes thought to be CuSn solid solution, occurs as skeleton crystals often in starlike form; above 8 per cent. copper, the compound Sn_3Cu crystallizes out first of all in the form of plates and is surrounded by an envelope of epsilon. The Cu_3Sn is very brittle, compared with the epsilon, and its presence in babbitt metals is, therefore, to be avoided.

The constitution of the tin-base babbitts containing copper and antimony has been published in the TRANSACTIONS,[9] while the copper-tin diagram is well known and can be found in most textbooks.

CHRISTOPHER H. BIERBAUM.—The advantage of using the Al_2O_3 crystal rather than steel is twofold; it enables the grinding of a point having a much higher degree of refinement and its hardness offers means for testing harder materials. In order to make comparative cuts in a hard and a soft material, whose widths shall have comparative values, it is necessary that the cutting point be symmetrical for both cuts, a condition that does not obtain unless the cutting point has been ground with a corresponding design and refinement.

The microhardness of any formation in the 90–10 copper-tin alloy can be determined, though the variation of microhardness of the solid solution may vary surprisingly within any one sample. The mount for the microcharacter shown in Fig. 14 is suitable in all cases, except when it is desirable to have the microcut pass through the optical axis of the microscope.

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[9] *Trans.* (1919) **60**, 162.

As the jewel is mounted in the jewel spring with a high degree of accuracy, it is only necessary that, with a reasonable degree of care, the jewel suspension be paralleled with the longitudinal feed of the microcharacter, an adjustment that is well within the skill of the ordinary microscopist. There is no looseness of the jewel bracket, when the same is in proper adjustment. For accurate work, the jewel should be held elastically; any constraining guides would introduce error due to friction. There should be no unusual difficulty in focusing, as the proper surface to focus on is the surface that is being tested.

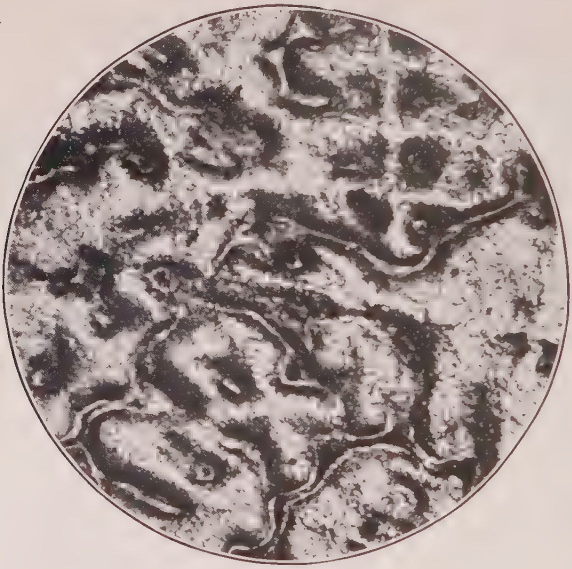

FIG. 15.—COPPER-TIN BRONZE SHOWING FUSION OF PART OF DELTA. × 200.

A photomicrograph of a section of a tooth of a motor-truck drive worm-wheel, normal to the bearing surface, is shown in Fig. 15; the lower edge represents the material on the bearing surface. In the upper portion is shown the delta crystal, in its original natural shape and position; whereas, in the lower portion is shown a complete fusion of the delta, which, under pressure, has formed distinct canals in which it flowed in order to effect an equalization of pressure. In this case, the fusion did not extend beyond a depth of 0.013 in. The heating of this "high spot" was practically instantaneous and the instant the fusion occurred and the flow was effected, all cause for heating was instantly removed; the chilling of the same also was almost instantaneous. This particular worm-wheel had given most excellent service with no indications of having "heated," having the standard composition of copper 89 per cent., tin 11 per cent. This illustration, together with studies of the phenomena of tin sweat on the foregoing alloy, seems to indicate that this transformation does not occur until a temperature has been reached appreciably above the melting point of the delta; and, furthermore, that an appreciable time interval is necessary to effect the same.

A photomicrograph of "Fahrig" babbitt of the composition, tin 90 per cent., copper 10 per cent., is shown in Fig. 16. The inner, or darker, crystals are the ones for which a microhardness of 1006 has been given. The outer light-colored envelope is softer and closely corresponds to the microhardness of that given for the delta in copper-tin bronze, the one discussed in the preceding figure. For that reason these two compositions were taken as identical. In the tin-rich alloy in Fig. 16, there can be little doubt that the inner, darker crystal contains the higher percentage of copper and the outer the lower; for which reason, in the table of microhardness, they were classified according to the chemical formulas in the manner given. The composition shown in Fig. 16 is not to be condemned as a bearing metal, except when used on the softer steels; when used on alloy or pack-hardened journals, it gives remarkably good service.

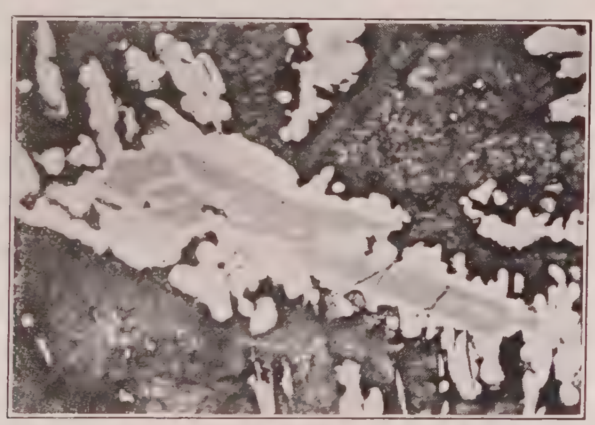

FIG. 16.—FAHRIG BABBITT, THE DARKER INNER CRYSTAL HAVING A MICROHARDNESS OF 1006. × 500.

Effect of Heat Treatment on Release of Stress in Bronze Castings*

By Robert J. Anderson,† Met. E., Pittsburgh, Pa., and Charles H. Eldridge,‡ E. M., New York, N. Y.

(Cleveland Meeting, May, 1923)

When a metal or alloy is poured into a mold, internal stresses are set up by the contraction in volume on passing from the liquid state at the temperature of pouring to the solid state at the ordinary temperature. It is assumed by the writers that all castings are internally stressed. That any casting is internally stressed is shown by its warping or distortion on maturing at the ordinary temperature, by warping or cracking on heating, or by cracking or distortion on machining.

In the investigation made by the writers, the internal stresses in cast non-ferrous alloys were studied with a view to examining the effects of heat treatment (*i.e.*, simple anneal) on the release of such stresses. Data have been obtained that show quantitatively that internal stresses can be removed by appropriate heat treatment. The investigation was limited to a study of the effect of heat treatment on stress release in cast rings of the 87:7:5:1 copper-tin-zinc-lead alloy.

The writers are indebted to E. G. Fahlman for the method used for inducing stress in cast rings; to M. E. Boyd for microscopic examination and for other aid; and to G. B. Dalrymple for the chemical analyses.

General Information

While casting stresses have been discussed considerably, they have usually been treated in a general way, and so far as the writers know no measurements of the magnitude of internal stresses in castings have been reported. It has been suggested by many investigators, particularly in the case of steel castings, that castings are internally stressed, and that these stresses can be removed by annealing. In a casting, the internal stresses set up on cooling may vary from a few hundred pounds to more than the ultimate strength of the alloy in the solid state at elevated temperatures. While the stresses may be, and probably usually are, small (below the proportional limit of the material) their prolonged action will cause distortion and warping, if not cracking, when the casting is allowed to stand at the ordinary temperature.

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† Metallurgist, Bureau of Mines.

‡ Research Metallurgist, Metropolitan Museum of Art.

Necessarily, internal stresses, and actual cracking of a casting in the mold, depend on a number of factors, of which the pouring temperature, mechanical properties of the alloy, contraction in volume, design of the casting, and method of molding are the most important. Internal stresses, set up largely by contraction in volume, are the cause of cracks in castings, as has been shown by one of the writers;[1] and whether internal stresses cause cracking or not is dependent on conditions in the mold at the time of pouring, and on their magnitude.

Stresses can result from differential thermal expansion of constituents during cooling through the solidification and transformation ranges in a bronze, but the writers think that this is a minor factor with reference to the magnitude of the stresses induced by the total contraction in volume of the alloy.

Rolfe[2] finds that phosphor-bronze castings crack on machining because of the release of internal stresses set up by contraction. He suggests preventing the occurrence of stresses by casting in chill molds instead of sand molds and releasing the stresses by heating for 1 hr. at 650° C. He deals with the mechanism of stress inducement on casting, and suggests chill casting so that the alloy is frozen "so suddenly that all contraction stresses are prevented." The writers do not agree that contraction stresses can be prevented by rapid freezing, but for lack of space, it is not possible to discuss here the mechanism of internal-stress inducement, nor to deal with Rolfe's explanation.

NEGATIVE EXPERIMENTS ON CASTING STRAINS

At the outset of the work, considerable time was spent by one of the writers and E. G. Fahlman in attempting to design castings that would have the same amount of internal stress in successive pieces cast under the same conditions, but the results were not satisfactory. If castings having substantially the same amount of internal stress in successive pieces could be made, the effect of heat treatment on the release of the stress could be readily determined. Several methods of gating parallelepiped castings were tried, and different forms of castings were poured. When such castings, as poured, are cut through at a given point, the ends spring apart, indicating release of stress. When such castings, after annealing, are cut through, the amount of spring is substantially less, indicating release of stress by heat. Several methods of gating ring-shaped castings were tried, and different types of rings were cast in other experiments. If a casting in the form of a ring is cast in a mold, and then cut through at one place, the two free ends will spring apart because of the release of the internal stress. Thus, in the case of one ring casting, which was 5⅝ in. outside diameter, 5⁄32 in. thick, and 7⁄16 in. deep, the gap between the ends after cutting was 0.2 in. When such a ring was heated for 1 hr. at 600° C., the gap after cutting was only 0.004 in. The calculated circumferential stress indicated by the springing apart of such a ring to 0.2 in. is 3800 lb. per sq. in.

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[1] R. J. Anderson: Cracks in Aluminum-alloy Castings. *Trans.* (1923) **68,** 833.

[2] R. T. Rolfe: Phosphor-bronze from the Engineering Standpoint. *The Metal Ind.* (London) (1922) **20,** 413–416; 437–438.

It was not possible to cast successive rings, or successive parallelepipeds that would spring apart the same distance on cutting, so that this method of attack was abandoned. It was finally decided to cast rings under the same conditions of molding, cut them so that the stress would be released, then induce circumferential stress in them by holding the free ends apart, and examine the effect of heat treatment on the release of the induced stress. This is an indirect method, but it permits quantitative measurement of the amount of stress released by heat.

Method of Experiment

The bronze used in the experiments had the nominal composition 87:7:5:1 copper-tin-zinc-lead, while the actual analysis was: Copper 86.80 per cent., tin 7.00 per cent., zinc (by difference) 5.09 per cent., lead 1.11 per cent.

The average physical properties of the bronze for sand-cast bars were:

Yield point, pounds per square inch	22,300
Ultimate strength, pounds per square inch	38,000
Elongation on a 2-in. length, per cent	22.5
Reduction in area, per cent	20
Brinell hardness (10 mm., 500 kg., 30 sec.)	53
Scleroscope hardness (magnifier hammer)	15
Determined modulus of elasticity, pounds	11,550,000

If a cast ring is cut through, the ends will spring apart because of the release of the internal stress. The circumferential stress is the largest, but the ring, when cut, opens out to a larger curvature as a result of a combination of stresses. The longitudinal internal stress in a cast ring of the type used in the experiments must be slight, and in the calculation of stress, other stresses than those in a circumferential direction are neglected. The indirect method used by the writers has been used also by Moore and Beckinsale[3] in examining the effect of heat on stress release in brass strips, wherein strips were bent around a circular form and held in position by clamps. The actual stress in a cut ring may be calculated readily from the formula[4]

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[3] H. Moore and S. Beckinsale: The Prevention of Season Cracking in Brass by the Removal of Internal Strains. *Trans.* Faraday Soc. (1921) **17,** 162–192; and a series of related papers elsewhere.

$$\text{Stress} = \frac{EtW}{4\pi r_1 r_2}$$

where E = modulus of elasticity of alloy;
t = thickness of ring;
W = width of opening (*i.e.*, amount of gap);
r_1 = radius of curvature after release of stress;
r_2 = radius of curvature before release of stress.

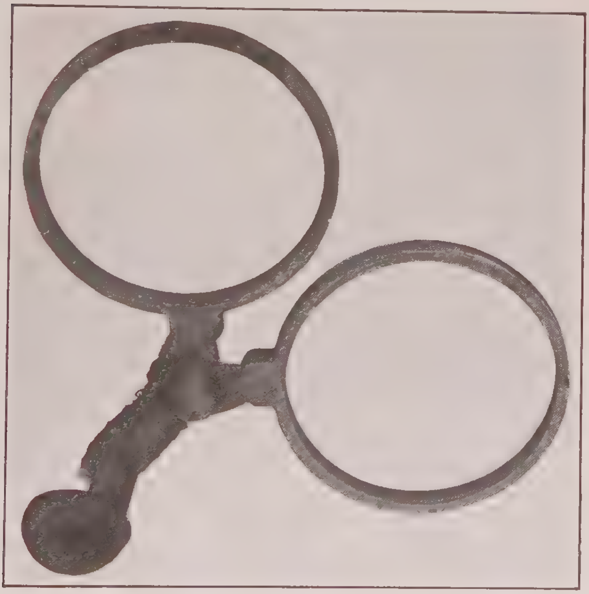

FIG. 1.—PAIR OF RINGS WITH GATES ATTACHED.

In the experiments, a large number of rings 6 in. outside diameter by ½ in. wide by 0.288 in. thick were cast in pairs in sand molds under the same conditions of molding; the pouring temperature was 1090° C. Fig. 1 shows a pair of rings with gates attached. After casting, the rings were split, by a radial saw, through the center of the lugs. After cutting, the rings were machined by milling the cut lug faces so that the gap was increased to 0.25 in., measured along the chord subtending the central diameter of the ring between the inside and outside diameters. Fig. 2 shows the dimensions of a machined ring.

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[4] W. H. Hatfield and G. L. Thirkell: Season Cracking. *Jnl.* Inst. of Metals (1919) **22**, 67–91.

Predetermined amounts of circumferential stress can be induced by opening the gap to definite distances, for example, by inserting a wedge so that the ring is opened to a larger curvature. It was desired to examine the effect of heat on the release of a small induced stress, 6400 lb. per sq. in., and on a larger induced stress, 11,200 lb. per sq. in. To induce these stresses in the rings, it is necessary to make the distance between the lug faces 0.45 and 0.60 in., respectively; that is, the initial gap of 0.25 in. must be widened by 0.20 and 0.35 in., respectively. Thus, from the formula

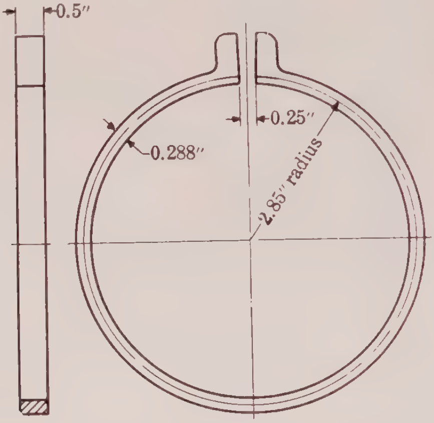

FIG. 2.—DIMENSIONS OF MACHINED RING.

$$\text{Stress} = S = \frac{EtW}{4\pi r_1 r_2},$$

$$\text{and gap} = W = \frac{S \times 4\pi r_1 r_2}{Et},$$

or for an induced stress of 11,200 lb.;

$$W = \frac{11{,}200(4\pi \times 2.91 \times 2.85)}{11{,}550{,}000 \times 0.288} = 0.35 \text{ in.}$$

Special wedges were made to force the lug faces apart to induce stress. They were made of two strips of low-carbon steel, of the form shown in Fig. 3, 2 in. long by ½ in. wide, beveled at the ends, and the two halves of each wedge were separated by a steel-ball bearing, resting in a recessed socket drilled in each strip. For inducing the larger stress, the strips were ¼ in. thick and rode on ¼-in. ball bearings; while for inducing the smaller stress, the strips were 3⁄16 in. thick and rode on 3⁄16-in. ball bearings. The wedges were made of such size that when properly driven into the lug gaps, so that a line scribed on the wedge face was in alignment with the central circumference of the ring, the initial gaps were

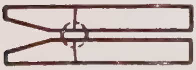

FIG. 3.—SIDE VIEW OF WEDGE, USED FOR INDUCING STRESS IN RINGS.

widened the required distance. Fig. 4 shows a ring wedged so that circumferential stress is induced. For the inducement of stress, the wedge was pushed into the lug gap until the chord measurement of the gap across the center line was exactly 0.45 in. (for 6400 lb.) and 0.60 in. (for 11,200 lb.). A slight error is introduced by measuring the chord rather than the circumferential distance of the opening.

The stressed rings were heated for various times and at different temperatures in the range 200° to 550° C. in an electric muffle furnace.

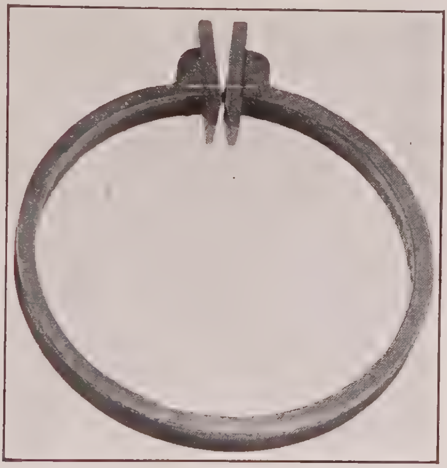

Fig. 4.—Ring wedged open so that circumferential stress is induced.

The rings were then cooled in air to the ordinary temperature, the wedges were removed, and the ring gap was measured. If the ring gap is widened by a wedge so that the induced stress is below the elastic limit of the alloy, when the wedge is removed the ends spring back so that the opening is the same as originally. However, if a ring so wedged is heated, when the wedge is removed the gap will be wider than originally; the final width of the opening will depend on the time and temperature of the heating.

Results of the Heat-treatment Experiments

Tables 1 and 2 give the results of the heat-treatment experiments on the release of induced stress in the cast rings. The data show the stress released in pounds and percentage of the original induced stress, as calculated. Table 1 gives the results of heating on the release of stress in the rings in which stress in the amount of 6400 lb. per sq. in. was induced. When heated for 1 hr. at 200° C., the width of the ring gap was 0.27 in., or the gap was 0.02 in. wider, indicating release of some of the stress. After heating for 20 hr. at 450° C., the width of the ring gap was 0.45 in., indicating that all the stress had been removed. The data in Table 1 show that the amount of stress released is a function of the time and temperature of heating. Thus, while all of the stress is removed by heating for 1 hr. at 550° C., it required heating for 8 hr. at 475° C. for complete removal. Fig. 5 gives the curves plotted from the heat-treatment data, in percentage of stress released against the time of heating

TABLE 1.—*Effect of Heat on Stress Release in Cast Bronze*

Induced Stress, 6400 Lb. per Sq. In.

Temperature of Heating, Degrees C.	Time of Heating, Hours	Ring Gap,[a] Inch		Stress, Pounds per Square Inch		Stress Released, Per Cent.
		Final [b]	Difference [c] W	After Heating	Released	
200	1	0.27	0.02	5,750	650	10
200	2	0.30	0.05	4,700	1,700	27
400	1	0.33	0.08	3,800	2,600	40
400	2	0.37	0.12	2,600	3,800	58
400	3	0.38	0.13	2,200	4,200	65
400	4	0.38	0.13	2,200	4,200	65
450	½	0.34	0.09	3,200	3,200	50
450	1	0.38	0.13	2,200	4,200	65
450	3	0.39	0.14	1,900	4,500	70
450	4	0.40	0.15	1,600	4,800	75
450	5	0.42	0.17	900	5,500	86
450	10	0.43	0.18	600	5,800	90
450	20	0.45	0.20	0	6,400	100
450	24	0.45	0.20	0	6,400	100
475	2	0.40	0.15	1,600	4,800	75
475	3	0.42	0.17	900	5,500	86
475	6	0.44	0.19	300	6,100	95
475	8	0.45	0.20	0	6,400	100
500	¼	0.34	0.09	3,200	3,200	50
500	⅓	0.35	0.10	3,100	3,300	52
500	½	0.40	0.15	1,600	4,800	75
500	1	0.42	0.17	900	5,500	86
500	2	0.43	0.18	600	5,800	91
500	4	0.45	0.20	0	6,400	100
550	⅙	0.37	0.12	2,600	3,800	60
550	¼	0.40	0.15	1,600	4,800	75
550	½	0.44	0.19	300	6,100	95
550	1	0.45	0.20	0	6,400	100

[a] Initial gap, before wedging, 0.25 in.; initial gap, wedged, 0.45 in.

[b] Final gap, width of opening on removal of wedge after heating.

[c] Difference *W* between final gap and initial gap (0.25 in. gap).

TABLE 2.—*Effect of Heat on Stress Release in Cast Bronze*

Induced Stress, 11,200 Lb. per Sq. In.

Temperature of Heating, Degrees C.	Time of Heating, Hours	Ring Gap,[a] Inch		Stress, Pounds per Square Inch		Stress Released, Per Cent.
		Final[b]	Difference[c] W	After Heating	Released	
200	1	0.28	0.03	10,270	930	8
200	2	0.29	0.04	9,900	1,300	12
200	4	0.39	0.14	6,700	4,500	40
400	1	0.42	0.17	5,700	5,500	48
400	2	0.47	0.22	4,200	7,000	62
400	3	0.48	0.23	3,800	7,400	66
400	4	0.48	0.23	3,800	7,400	66
450	½	0.43	0.18	5,400	5,800	52
450	1	0.48	0.23	3,800	7,400	65
450	2	0.50	0.25	3,200	8,000	71
450	3	0.51	0.26	2,800	8,400	75
450	5	0.54	0.29	1,900	9.300	83
450	10	0.59	0.34	300	10,900	92
450	20	0.60	0.35	0	11,200	100
450	24	0.60	0.35	0	11,200	100
475	3	0.57	0.32	1,000	10,200	91
475	6	0.59	0.34	300	10,900	92
475	8	0.60	0.35	0	11,200	100
500	¼	0.43	0.18	5,400	5,800	52
500	⅓	0.47	0.22	4,200	7,000	62
500	½	0.54	0.29	1,900	9,300	83
500	1	0.57	0.32	1,000	10,200	91
500	2	0.57	0.32	1,000	10,200	91
500	4	0.60	0.35	0	11,200	100
550	⅙	0.47	0.22	4,200	7,000	62
550	¼	0.55	0.30	1,600	9,600	85
550	½	0.59	0.34	300	10,900	92
550	1	0.60	0.35	0	11,200	100

[a] Initial gap, before wedging, 0.25 in.; initial gap, wedged, 0.60 in.

[b] Final gap, width of opening on removal of wedge after heating.

[c] Difference W between final gap and initial gap (0.25 in. gap).

for different temperatures. The calculations of values given in the column "stress released" were made by using the formula,

$$\text{Stress} = S = \frac{EtW}{4\pi r_1 r_2}.$$

The column showing the difference (W) in the ring gap gives the actual values found for the difference between the original width of opening and the final width on removal of the stress for the different heat treatments. A specimen calculation shows the method of quantita-

TABLE 3.—*Effect of Annealing on the Physical Properties of* 87 :7 :5 :1 *Copper-tin-zinc-lead Alloy; Sand Cast*

Heat Treatment Temperature, Degrees C.	Time, Hours	Yield Point, Lb. per Sq. In.[a]	Ultimate Strength, Lb. per Sq. In.	Elongation in 2 In., Per Cent.	Reduction in Area, Per Cent.	Brinell[b] Hardness	Scleroscope Hardness[c]
as cast		22,300	38,000	22.5	19.6	51–55	15
371	20	26,150	34,250	16.7	16.5	54–57	12
482	12	21,500	39,500	29.0	20.0	54	12
566	3	22,250	34,400	27.5	15.9	50	12
566	7	22,500	38,750	38.2	26.6	50–54	12
621	1	23,100	34,600	22.2	20.9	50	12
621	2	21,750	38,100	35.0	25.8	50–54	12
700	1	21,750	38,600	40.0	29.2	50–54	11

[a] Yield point with Berry strain gage.
[b] 10-mm. ball, 500-kg. load, for 30 sec.
[c] Magnifier hammer.

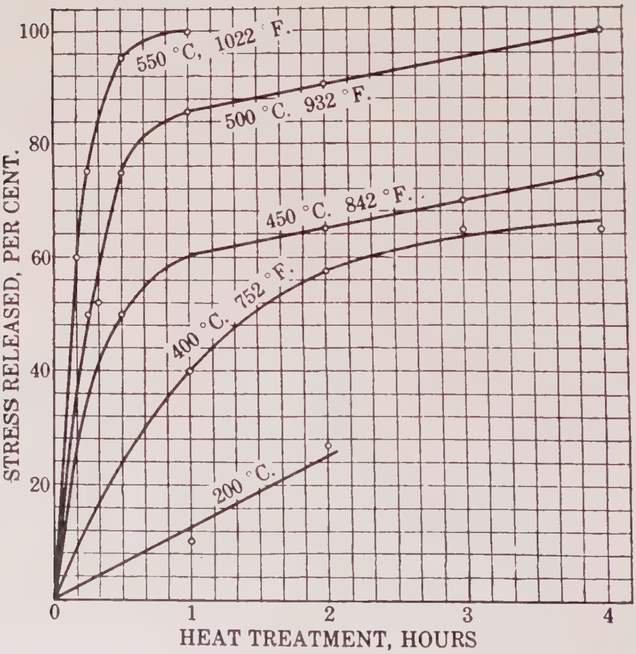

FIG. 5.—EFFECT OF HEAT ON STRESS RELEASE IN BRONZE; INDUCED STRESS, 6400 LB. PER SQ. IN.

tively estimating the amount of stress removed by heat treatment, according to the method used. Taking the case where a ring was heated for 2 hr. at 400° C., as shown in the fourth line of Table 1:

$W = 0.12$ in. $\qquad E = 11{,}550{,}000$ lb.

$t = 0.288$ in. $\qquad r_2 = 2.85$ in., original radius of curvature

$$r_1 = \frac{W + 2\pi r_2}{2\pi} = \frac{0.12 + 2\pi(2.85)}{2\pi} = 2.87 \text{ in.}$$

$$\text{Then, stress} = S = \frac{EtW}{4\pi r_1 r_2} = \frac{11{,}550{,}000 \times 0.288 \times 0.12}{4\pi(2.85)(2.87)} = 3800 \text{ lb.}$$

per sq. in., stress released.

The results given in Table 1 indicate that the first effects of heat are to remove stress rapidly, and this is followed by a much slower removal of stress with an increasing period of exposure at a given temperature.

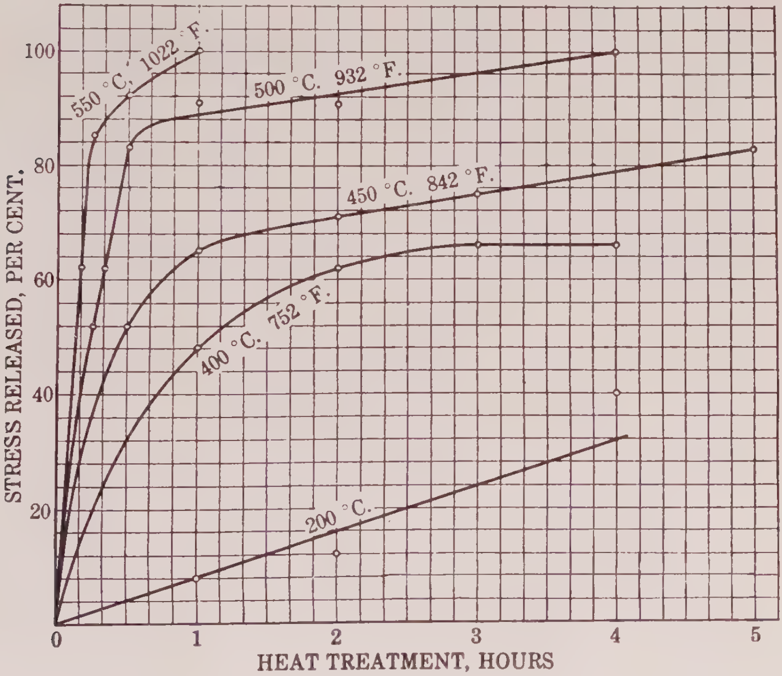

Fig. 6.—Effect of heat on stress release in bronze; induced stress, 11,200 lb. per sq. in.

Table 2 gives the results of heating on the release of stress in the rings in which stress to the amount of 11,200 lb. per sq. in. was induced; the results are shown in Fig. 6. The release of stress is rapid during the early stages of heating above 400° C. Thus, at 550° C., 85 per cent. of the induced stress was released in 15 min., but it required heating for 1 hr. to remove all of the stress. At 500 ° C., 1 hr. was required to remove

91 per cent. of the stress, and 3 hr. more to release the remaining 9 per cent. The curves in Figs. 5 and 6 are similar for a given temperature, and the higher the initial induced stress the greater is the amount of stress removed by a given heat treatment.

Tensile tests were made on sand-cast bars of the bronze after heating for various times and at different temperatures, with the results given in Table 3. Annealing this cast bronze raises the elongation and reduction in area substantially. The strength, yield point, and hardness are affected but little. Within the limits of time and temperature employed, heat treatment causes a partial removal of the typical coring of the alloy as cast. Fig. 7 shows the average microstructure of one of the rings as cast, and Fig. 8 shows the structure, after heating for 1 hr. at 550° C., of a ring that had been stressed to 6400 lb. per square inch.

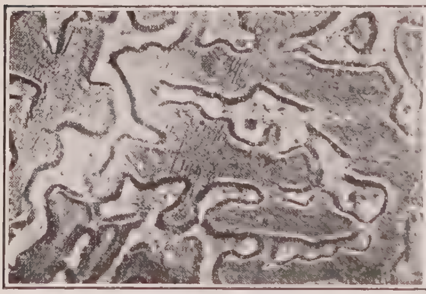

Fig. 7.—Average microstructure of 87:7:5:1 copper-tin-zinc-lead alloy; sand cast; etched $NH_4OH + H_2O_2$. × 300.

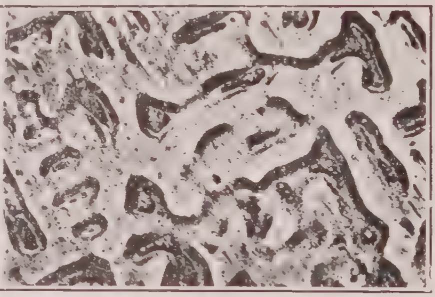

Fig. 8.—Same as Fig. 7, but after stressed to 6400 lb. per sq. in. and annealed for 1 hr. at 550° C. × 300.

Conclusions

The following conclusions may be drawn from the results of the experiments:

1. Internal stresses exist in castings, largely because of contraction in volume of the alloy on freezing and cooling.
2. Stresses induced, or occurring, in a cast bronze can be released by heating; *i.e.*, both the stresses induced intentionally, and the actual internal casting stresses, within the elastic limit.
3. Measurement has been made of the actual internal stress in cast forms, and it has been shown that internal strains can be removed by heating.
4. A method has been devised for inducing stress in cast rings and for measuring the amount of the stress.
5. In the case of different induced internal stresses (6400 lb. and 11,200 lb.), the higher the initial stress the greater is the amount of stress removed by a given heat treatment.

6. The higher the temperature of heating, the more rapid is the removal of stress.

7. The rate of reduction of induced stress increases with increasing temperature, but the rate decreases as the stress is reduced by heating at a given temperature.

8. With temperatures above 400° C., for a given temperature, the greater part of the stress is released quickly, and the time required for the release of the remaining stress is less the higher the temperature.

9. For the removal of casting stresses in bronzes, heating for 1 hr. at 550° C. should remove all injurious stresses. The heating may be for a shorter time at a higher temperature or for a longer time at a lower temperature.

10. In the heat treatment of castings for the removal of stress, estimation of the approximate amount of stress can be made by machining, followed by measurement of the distortion and calculation, and the heat treatment to be employed can be determined empirically. While heating for 1 hr. at 550° C. removed both casting and induced stresses up to 11,200 lb. in particular castings, the time period of exposure and temperature to be employed in treating industrial castings will depend on the size and the shape of the castings.

11. The heat treatment found suitable for the release of stress in the rings has been applied to commercial castings with satisfactory results.

DISCUSSION

N. K. B. Patch, Buffalo, N. Y.—Our experience is that heat treatment affects the microstructure of aluminum bronze, which is desirable for certain types of service. We have, however, had rather interesting experiences with the copper-tin-bronze, more particularly those having a eutectic alloy. In one case, a flat plate warped when machined because of the internal stresses set up in the cooling; by annealing the plate these stresses are removed and the plate stays flat after it has been machined.

Wm. K. Frank, Pittsburgh, Pa.—Some years ago, we made some castings according to specifications that gave the chemical and physical properties desired. Our preliminary tests showed that the physical properties could be met with the composition given, so we made up a large batch of the castings; but to our surprise a number of the pieces showed low tensile strength, low ductility and low elongation. When we realized that we had the eutectic alloy in this bronze, it was about 12 per cent. tin, we heated the castings to a cherry red and quenched them; our physical properties were very much improved. However, these were plain castings; this heating and quenching would not be possible for intricate castings.

Clair Upthegrove, Ann Arbor, Mich.—Can anything be done to determine the extent to which the internal stresses must be removed to prevent trouble?

Robert J. Anderson.—No; we assumed they should be removed entirely. In experimenting with different designs of castings, it was found that in order to remove stresses completely in the shortest possible time, about one hour is required at 600° C. and a longer time when heating at lower temperatures.

N. K. B. Patch.—There are occasions where a proper anneal in the case of an alloy like 88–10–2 will correct the trouble of leakage due to hydraulic strain, whereas without annealing the casting might leak at some point where thick and thin sections join one another because of the separating out of the eutectic at those points.

Guilliam H. Clamer, Philadelphia, Pa.—That is an interesting fact for those who have tried to meet the Government's specifications on 88–10–2, without any lead content. The addition of about 1 per cent. lead, where permitted, will make the castings capable of withstanding internal pressure.

Solid Solutions*

By Walter Rosenhain, F. R. S., London, England.

In selecting solid solutions for the subject of this lecture I have been guided by several considerations. The bodies known under that somewhat paradoxical name play a most important part in all types of alloys; consequently, a deeper knowledge of their constitution must be of fundamental importance to both metallurgical science and practice. Further, having been asked to deal with some of the more theoretical aspects of metallurgy, I have been tempted to describe certain considerations, which I believe to be novel, in regard to the nature of solid solutions. These considerations constitute a theory that requires much experimental confirmation, but they have passed the stage of speculations, for experimental evidence has become available and one inference after another, that can be derived from the theory, is found to be in accord with well-known facts. In putting before you these ideas and inferences, it must be understood that detailed quantitative proof of every step cannot be furnished. Our conclusions must therefore be stated with the reservation that their acceptance must depend on the manner in which the discovery of fresh facts may either confirm or confound them.

If we wish to understand the structure of solid solutions we must carefully realize the manner in which these bodies are formed in alloys. The formation of solid solutions is by no means confined to alloys, but it is desirable to confine attention, at the outset, to intermetallic solid solutions.

It is not easy to formulate a precise definition of the term "solid solution." It may be said that a solid solution is a crystalline body formed from a liquid solution or from an intimate mixture of solids, or of liquid and solid, that retains in the solid state an admixture of the two substances (solvent and solute) as intimate as that which exists in a liquid solution. The term is undoubtedly paradoxical, for one usually associates solution with a liquid, but the difficulty cannot be evaded by using such a term as solidified solution, for this might refer to a solution that has set, or congealed, without undergoing crystallization. The German term *mischkrystall* avoids the difficulty and appears to be a more exact description of these bodies; but the English translation "mixed crystals" conveys the misleading idea that the resulting solid substance is a mixture of two or more kinds of crystals. For want of a better term, therefore, we may well retain "solid solution;" its paradoxical character is soon obliterated by use, and a fuller understanding of the nature of such bodies will assist us to fix our ideas as to what the term implies.

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*Second Annual Lecture of the Institute of Metals Division, the American Institute of Mining and Metallurgical Engineers, at the New York Meeting, February, 1923.

EQUILIBRIUM DIAGRAMS

In order to appreciate the manner in which solid solutions are formed, we may consider briefly the manner in which a liquid alloy solidifies. This process is best represented by means of equilibrium diagrams. We have first the extreme types in which the two metals form no solid solutions at all, as represented by Fig 1, and where they form an unbroken series of solid solutions, as in Fig. 2. It is questionable whether any alloys are of the type shown in Fig. 1, but as the limiting type this may be regarded as something which may be closely approached but never quite reached.

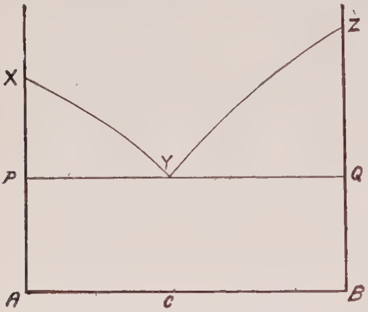

FIG. 1

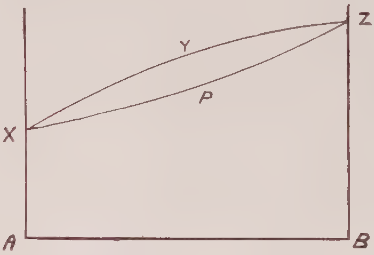

FIG. 2

FIGS. 1 AND 2.—EXTREME TYPES OF EQUILIBRIUM DIAGRAMS.

In a system of the first type, the process of solidification is simple. Any alloy, the composition of which lies between *A* and *C*, will begin to solidify as soon as its temperature falls sufficiently to bring the alloy to the point on the line *XY* that corresponds with its composition. At that point, pure metal *A* begins to be separated in the form of solid crystals, and this separation continues until the temperature falls to that of the line *PQ* (the eutectic temperature). By the separation of solid *A*, the residual liquid becomes richer in *B*; at the same time, as the temperature reaches that of the line *PQ*, the concentration of the liquid reaches that corresponding to the eutectic concentration of *Y* or *C*, and the remaining liquid solidifies by the simultaneous crystallization of *A* and *B*, generally in a finely laminated condition, forming the well-known eutectic constituent. A similar description with *A* and *B* reversed, applies to the alloys lying between *C* and *B*.

In a system of the second type, the separation of *A* from *B* does not occur except to a small and transitory extent. As soon as an alloy cools down to a temperature corresponding to that of the point on the line *XYZ*, which represents the composition of that alloy, solidification commences; the solid first formed, however, is not pure *A* nor pure *B*, but a crystalline mixture or solid solution of one in the other. The material that first crystallizes from such liquid alloy has not the same composition as the liquid, but if solidification has been slow enough to allow equilibrium to be attained, the resulting crystalline solid is completely homogeneous in regard to composition and has the same composition as the liquid solution from which it has been formed. In such a system, of which copper-nickel, iron-nickel, silver-gold, tungsten-molybdenum, and silver-palladium are examples, we must have such conditions that the solubility of the two metals in one another is unlimited in the solid state. If we could find a system like that illustrated in Fig. 1, we would realize the condition that the two constituent metals are completely insoluble in one another in the solid state.

As this condition is probably never realized, there is a series of intermediate types of alloy systems having a limited range of concentration within which one metal is soluble in the other when solid, but beyond this range separation on the lines indicated in Fig. 1 occurs. One of these intermediate types is shown in Fig. 3. In such a system, solidification always begins along the lines *XY* and *ZY* by the separation of solid solution. In alloys in which concentration lies between *A* and *P* (which is not necessarily or usually vertically above *C*), solidification is completed when the line *XP* is reached; and for alloys in which concentration in *B* is less than that of *P*, the resulting solid solution is still unsaturated, *i.e.*, it contains less than the limiting possible amount of *B*. The point *P*, therefore, represents the limit of solubility of *B* in the solid *A* at that temperature. This limit may increase or decrease with falling temperature and if *C* represents the limit at the ordinary temperature, the line *PC*, which we may call the solid solubility line, will slope toward *A* or *B*. I have shown elsewhere the great importance of such a sloping solubility line, as on it depends the power of the alloy to undergo hardening by quenching and tempering. For alloys, when the composition is between those of *P* and *Y*, this process begins by the separation of the solid solution of *B* in *A*, but when the temperature of *PY* is reached, the solid solution has reached saturation and the liquid has attained the eutectic composition (corresponding to the point *Y*). The residual liquid then solidifies as eutectic, but the constituent crystals of the eutectic are saturated solid solutions of *B* in *A* and of *A* in *B*.

One somewhat simpler type of equilibrium diagram is illustrated in Fig. 4. Here, again, are two series of solid solutions, but there is no formation of a eutectic, in which respect the type resembles that of Fig. 2. However, at *Y*, where the two types of solid solution meet, there is a transition point and connected with it a horizontal transition line. Alloys between *X* and *Y* solidify wholly as a solid solution of *B* in *A*; alloys between *Z* and *Q* wholly as solid solutions of *A* in *B*. Along the line *YQ* the solid solution of *A* in *B* is transformed, with falling temperature, into the solid solution of *B* in *A*. Alloys between *P* and *Q* begin their solidification by depositing crystals of the solid solution of *A* in *B*, but at the line *PQ* the remaining liquid solidifies as crystals of the saturated solid solution of *B* in *A*; while for alloys between *Y* and *P*, solidification begins by the formation of crystals of the solid solution of *A* in *B*, but these are transformed, at the line *YP*, into crystals of the other solid solution.

These typical systems, while apparently complex enough in themselves, are found in actual alloy systems in complex combinations and as yet we are not in a position to predict what type of solidification the binary alloys of any pair of metals will follow. One can see, however, that a fundamental principle underlies all the diagrams so that, if we knew enough about the manner in which the metallic atoms act upon one another, we might be able to write down an equation from which any desired equilibrium diagram could be derived by the insertion of the proper constants. That is not yet possible, but the considerations to be put forward later should, if they are found to agree with all the facts, furnish at least one step in that direction.

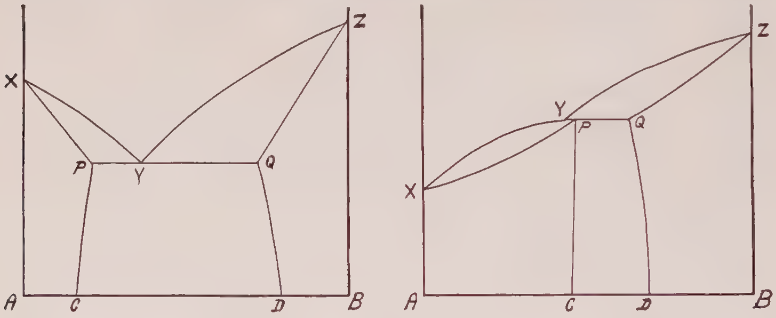

FIG. 3 FIG. 4

FIGS. 3 AND 4.—INTERMEDIATE TYPES OF EQUILIBRIUM DIAGRAMS.

If we think, now, of the typical equilibrium diagrams which we have considered in Figs. 1 to 4, and particularly of the last three, the question naturally presents itself, what is the reason for or the mechanism of the formation of solid solutions?—why are they formed and what are the factors or conditions that govern the limit of solid solubility? These questions are of a thermodynamical nature and a good deal of mathematical theory on this subject has been written. Without desiring to detract from the value of that method of treating the subject, I cannot regard it as satisfying. To help us understand what happens in the formation of solid-solution crystals it is desirable to have physical conceptions and these are not afforded by thermodynamic equations. No doubt it is a more difficult task to endeavor to build an adequate satisfactory physical conception, but the result is likely to be more fruitful. It must not be forgotten, however, that a physical conception can be nothing more than a symbol by means of which we represent one aspect of the physical reality, and the thermodynamic equation is but a symbol of a different type.

NATURE OF SOLID-SOLUTION CRYSTAL

The nature of the solid-solution crystal may be considered from two points of view: its behavior in regard to melting and in regard to solidification. In the case of alloys following one of the modifications of the type shown in Fig. 3, the melting temperature falls with increasing concentration of the solid solution. This means that the amount of kinetic energy that must be communicated to the constituent atoms or molecules of the crystal to bring about disruption or melting is lessened by increasing concentration of the solid solution; therefore, in some form or other these crystals already hold a store of energy that tends to assist the kinetic energy of rising temperature in bringing about disruption of the crystal. We are thus led to the assumption that the solid-solution crystal is a self-strained structure in which potentially disruptive energy is stored and the limit of solubility is determined by the amount of such energy that can be stored in the internal structure of the crystal.

If we think of the manner of formation of a solid-solution crystal, or rather of a duplex aggregate in an alloy containing just too much of the solute metal to form a homogeneous solid solution, we are led to much the same conclusion. The solid solution is formed from the liquid solution because it constitutes that arrangement, in the solid state, which contains the least amount of stored potential energy. If any other arrangement, such as separation into an aggregate of two kinds of crystals, offered a means of reducing the total amount of energy stored, we know (from elementary thermodynamic principles) that such an alternative would be instantly adopted. We conclude, therefore, that the solid solution is still formed, in alloys of increasing concentration, until a point is reached where the total energy content of a more highly saturated solid solution would be greater than that implied by a duplex structure, whether eutectic or otherwise. This implies that increasing concentration of the solid solution brings with it an increasing storage of internal energy until the limit is reached. The earlier appearance of a second kind of crystal, or phase, would involve a greater storage of energy, and this might reside within the crystals of the second phase, which must themselves be saturated solid solutions, or it might reside (and in part must reside) in the interfacial energy that must exist at the junction of dissimilar crystals.

Were we able to express the relations just indicated in a quantitative manner, we should be able to calculate the limits of solid solubility and the eutectic or transition temperatures of all alloy systems. As yet the data are lacking, but measurements of the latent heat of fusion of solid solutions of different concentrations and of the specific heats of the corresponding alloys in both the solid and liquid state would furnish at least some of the data needed. For the moment we must content ourselves with the general conclusion that solid solutions do contain internally stored energy and that it is the amount of this energy which determines the limits of solid solubility in these bodies.

X-ray Analysis

The investigations of Laue, Debye, Scherrer, Sir William and Prof. W. L. Bragg, Hull, Westgren and other *x*-ray analysts have given us a large amount of precise and detailed data where our knowledge had been largely qualitative. The truly crystalline character of metals had been recognized and demonstrated by metallographers and so had the persistence of crystalline structure after moderate amounts of plastic strain, as well as the whole mechanism by which crystals undergo plastic deformation. To one who investigated these problems more than twenty years ago, the present-day confirmation of many of the inferences drawn by means of much cruder methods is particularly gratifying and satisfactory.

To most of those interested in the deeper problems of physical metallurgy, the knowledge of the internal structure of crystals that has been obtained by *x*-ray analysis is probably familiar. The *x*-ray analyst measures the diffraction or scattering that occurs when a homogeneous beam of *x*-ray falls upon the surface of either a single crystal or an aggregate of a large number of minute crystals. His procedure is similar to that used when the "spacing" of a ruled grating is measured by means of the diffraction spectrum it produces upon an incident beam of visible light. If the wave length of the light is known the width of separation of the lines of the grating can be determined, and vice versa. It is, however, essential that the grating spacing should be proportionate in its dimensions to the wave length of the light used; otherwise the diffraction spectra become unduly weak. The wave lengths of the characteristic *x*-radiations of various substances are now well known and the spacing of the rows or planes of atoms in a crystal is so proportioned to the *x*-ray wave length as to furnish strong diffraction spectra; consequently, the spacing of the crystal planes can be determined from the *x*-ray spectrum in much the same way as the line spacing of a grating from a visible spectrum. The crystal, however, is not a plane reflector so that the diffracting objects are spaced out in three dimensions, which complicates the calculations but the principle remains the same. We need not discuss the various experimental methods that have been successfully used, beyond saying that it is possible to obtain clearly defined and easily interpreted spectra in an x-ray spectrometer from an aggregate of small crystals so that a large single crystal is no longer needed. It is by these latest methods that some of the measurements referred to here have been made by my colleagues at the National Physical Laboratory—Doctor Owen and G. D. Preston.

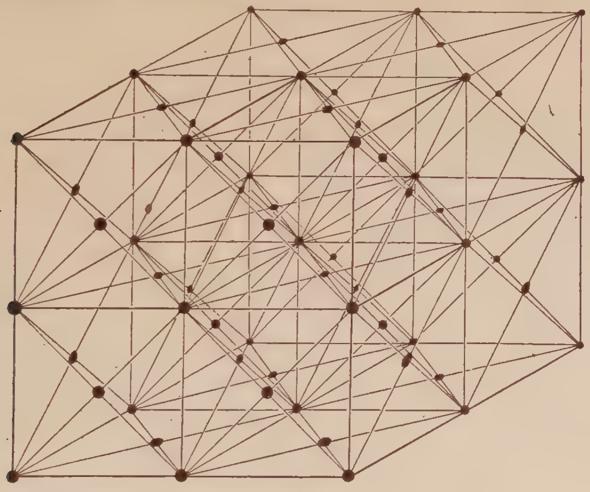

Fig. 5.—Face-centered cubic lattice.

The net result of x-ray crystal analysis may be briefly summed up. Metallic crystals yield very well-defined diffraction spectra from which, in all the simpler cases, the arrangement of the atoms can be determined. The results are most readily understood in terms of what is known as "space lattices." A space lattice is simply a system of lines running in three directions in space and crossing one another in a regular manner; at some or all of these intersection points are located the centers of the atoms. For instance, a lattice might consist of three sets of uniformly spaced lines at right angles to one another intersecting in such a way as to divide space into a number of equal and adjacent cubes. This would constitute the simplest possible cubic space lattice, but crystals built on such a lattice are not found among simple metals. By adding lines parallel to the three original sets, and placing atoms upon certain of the additional points of intersection thus provided, we may obtain the two types of cubic lattice most commonly met in metals. In one case, in addition to an atom at the corner of each cube, there is an atom at the center of each cube face; this gives the "face-centered cubic lattice," which is illustrated diagrammatically in Fig. 5. If instead of the atoms at the centers of the cube faces, there is an additional atom at the center of the cube itself, the "body-centered cubic lattice" is obtained; this form also is found in metals. By a slight shift of alternate sets of lines, the cubic system can be transformed into the "close-packed hexagonal" often found in metals. More complex types of lattice, possessing a lower degree of symmetry, are only rarely found in simple metals, of which bismuth and antimony are examples. In the case of intermetallic compounds, however, more complex types of arrangement are to be anticipated, although the exact structure of an intermetallic compound has not been worked out.

STRUCTURE OF CRYSTALS

We must think, then, of a crystal as being built up of atoms fixed upon an imaginary, but none the less effective, framework in space, the average distance of the atoms from center to center being most definitely governed in this way. Under the influence of thermal agitation, the atoms may be regarded as oscillating about these equilibrium positions. Thermal expansion, which is the consequence of this thermal oscillation and the resulting tendency to drive the atoms farther apart, brings with it a corresponding expansion of the lattice. This expansion cannot be carried beyond a certain point. If a certain limiting interatomic distance is exceeded, the interatomic linkage that holds the atoms in place breaks down. The result is the formation of a new phase. If the extension of the lattice is caused by thermal expansion, when the limiting extension is passed, the lattice breaks down and disappears more or less completely: the crystal melts, the new phase being liquid. If the lattice is extended by mechanical stress, the phase formed is Beilby's amorphous metal. In other cases, the new phase may be a crystalline solid having a different space lattice.

For the present discussion, the simplest plan would be to adopt the atom, suggested by the theory of Langmuir, consisting of a relatively large heavy positive nucleus with electrons placed about it in definite positions. Even if obliged to adopt the Bohr-Rutherford atom, with inner and outer electron shells rotating about the nucleus, we must realize that an atom which, in a state of equilibrium, comes to rest upon a definite space lattice must have certain strongly defined directional properties, which we may call "bonds." In a cubic substance, these bonds must tend to lie in directions parallel with the cubic axes. That the outer electrons play an important part in this interatomic bonding is indisputable, even though there is divergence of opinion on the precise mechanism by which they bring it about. None the less, the approximate dimensions of the atoms are known and, if we regard the "size" of the atom as determined by the diameter of its outer electron shell, we find that, in most metals, the dimensions of the space lattice are such that the atoms almost or quite touch one another. It must be borne in mind, however, that the atom, looked at in this way, is not an extremely hard, rigid body, but that the outer electrons or their orbits can be deflected not only by their mutual interaction as between an atom and its neighbors, but even by the application of external stress. Whatever, therefore, tends to change the dimensions of the space lattice must, to a corresponding extent, bring about a distortion of the atoms or at least of their outer electron shells.

With this conception of the atoms and their arrangement on the space lattice in our minds, we can form an approximate picture of the internal structure of a crystal—or at least of such portions of a crystal, well away from boundaries or other disturbing causes, in which truly uniform orientation of the atoms reigns. We see a perfectly orderly regular arrangement of the atoms, each in a position of equilibrium so far as its attachments to its neighbors are concerned, oscillating about the fixed points of the space lattice and, together with the outer electron shells, occupying space quite completely. That is to say that in the case of the more closely packed types of lattice, at all events, there would be no interstitial space comparable in dimension with the space occupied by any single atom. It must, however, be borne in mind that the atoms at best only "occupy" the space defined by their outer electron shells with extreme "thinness." The outer electrons may be moving great rapidity, but they are very minute compared with the radius of the atom. The atom, therefore, occupies its own region of space in much the same way as the solar system occupies the space included in the orbit of Neptune. Just, therefore, as comets are able to wander freely about the inter-planetary spaces, so free electrons such as those carrying electric current should be able to move about and, in a sense, through the atoms. Like the comets, however, they are apt to be seriously deflected if they approach the central nucleus.

Structure of Crystals of Solid-solution Alloys

But our present interest centers upon a type of crystal that is much more complex—that of a solid-solution alloy. Within the organization of the crystal we must find room for two kinds of atoms, which may be called the solvent and the solute atoms, respectively. That the general organization of the crystal remains much the same as that of a pure metal is evident from the fact that, although the properties of solid solutions differ in most important ways from those of the pure metal, the crystalline nature of the material remains strongly marked and closely resembles that of the solvent metal. Thus all the alpha solid solutions based on copper share the tendency of pure copper to undergo twinning. We must, then, find room for the solute atoms in such a way that their presence affects, but does not destroy, the crystalline organization. It is possible to think of the solute atoms introduced into the crystal in several ways. One of the earliest ideas was that the solid-solution crystals were not simple crystals but ultra-microscopically duplex—intimate mixtures of two kinds of crystals built up together as true "mixed" crystals. This view has been credited on metallurgical considerations alone, but recent determinations of the space lattices of a series of solid solutions, made by Owen and Preston at my suggestion, have disproved it, for the solid solution exhibits a single well-defined space lattice that differs only slightly in dimensions from that of the solvent metal.

The second possibility is that the solute atoms find their way into the interstices of the atomic spacing of the parent metal. Such an introduction of additional atoms could occur only in the case of extremely small atoms and then probably only in such lattices as the face-centered cube where there is a certain amount of interstitial space at the body center of the cube. Larger atoms, if thus introduced, would cause an enormous degree of distortion or distension of the parent lattice, and we should expect to find ready solubility in the solid state for the smallest atoms and a systematic decrease in solubility as the size of the solute atom increased. Actually, this is not the case and, in itself, serves to discredit the general correctness of this view, although it may still be possible, in special cases, that small atoms can find a place at such points as the cube centers of a face-centered lattice. Of course, in the case of extremely small atoms, like those of hydrogen, the position is entirely different and "solution" may mean simple penetration into or through the interstices of the space lattice.

In view of these difficulties attaching to the two possible schemes of solid-solution structure just outlined, I have adopted the third possible view: that the solute atoms are placed upon the actual space lattice of the solvent metal in simple substitution for the atoms of the solvent.[1] This view, at first put forward in a somewhat speculative manner, has found extensive confirmation, not only by actual x-ray measurements of solid-solution space lattices, but by the easy and simple manner in which the assumption of such a structure serves to explain many of the most striking properties of solid solutions. With the aid of one or two simple and, in my view, natural conceptions, this view of the structure of metallic solid solutions leads to a somewhat general view of the atomic structure of solids, both vitreous (amorphous) and crystalline, that serves to connect and to explain a wide range of known facts in regard to their behavior and properties.

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[1] Since writing the above, my attention has been called to a paper by Tammann (*Zeitschr. fur Anorg. u. Allgem. Chemie*, July, 1919) which also deals with the arrangement of the atoms of solid solutions on the space lattice. Tammann appears to assume what is here termed the "substitution" theory as self-evident, but his considerations turn mainly on the distribution of the solute atoms on the lattice. From these considerations, he arrives at certain inferences in regard to the solubility of solid-solution crystals in certain reagents and to their electrolytic potential which he has been able to verify experimentally. His views and results are neither in conflict with nor, beyond the assumption named, an anticipation of what is here set forth. The two points of view are, on the contrary, entirely supplementary to one another and mutually helpful.

INFERENCES DRAWN FROM SUBSTITUTION THEORY OF STRUCTURE OF SOLID SOLUTIONS

The substitution of an atom of solute for an atom of solvent in the space lattice of a metallic crystal has great effect on the entire space lattice in the vicinity. If the solute atom differs appreciably in properties from the solvent atom, a state of local strain or dissymmetry must result. This can be inferred from a very simple consideration. Figure 6 (*a*) shows the simplest possible distribution of atoms in one plane of a cubic crystal, where the atoms lie at the corners of squares. In a pure-metal crystal, where all the atoms are exactly alike, the arrangement will be perfectly uniform and symmetrical. Now, suppose that one of the atoms is replaced by a slightly different kind of atom, indicated in (*b*) by a circle, and further suppose that the natural space lattice upon which this atom would fit (*i.e.*, the space lattice of a crystal consisting entirely of that kind of atom) is slightly larger in scale than that of the solvent metal. This would probably mean that the solute atom would tend to allow the adjacent atoms of the solvent to remain farther away than they are in their normal position; in other words, the forces acting upon the adjacent atoms of the solvent and emanating from the atom of solute would be smaller than those emanating from adjacent atoms of the solvent itself. The adjacent atoms of solvent, therefore, will no longer be in an equilibrium position in their normal space-lattice places, and some sort of

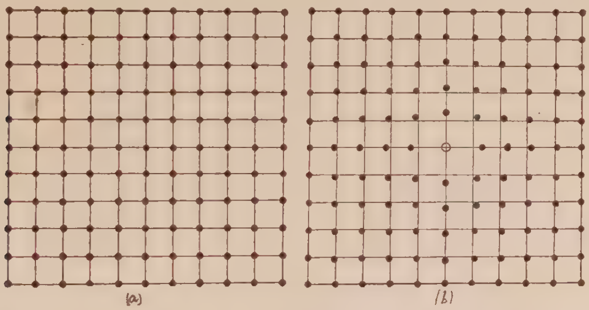

FIG. 6.—POSSIBLE DISTRIBUTION OF ATOMS IN ONE PLANE OF A CUBIC CRYSTAL.

distortion of the lattice is bound to follow. As a first effect, one might suppose some such distortion as that indicated by the dots in (*b*), but such purely local action is quite impossible—the effect must propagate itself through many layers of atoms, producing a slight expansion, tapering off as we pass outwards in all directions from the disturbing atom of solute. None the less, each such disturbing atom must affect a relatively large number of the atoms of the solvent. For instance, if we think of a plain cube-corner lattice and suppose that the disturbing effect is limited to a total range of five atoms on each side of the solute atom, a concentration of only one atom in eight thousand atoms of solvent would be sufficient to affect the entire space lattice, bringing about not only a series of locally relatively intense distortions of the lattice, but also a slight but general expansion of the entire lattice. A concentration of one atomic per cent. might, therefore, well produce a measurable change of dimension in the lattice as a whole in addition to causing more numerous local disturbances.

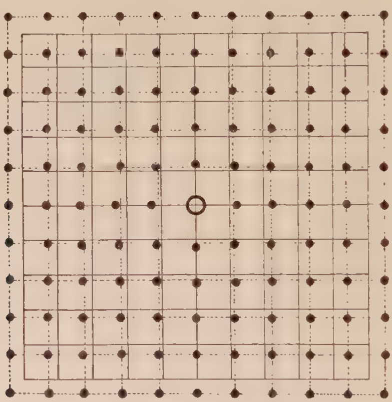

FIG. 7.—GENERAL DISTORTION OF LATTICE.

It is important to note that the relative amounts of what we may term "local" and "general" extension of the lattice will depend on the nature of the lattice itself. If the lattice is comparatively soft or flexible, there is likely to be something of the sort of distortion shown in Fig. 6 (*b*), which is mainly, if not entirely, local. In a stiff lattice, on the other hand, local distortion will be resisted, and therefore diminished, while there will be a correspondingly increased general distortion, somewhat as indicated in Fig. 7, where the whole lattice is shown slightly expanded. This difference in the manner of distortion is of great importance in determining the behavior of various alloys. The cause of stiffness in a lattice is not obvious except that it must be caused by the nature and location of the interatomic bonds. Where these are not only strong but firmly held, as regards direction, the lattice will be stiff. We should expect to find this in the harder and stronger metals having a high elastic modulus.

The question is, how such disturbance of the lattice might be expected to make itself felt in the x-ray spectrum of the crystal? A slight general expansion of the lattice will be measurable when it attains the order of 1 per cent. of the normal lattice spacing; this need not be a uniform extension, the x-ray spectrum will lead to the determination of the average size of the lattice unit. If the local disturbance in the immediate vicinity of a solute atom is not considerably greater than the amplitude of thermal vibration, we could not expect to find any definite effect of such distortion on the sharpness of the spectrum lines obtained with the crystal, so that the only measurable effect would be a slight expansion, or in some cases contraction, of the average lattice spacing.

It may be suggested that on the other view—that the solute atoms are located in the interstices of the solvent lattice—there would also be local distortion and general expansion of the lattice. This is true, but there must be a wide difference in the degree of the expansion of the lattice in the two cases, and a study of the densities of solid-solution alloys together with measurements of their lattice dimensions makes it possible to distinguish critically between the alternatives. It is known that the densities of solid solutions differ only slightly from the values found by taking the mean of the densities of the constituent metals in proportion to their concentration in the alloy. If the solute atoms are supposed to be "pushed in" as it were, without replacement of solvent atoms, the space lattice of the solvent metal must be very much enlarged to compensate for the additional density due to the interpolated solute atoms. If the substitution theory is correct, however, we should have an exact mean density if the parent lattice remained unaffected, and a slight departure from mean density fully accounted for by a slight change in the scale of the parent lattice. Such changes of lattice dimension are very small; for a time it was supposed that they did not exist at all. Recent measurements by Owen and Preston, however, have shown these small changes in the scale of at least one solid-solution lattice and that these correspond with a considerable degree of exactness with what must be expected by the substitution theory. For the alloys so far investigated, therefore, the truth of the substitution theory must be regarded as established, while its wide applicability to most intermetallic solid solutions is extremely probable. Where comparatively small atoms, such as carbon or phosphorus (*i.e.*, metalloid atoms) are concerned, however, the possibility of interstitial location is not excluded and may even be regarded as probable in a few instances.

The data of x-ray analysis bearing upon this point are of such fundamental importance that it is desirable to quote the actual results obtained by Owen and Preston in their x-ray analysis of the alloys of copper and aluminum. In the alloys rich in copper, they found that the spectra of the alloys containing 2.4 and 8 per cent. of aluminum were completely examined; they showed the characteristic features of a face-centered cubic lattice, but the intensities of the reflections fell off markedly as the percentage of aluminum was increased. No new lines were observed in any of these spectra. The results found for the mean value of the side of the elementary cube, in angstroms, are:

Aluminum, Per Cent.	Mean	Calculated Densities	Observed Densities	Difference
	α	ρ'	ρ	
0	3.616	8.878	8.93	−0.05
2	3.635	8.811	8.63	+0.18
4	3.652	8.175	8.33	−0.15
6	3.655	7.952	8.04	−0.09
8	3.663	7.717	7.74	−0.03

There is, thus, a steady slight increase in the lattice dimensions. If the values of α thus measured are the mean values in a slightly distorted lattice, the densities of the series of alloys can be calculated, inserting a value for the "mean atomic weight" by taking the means of the atomic weights of copper and aluminum in proportion to the atomic concentration of each alloy. The values thus obtained are stated in the column headed ρ'; the observed densities of the actual alloy specimens are given in the column headed ρ. The differences between observed and calculated values, given in the column headed difference, are small. It must be borne in mind that in the density calculated from x-ray data the errors of experimental measurement are trebled. The differences found are, therefore, readily accounted for by errors of ½ per cent. in the measurement of the x-ray angles. These results are shown graphically in Figs. 8 and 9, where the values of α, ρ, and ρ' are plotted against aluminum

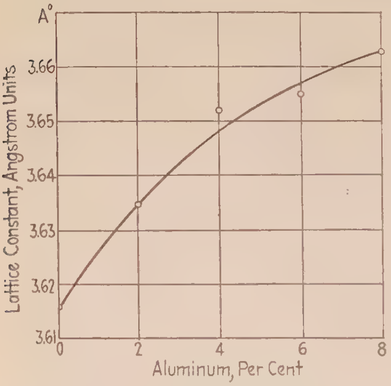

FIG. 8.—VARIATION IN LATTICE CONSTANT IN COPPER-ALUMINUM SOLID SOLUTIONS. (*Owen and Preston.*)

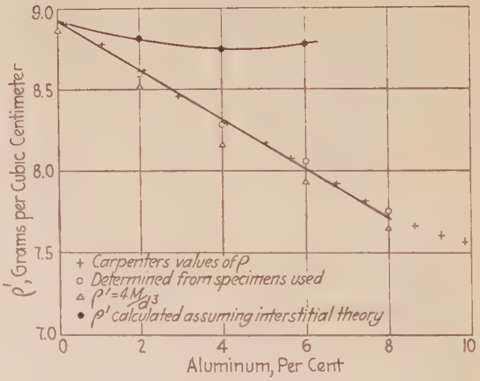

FIG. 9.—OBSERVED AND CALCULATED DENSITIES OF COPPER-ALUMINUM SOLID SOLUTIONS. (*Owen and Preston.*)

concentration. The values of densities found with the foregoing data as to lattice size, but based on the assumption that the aluminum atoms are present not in substitution for but in addition to the atoms on the copper space lattice, are also plotted. The comparison shows that the observations entirely contradict the latter assumption while they are well within the limits of experimental error, in accordance with the substitution theory.

In the alloys rich in aluminum, similar measurements were made on alloys containing up to 2 per cent. of copper. In these, the lattice constant was found to be the same, within the errors of experiment, no definite evidence of any expansion of the lattice being obtained. The densities calculated from the value found, however, agree closely with the observed densities. The maximum atomic concentration, though, is of the order of 1 per cent., while at the other end of the series the maximum concentration was as much as 16 atomic per cent. It is not surprising, therefore, that the effects at the aluminum end should be much smaller and apparently beyond the limits of accuracy of the method employed.

If we accept the fundamental conception that in metallic solid solutions the solute atoms replace atoms of the solvent on the normal space lattice of the latter, and that a local distortion as well as a general change of average lattice scale accompanies this replacement, consider a number of interesting inferences which follow in a simple and direct manner. The first of these relates to the limit of solid solubility of one metal in another.[2]

Storage of Energy within Crystals

We have good reason for supposing that the energy content of a solid solution must be greater than that of the simple solvent metal and are in a position to see where the potential energy content of the solid solution is stored; it is obviously in the distortion and change of dimensions of the lattice. Where there is a general expansion of the lattice there must be an artificial increase in the normal interatomic distance; an increase that, on the average, may be slight but is none the less important locally. The strength of the interatomic bonds is great and a small displacement of atoms against the action of these interatomic forces implies the storage of a large amount of energy. Indeed, the distortion of a lattice must mean a corresponding, although slight, distortion of the atoms themselves. Whether we think of stationary electrons or of electron-orbits, their position must be intimately related to the interatomic bonds; and if these are displaced so must the electrons or their orbits be displaced. It is in this slight internal change of form of the atom itself that we must seek the real locus of energy storage. It follows, however, that there must be a limit to the amount of energy that can be stored in this way without disruption of the lattice.

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[2] Rosenhain: The Hardness of Solid Solutions. *Proc.* Roy. Soc. London (1921) **99.** A.

There are several ways of arriving at this conclusion. We know that the amount of thermal energy that can be stored in a crystalline array of atoms is limited and that if this amount is exceeded, melting occurs. One way of looking at this fact is that when the atoms are forced apart beyond a certain definite distance, their mutual bonds cannot hold them together and the lattice breaks up; this way of regarding fusion is of considerable importance, but for the moment it suggests that the distortion of a lattice is limited in amount. Consequently, the number of solute atoms that can be introduced into the space lattice of a solvent must be limited, but this limit will depend on the amount of distortion that the introduction of each atom produces. We find, therefore, a ready explanation for the fact that the solid solubility of metals in one another varies widely: the more similar the atoms of the solute are to those of the solvent, the greater the degree of solid solubility; where the similarity is very great, there is an unbroken series of solid solutions in the alloys of the two metals, but with increasing dissimilarity there are correspondingly smaller degrees of solid solubility. The only difficulty is, with our present knowledge, to decide what are the criteria of "similarity" between atoms of different metals. One of the most important factors must, if our present view is correct, be the normal size and type of space lattice upon which pure crystals of each metal are built. Where these are closely alike in scale and arrangement, we shall expect to find large solubility; and vice versa. A glance at the known facts will confirm this conclusion. Copper normally forms a face-centered lattice, the constant (size of unit) of which is 3.60 A; the constant of nickel is 3.54; between these two metals there is an unbroken series of solid solutions. The same thing applies to gold which has a lattice constant of 4.08, and silver, which has a lattice constant of 4.06. On the other hand, copper (3.60) and silver (4.06) show only a limited range of solid solutions at either end of the series. Copper and zinc (2.67) show a more limited degree of solid solubility at one end of the system, where the smaller atom replaces the larger, and practically no solid solubility at all at the other end, where the replacement would occur in the opposite way. Copper and aluminum (4.05) show very limited solid solubility at both ends, while copper and tin (6.46) show still more limited solubility at the copper end of the system. It is clear, however, when the details of these systems are examined, that the lattice constant is by no means the only governing factor, nor would we anticipate that this should be the case. What one might term, tentatively, the "shape" of the atom and perhaps the facility with which its bonds can adapt themselves to a slightly different configuration must play a very important part. It is not possible to pursue this aspect of the matter further, and we must content ourselves with the fact that the amount of energy stored in the composite lattice determines the limit of solid solubility and that this is dependent on the amount of distortion which the introduction of a solute atom produces.

EFFECT OF DISTORTION ON HARDNESS, STRENGTH, AND DUCTILITY

The amount of distortion that a lattice undergoes, however, will determine the extent to which certain properties of the crystal are affected. Perhaps the most interesting of these is its hardness or resistance to plastic deformation. It is generally accepted that plastic deformation in a metallic crystal occurs by a process of slip, which takes place on some of the crystallographic planes of the crystal. This means that when external stress is applied to a crystal there is an elastic (and slight) distortion of the lattice; when this has reached the limit of the atomic bonds, slip occurs and layers of the crystal slide over one another. Such sliding will be most readily brought about in a perfectly regular and symmetrical crystal with perfectly uniform atomic spacings. The layers of atoms lying on the two sides of a slip plane may then, in a general way, be regarded as two perfectly smooth surfaces sliding over one another; at all events, for a given strength of atomic bond, the resistance to slip will be least in such an arrangement. If, however, the arrangement is distorted in the least degree, there will be a corresponding increase in resistance to slip. Thus a slightly distorted lattice must imply an increased degree of hardness and strength and also a diminished degree of ductility.

At this point, we arrive at an inference that is directly amenable to verification. If the power of a solute atom to cause distortion of the lattice of the solvent is closely related to the hardening effect it produces, and this same power of producing distortion also determines the limiting solid solubility of one metal in another, we must expect to find that the hardening effect of one metal on another in the form of a solid solution is closely and inversely related to its limiting solid solubility. The greater the range of solid solutions formed, the less should be the hardening effect of the added metal per atom added.

An examination of the well-studied alloy systems in which the limits of solid solubility have been reliably determined will show that this generalization holds good. If the copper alloys are taken, the hardening effect, within the range of solid solubility in each case, of alloying metals may be placed in the following order: nickel, manganese, zinc, aluminum, tin. The limits of solid solubility, in atomic per cent., for these metals are nickel, 100; manganese, 100; zinc, 36; aluminum, 14; tin, 67. In the case of nickel and manganese, although there is an unbroken series of solid solutions in both cases, the shape of the liquidus curve of the copper-manganese system indicates that the formation of a eutectic near the middle of the series is only just avoided; an alteration of pressure, for instance, might easily result in the formation of two separate phases—probably a eutectic. The solid solubility of nickel in copper may, therefore, be fairly regarded as being greater than that of manganese. A similar rule holds for the alloys of aluminum; zinc forms a much longer range of solid solutions with aluminum (up to 15 atomic per cent.) than does copper (up to 1.5 atomic per cent.), and their hardening effect is in the inverse order. In the alloys of iron, in so far as they have been studied apart from the effect of carbon, a similar law holds good.

The qualitative verification of the inference drawn from our theoretical conceptions, therefore, is good; so good that the inference may be regarded as a new general principle governing the properties of solid solutions, a principle that had not been recognized until the present considerations brought it to light. The law or principle is rather more than qualitative, as it serves to group alloy systems in a rough quantitative order. Any accurate quantitative verification is difficult at present. We do not possess data concerning even the average expansion or contraction of solid-solution space lattices except for the one system for which preliminary figures have been given. Even if these were available, we would not know the degree of local distortion produced by the solute atoms. Further, we are not justified in assuming that successive additions of solute atoms to a solid solution produce equal increments of hardness; in fact, the reverse is known to be the case. From the point of view of the substitution theory, this is to be expected as an increase in the number of solute atoms would result in a greater average change of the lattice size, but need not, and probably would not, increase the degree of local distortion in the lattice. It may be that, with a fuller knowledge of the atomic forces involved in these complex relations, a quantitative formula for the relation between internal energy, lattice distortion and hardness may be worked out. As yet we seem to be some way from that stage, so that the rough quantitative verification of our inference is as much as we can hope to attain.

Reduction of Melting Point

A property of solid solutions, even more typical, perhaps, than hardness, which also finds ready explanation on the basis of the "substitution" theory is the lowering of the melting point and the spreading of melting and freezing over a range of temperature. This difference is indicated in the equilibrium diagrams of Figs. 2, 3, and 4. To arrive at the explanation which our theory affords for these phenomena, let us consider a crystal of a pure metal in which the atoms are arranged in perfect regularity upon the normal space lattice. With thermal expansion there is, inevitably, a corresponding expansion of the lattice. This has been observed, although in certain cases the lattice does not expand equally in all crystallographic directions; in the case of graphite there is a slight contraction of the lattice in one direction compensated by a correspondingly greater expansion in another. In a cubic or close-packed hexagonal lattice, however, there is no reason to suspect this and the fact that Westgren has found the characteristic lattice of gamma iron at 1100° C. to be a face-centered cube, while delta iron at 1450° C. shows the typical diffraction lines of a body-centered cubic lattice, provides direct evidence that in this metal there is no serious distortion of the symmetry of a cubic lattice by very considerable thermal expansion. We may, then, assume that the cubic metals undergo a regular and symmetrical lattice expansion proportional to their mass-coefficient of expansion.

We return, now, to the conception that for every lattice there is a maximum distortion (in most cases an extension) that cannot be exceeded without causing the lattice to break down with the resultant formation of a new phase. We cannot, however, assume that the actual maximum extension that can occur is independent of the nature of the second phase which results from a breakdown of the lattice. Considerations to be discussed later, make it certain that the maximum lattice extension is considerably dependent on the nature of the alternative structure that arises when that maximum is exceeded. With this reservation in mind, it is possible to derive from the "substitution" theory of solid solutions a complete explanation of the behavior of these bodies on melting and freezing.

A crystal of a pure metal possesses a perfectly uniform lattice, which expands uniformly with rising temperature, at all events in the case of cubic lattices. The ground for the latter assertion is that a number of lattices of this kind have been studied at quite high temperatures; for instance, the lattices of gamma and delta iron at 1200° and 1450° C., respectively, by Westgren, which, at those high temperatures, have shown the characteristics of a cubic lattice—an observation that disposes of any serious degree of unequal expansion in different directions in the lattice. In the case of graphite, it has been found that the expansion in one direction is much greater than that to be anticipated from the mass-coefficient of expansion; while in another direction there is a small contraction. In view of the other observations, however, this seems to be a special case and not to be applicable to cubic lattices. The result of uniform thermal expansion on a simple regular lattice must, therefore, be that at a definite temperature the whole of the lattice attains the limit of its extensibility; a further rise of temperature leads to the breakdown of the lattice and the formation of liquid—the whole of the metal melts at one uniform temperature. This conception of melting is in accord with the well-known fact that a high melting point is definitely associated with a low coefficient of expansion, while the harder and stronger metals are, in general, those with high melting points.

On the basis of the present theory, a crystal of solid solution possesses a lattice that is slightly extended throughout and, in addition, in the neighborhood of solute atoms, there are regions of local and more marked extension. If such a crystal is heated, the locally distended regions around solute atoms will reach the limit of lattice extension first, and melting will begin at those places at a temperature at which the bulk of the lattice is still stable. We thus have local melting with the formation of a liquid whose content of solute is higher than the average of the whole crystal. As the temperature rises, other regions of less distorted lattice will melt, until the whole crystal has melted. The highest temperature required to bring this about will be lower than in the case of the pure metal, because the entire lattice is extended and therefore reaches its limiting extension after a smaller thermal expansion. The limiting extension, however, is not constant throughout a series of solid solutions, as it must depend on the concentration of the liquid formed. The mechanism of this dependence, which can be accounted for in terms of of energy content of the two phases, is probably connected with the internal, or rather the osmotic, pressure of the liquid solution formed. This pressure may act by restraining the expansion of the lattice under thermal expansion, but that point cannot be easily tested. The reality of the effect, however, is undoubted.

From the preceding, it is possible to draw interesting inferences as to the behavior of solid solutions on melting and freezing, as the freezing operation is exactly the reverse of the melting process as described. In a solid solution, two lines in the equilibrium diagram, known as the solidus and liquidus, determine the beginning and the completion of melting. On the basis of the substitution theory, the beginning of melting (the position of the solidus) is determined by the maximum local extension produced in the lattice by the presence of solute atoms; while the completion of melting (the liquidus) depends essentially on the general expansion of the lattice. If this is correct, we should find that where local distortion is large and general extension small, the solidus would fall rapidly with increasing concentration of the solid solution, while the liquidus will be much less steeply sloped. Further, the slope of the solidus will be increasingly steep the lower the solid solubility of the solute in the solvent metal. This latter point is borne out by the well-established equilibrium diagrams. Wherever there is a long range of solid solutions, the solidus falls slowly and does not lie far below the liquidus; on the other hand where solubility is low the solidus may be nearly vertical while the slope of the liquidus is often relatively gentle.

A slight local distortion of the lattice is more likely to occur in the harder and stronger metals of high melting point, as the lattice is much more rigid and the presence of a disturbing solute atom causes little local and not very much general distortion. This consideration connects strength and hardness, as well as high melting point, with the tendency to form long ranges of solid solutions. This connection, once pointed out, seems so obvious that I am surprised it has not been noticed before. The high-melting-point metals, such as iron, nickel, cobalt, chromium, copper, silver, gold, are known to form alloy systems mainly consisting of solid solutions—it is among them only that we find series showing an unbroken line of solid solutions between pairs of metals in all proportions. Metals having lower melting points, like zinc and aluminum, form decidedly more limited series of solid solutions; while the soft metals having a low melting point, like lead, tin, cadmium, etc., show alloy systems that tend to approach the entirely eutectiferous type arising from very slight solid solubility. In these two general and fundamental ways, therefore, the theory here advanced explains satisfactorily well-ascertained facts. The accordance is greatly strengthened by consideration of what, at first sight, appears to be an anomaly.

In the preceding regarding melting and freezing, the argument was based on the supposition that the space lattice of the solvent metal is, on the average, enlarged by the addition of the solute. It is equally important to determine how the matter stands when the lattice of the solid solution is slightly smaller in scale than that of the pure solvent. The preceding line of reasoning is applicable but in the inverse sense; the melting point of the alloy should be higher than that of the pure metal and during the freezing process the first portions to crystallize will be the richest in solute. This conclusion is in striking agreement with those known equilibrium diagrams of alloy systems in which it is quite certain, even before actual *x*-ray measurements have been made, that the introduction of the solute will lead to a slight contraction of the lattice. One example of this kind is furnished by the alloys of copper with nickel. In this system, there is a well authenticated series of solid solutions extending from pure copper to pure nickel. The lattice constant of nickel is slightly smaller (2.50) than that of copper (2.54), and as there is a perfectly continuous change in melting point and other properties from one end of the series to the other, the lattice constant must change gradually from the larger value for pure copper to the smaller value for pure nickel. This steady, though slight, contraction is accompanied by a continuous rise in melting temperature, while the shape of the equilibrium diagram shows that even near the copper end of the series the portions that solidify first have a higher nickel content than the liquid from which they form. A precisely similar case is presented by the alloys of molybdenum and tungsten, but such cases, where the melting point rises as the result of solid-solution formation, are not numerous.[3]

Another point that arises from the picture of the melting and freezing processes which we derive from our theory is that there must be an intimate relation between the extent to which the freezing point of a solvent metal is depressed by the addition of solute and the amount of lattice distortion that this addition brings about.

Considerations of this kind (those relating to the depression of the freezing temperature and the width of the range between liquidus and solidus) suggest the possibility of accurate quantitative calculation, provided that the atomic constants concerned were known. The normal lattice constant of an atom is but one of these factors and the actual strength of the interatomic bonds, their maximum effective range, and the work done in either breaking them or displacing them by a definite small amount enter into the problem. To attack the matter from this side, additional data regarding the atoms are needed. On the other hand, the phenomena of freezing, also others that have been referred to, lend themselves to accurate measurement and it may be possible to reverse the process, in a few of the simpler cases, in order to utilize the data obtained from solid solutions to determine the constants of the atoms. Such an attack would prove valuable to the physicist and the metallurgist. For the latter, it opens the prospect of being able, ultimately, to predict by calculation the entire properties of an alloy system.

Another phenomenon that requires explanation is the occurrence of diffusion within the crystals of a solid solution. All metallurgists know that the crystals formed during the freezing of a solid-solution alloy at any ordinary rate are by no means homogeneous in composition. When suitably etched, they show a well-marked cored structure sometimes so strongly marked that it is difficult to realize that one is not dealing with a duplex alloy. Yet, when an alloy containing such definitely cored crystals is suitably annealed, the coring vanishes and the structure resembles, in every way, that of a pure metal. This change implies a redistribution of the solute atoms that, at solidification, are considerably concentrated in the outer layers of each crystal. This redistribution occurs without anything like an approach to fusion; by what mechanism can it occur?

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[3] After the completion of the text of this lecture, a paper by McKeehan (*Phys. Review*, 1922) on the x-ray analysis of the alloys of silver-gold and silver-palladium was received. The data for the silver-palladium system are a striking confirmation of the views here given. This system consists of an unbroken series of solid solutions between two metals differing appreciably in melting point and lattice constant. McKeehan's results show a steady decrease of lattice constant with increasing additions of palladium to silver; this is accompanied, exactly as in the case of copper-nickel, by rising melting point and an appreciable divergence of solidus from liquidus in the manner required by my theory. Measurement has, in this case, clearly demonstrated precisely what is predicted for a similar system (copper-nickel). In the gold-silver system, the difference of lattice constant and other factors between the two kinds of atoms is extremely small; there is little difference in melting points between the metals and there is little departure of solidus from liquidus. This is in accord with the approximate constancy of the lattice constant throughout the system, as found by McKeehan. Certain abnormalities, however, are found by him in some of the alloys, which show a slightly larger lattice constant. The explanation of the present theory is probably to be sought in the existence of a certain small degree of chemical affinity between the two metals; in the absence of lattice distortions, these may make themselves felt to a greater extent than in other types of alloy systems.

It is, first of all, important to realize that the diffusion whereby the crystals become homogeneous can, and does, take place without anything like a complete recrystallization of the alloy. It is known that, in favorable circumstances, if an alloy is heated to a suitable temperature for a sufficiently long time complete rearrangement of the crystals may occur. In order to test whether the tendency toward diffusion is enough to bring about such crystalline rearrangement, some experiments on a variety of alloys have been carried out, at my suggestion, by F. Adcock. He examined carefully marked regions on a polished and etched specimen and photographed the crystal outlines, as well as the dendritic cores, before and after annealing. He found that the crystal boundaries undergo very slight changes, of such a nature as to smooth out minor irregularities and sharp curvatures, but that there is, in cast solid solutions (mainly of alloys of copper) no tendency to general recrystallization. The small boundary changes are interesting, particularly as there is strong evidence that they occur during the first cooling of the alloy, long before the dendritic cores have begun to disappear. On the other hand, these observations make it clear that the mechanism of diffusion must exist within the organization of each crystal and that it is not dependent on complete rearrangement with its probable temporary passage through an amorphous condition.

With a crystal, however, we must think of the atoms as fixed upon or vibrating about fixed points on a space lattice, with only small interstices between the atoms or rather between their outer electron shells. In such a system, there can be no room for wandering atoms of the same order of dimensions as those upon the space lattice. Fortunately, our substitution theory at once suggests a simple and an adequate explanation.

At ordinary temperatures, it is possible to bring about slip within the crystals of most metals and solid solutions by the application of external stress. The intensity of the stress required to produce slip decreases very rapidly with rising temperature; *i.e.*, metals become very soft when hot. But the stress required to bring about plastic deformation (slip) in a crystalline aggregate is larger than that which would be needed in a single crystal, as there is no doubt as to the stiffening and strengthening effect of the numerous crystal boundaries. Further, it is only a crystal here and there that is most favorably oriented in respect of the particular external stress applied. It follows that the stress required to produce slip within a single crystal at high temperatures is not very great.

The essence of the substitution theory of solid-solution structure is that the presence of a solute atom in a space lattice produces somewhat intense stresses in its neighborhood. At a high temperature, if such stress can relieve itself by the occurrence of internal slip, one would expect such slip to occur. But here, all that is necessary is a slip of one atomic step—a slip about 0.002 as large as those that occur during plastic yielding. Under external stresses, the atoms generally slip in layers or planes; under the internal stress caused by a solute, only a single row or line of atoms need move at one time. If the solute atoms are uniformly distributed through the crystal lattice, the resulting stress will be balanced and no slip need occur or the amount of slip occurring in different directions will be equal and opposite, so that the net result on the distribution of the solute atoms will be nil. On the other hand, if the concentration is not uniform, there will be more slipping in the direction leading toward the region of low concentration than in any other. Slips of one step at a time, taking place at intervals in the various principal planes of the lattice, will be sufficient to account completely for diffusion and it is possible by a suitable number of successive slips of this nature to carry a solute atom from any one position in the lattice to any other.

The question, "what happens to the atom at the end of a row when slip occurs?" opens the question of crystal boundaries, which I have avoided although it is readily treated on the lines followed in our conception of the crystal structure itself. I need only say that the end atom, which will be pushed out when slip occurs, probably passes into the more or less non-crystalline or amorphous layer present at the boundary; while at the other end of the row, room is made for the entry of an atom passing into the lattice from the intercrystal layer. As there is reason to suppose that, particularly at high temperatures, such an interchange of atoms between crystal lattice and intercrystal layer is constantly in progress, our view of the mechanism of diffusion within the crystal requires no fresh assumption.

It does not seem easy to find any ready means for testing the probability of this suggested mechanism for intercrystalline diffusion. At first sight, it would seem that solute atoms which cause the greatest amount of distortion in the solvent lattice should exhibit the most rapid rate of diffusion, for presumably the stresses in the lattice tending to cause slip would be greatest for such atoms. It is true that the resulting distortion of the lattice would tend to hinder general slip under externally applied stresses, but the extremely local slip required for diffusion would occur in such a way as to relieve distortion and would not be hindered in that way. It is, however, difficult to determine the relative rates of diffusion of different kinds of solute atoms in the same solvent lattice, partly because it is not practicable to determine the initial concentration gradient in a cored crystal nor the precise moment when that gradient has either disappeared or fallen to a known lower value. We can, therefore, form nothing more than vague estimates of the rapidity with which diffusion occurs in different solid-solution alloys. In this connection, we may recall the extremely slow rate of diffusion in the copper-nickel alloys, in which the lattice distortion is very slight. On the other hand, carbon in gamma iron is known to diffuse with great rapidity and its solid solubility is by no means high. However, we have the equally well established fact that phosphorus in solid solution in iron only diffuses with great difficulty, yet its solid solubility is by no means high and its hardening effect considerable.

A less well-defined case is that of zinc in copper, which is thought to diffuse more rapidly than tin in the same solvent; in all these cases, however, there are no quantitative data and it is by no means certain that lattice distortion is the only, or even the most powerful factor affecting diffusion by the mechanism suggested. The facility with which the interatomic bonds can undergo slight displacement, thus enabling an atom in a slipping layer, or row, to transfer its bonds from one atom to the next in the adjoining row, must play a most important part. It seems quite possible that such slipping may not be the simple geometrical process pictured, but that the transfer of interatomic bonds involves a certain degree of temporary dislocation of the whole lattice structure in its neighborhood. In the case of carbon and phosphorus, we are dealing with metalloid elements the interatomic attachment of which to the metal atoms may be of quite a different nature from that between two metal atoms. Sir William Bragg has definitely suggested that in the solid solution of carbon in gamma iron, the carbon atoms are situated at the body centers of the face-centered cubic lattice. The evidence for this view is by no means strong; indeed, the latest view of Westgren, to the effect that the gamma-iron lattice is not appreciably expanded by the introduction of carbon, seems to indicate that here we must be dealing with a substitution structure, for otherwise the density would increase much too rapidly with the addition of carbon. It is none the less possible that smaller atoms, such as those of carbon or phosphorus, may be able to find room in such lattice interstices as that indicated. In the case of phosphorus, although measurements are not available, one fact that points in that direction is the extreme slowness with which phosphorus diffuses in iron. If the phosphorus atom is located at the cube body centers, diffusion by slipping would require that the row of atoms in which a phosphorus atom is present would move in such a way that the phosphorus atom would pass from one body center to the next, and in doing so the phosphorus atom would have to pass through the normal position of one iron atom in the cube, while a number of iron atoms would have to pass through the body-center position, and this would be almost impossible. In such a case, diffusion within a stable crystal might be impossible and the process could occur only during a recrystallization process. We have here a point that might be tested experimentally, but the difficulties to be overcome are considerable.

Solid Solutions Containing Several Kinds of Solute Atoms

Solid solutions in which several kinds of solute atoms are present simultaneously are well known. If the substitution and lattice-distortion theory is correct, the presence of one kind of solute atom must affect the limiting solubility of the other. If the two kinds of atoms produce similar kinds of distortion in the solvent lattice, they will mutually tend to diminish solubility; the atom that causes the lesser degree of distortion will tend to throw the other out of solid solution. The limiting solubilities are a matter of the balance of energy content as between a more severely distorted solution lattice or a duplex system of two distinct phases.

The study of ternary alloy systems at the National Physical Laboratory by Doctor Hanson, Doctor Haughton, Miss Gayler, and Miss Bingham has brought to light a number of cases where the introduction of one metal has seriously diminished the solubility of another, magnesium and copper in aluminum being a good example. Magnesium and copper have smaller lattice constants than aluminum, but copper (3.60) is nearer aluminum (4.05) than is magnesium (3.22); while one would, on this basis, expect these two metals to interfere with the solubility of one another, copper would be expected to win and the solubility of magnesium to be reduced. Other complicating factors intervene, however, for the solubility of magnesium in aluminum (about 9 atomic per cent.) is much higher than that of copper (2 atomic per cent.), in accordance with which fact, magnesium reduces the solid solubility of copper in aluminum. This may be because the coefficient of thermal expansion of magnesium is larger in the ratio of 25 to 17 than that of copper, so that at the high temperatures where solid solutions are formed, the difference in lattice size between the two metals may be much less than at the ordinary temperature. Magnesium may be present in the solid solution in a form slightly different from that in which it forms its own hexagonal lattice, so that its effective lattice constant may be much closer to that of aluminum than the figures quoted would suggest.

Another interesting example of incompatibility in solid solution is that of carbon and phosphorus in gamma iron. There can be no doubt that carbon, which otherwise diffuses with great readiness through hot gamma iron, persistently refuses to enter the high-phosphorus regions that originate from the coring of the iron-phosphorus solid solution. Phosphorus is distinctly more soluble in gamma iron than is carbon, but the manner in which phosphorus enters the gamma-iron space lattice may prove the determining factor in this case.

One more interesting point, although its complete explanation is not plain, is the influence of dissolved impurities on the magnetic properties of iron. Most of such impurities tend to "harden" iron magnetically, particularly increasing the hysteresis loss. Ewing recently put forward a new model of atomic structure that serves to account for the observed facts of magnetism in a complete manner. In this model he assumes the atom is slightly unsymmetrical in certain respects. Any distortion of the lattice into which such an atom is built must bring with it a corresponding distortion of the atom itself, however slight. Such a slight further distortion of the atom, however, according to Ewing, would serve to account for the marked effect on magnetic properties of both added elements (impurities) and mechanical deformation. There are at least two apparent marked exceptions to the rule in regard to the effect of added elements, for the addition of silicon and aluminum to iron improves it magnetically in regard to hysteresis loss. The present view would readily explain this action if it were found that the impurities always present in ordinary iron tend to produce one kind of distortion in the lattice while the two elements named tend to produce an opposite, and therefore neutralizing, kind of distortion. This, however, opens a field of inquiry I do not propose to enter at this time.

NATURE AND BEHAVIOR OF INTERMETALLIC COMPOUNDS

The nature of intermetallic compounds and their behavior, both as solutes and solvents, is a wide subject and I can only hope to touch its fringes. A good deal depends on the definition we adopt for intermetallic compound. If we define it merely as a homogeneous crystalline alloy in which the two metals are present in such a ratio as to conform to the law of multiple proportions, it is practically impossible to distinguish between solid solutions of certain compositions and compounds. On the other hand, the study of alloy systems has revealed the existence of a number of bodies possessing well-defined characteristic properties and located in the equilibrium diagram in one of a few special ways, which we must certainly regard as definite compounds possessing a nature essentially different from that of solid solutions. Unfortunately, direct recourse to results of x-ray analysis fails us here, for there are no published data giving the lattice structure of a well-defined intermetallic compound.

We might not be on safe ground if we argue from analogy with ordinary crystalline compounds of inorganic chemistry, such as chlorides and fluorides, for several of which the lattice structures have been worked out, were it not that in the case of at least one typical intermetallic compound $CuAl_2$, a characteristic x-ray spectrum has been worked out by Owen and Preston. The interpretation of this spectrum is difficult and a corresponding lattice structure has not been worked out, but the fact that this compound exhibits a spectrum totally different from that of either copper or aluminum strongly supports the view that this compound resembles such bodies as chlorides, etc., in having a more complex lattice structure than a pure metal. By analogy, we may, therefore, expect intermetallic compounds to possess, like inorganic compounds, a lattice structure that may be regarded as consisting of two or more interpenetrating lattices, in which the arrangement of the atoms is much less symmetrical than in the regular pure metals.

By the substitution theory, a solid solution cannot be regarded as having anything but a single lattice, that of the solvent metal, and here we have an important and critical distinction. This distinction is reflected in one of the most striking properties of compounds, their brittleness or inability to undergo plastic deformation by slip. When one finds this brittleness in compounds of pairs of such ductile metals as gold and aluminum, copper-aluminum, copper-tin, etc., a fundamental difference of internal structure is suggested; a difference not shared by solid solutions that, though harder and less ductile than the solvent metals, are far from being really brittle. On the other hand there are pure metals, like antimony and bismuth, that are quite as brittle as typical compounds.

There is an evident common explanation for these striking differences. Plastic deformation occurs by slip within the lattice, and such slip, if it is to occur without loss of cohesion, must be accompanied by a rapid "handing on" of atomic bond to atomic bond as the moving atoms slide past one another. The explanation suggested is that, where the atoms are evenly, or almost evenly spaced, such handing on will occur easily, particularly if the lattice is not initially unduly strained. On the other hand, if the atomic spacing is uneven, any handing on process would require a rapid readjustment of the atomic bonds in the act of transfer from one atom to another placed farther away or out of the straight line of motion. It may be, that the maximum distance over which an interatomic bond can be stretched is exceeded when movement begins. In either case slip without loss of cohesion cannot occur unless the two slipping surfaces are held in contact by powerful external pressure long enough for fresh bonding to occur in any new position that rows of atoms may assume. Stressed in the ordinary way, such a crystal would be brittle, but under a heavy external pressure it might prove ductile.

Such uneven interatomic spacing must occur whenever we have a lattice structure of lower symmetry; consequently, it is in accordance with expectations that the metals whose structures show the highest degree of symmetry, those having a cubic lattice, are by far the most ductile. The hexagonal group is markedly less ductile (zinc being a typical example). Those having still lower symmetry, such as bismuth and antimony, are brittle in the ordinary way but can flow under heavy hydrostatic pressure, as in extrusion. Intermetallic compounds, and indeed most inorganic crystalline compounds, are typically brittle, although some of the latter are also capable of flow under pressure. Certain exceptions to this rule may be found both in metals and other substances, for the reason that the structure resulting from the interlaced lattices of a compound may quite closely approach the symmetrical lattice of a simple substance if the two component lattices are very similar and are symmetrically situated. Such a lattice as that of potassium chloride might be cited as an example, while in alloys the compounds of copper and zinc (in so far as they are true compounds at all) are much less brittle than many others.

BEHAVIOR OF INTERMETALLIC COMPOUNDS

Can intermetallic compounds enter into solid solutions as compounds, that is, in molecular association? All the considerations discussed here point to the view that, when the solvent is one of the constituents of the compound, we cannot think of the compound going into solid solution in molecular association. Such a molecule as $CuAl_2$, for instance, would be too large to find room upon the space lattice of aluminum. If it did exist in the solid solution, even in only molecular aggregation, it would exhibit a typical diffraction spectrum under the x-rays and no such spectrum is found. The x-ray spectrum of the solid solution of copper in aluminum is substantially the same as that of pure aluminum even when 4 per cent. of copper is present. We must conclude, therefore, that in the solid solution we deal with individual copper atoms and that the compound $CuAl_2$ is formed from groups of atoms at the moment when it comes into existence as a separate crystalline phase. It is not at all certain that this conclusion holds in the case of all compounds, although where the solvent metal enters into the compound one cannot accept the idea of so relatively large a thing as the compound molecule occupying the space ordinarily assigned to any one of its constituent atoms. Where the compound does not contain an atom of the solvent material, a different case may arise; the molecule of the compound may be better fitted to occupy a place on the lattice than its atomic constituents. Whether or not such cases actually exist remains to be determined. In most cases, we must regard the compound as dissociated when in solid solution. But this dissociation may not be similar to that which, in some cases, occurs in liquid solutions. In the compound $CuAl_2$, it is possible to regard each copper atom as in some way specially attached to two of the adjacent aluminum atoms on the space lattice: there may be some difference in the manner of the interatomic linking in these cases, which may bring with it some special disturbance of the lattice structure. These are details upon which it is not very fruitful to speculate, but the possibilities must be borne in mind if false conclusions are to be avoided.

Intermetallic compounds quite well defined in character may act as solvents in the formation of solid solutions. The cases so far studied relate to solid solutions in which one of the component metals of the compound is the solute. Here we meet with a curious consideration. If the substitution theory holds in these cases, in a compound AB of metals A and B, metal A passing into solid solution in the compound obviously cannot replace atoms of A already present on the compound lattice and must therefore replace atoms of B. The atomic similarity of A to B must therefore govern the solubility of A in the compound AB as it governs its solubility in B itself. The degree of solubility is not likely to be quantitatively identical, for the behavior of B in the compound lattice is likely to be different from that of B on its own space lattice, but we should expect to find a distinct correspondence between the solubility of a metal in the pure metal at the extreme end of a binary series and in an intermediate compound. A much fuller knowledge of limiting solubilities in the solid than we possess is required to allow this generalization to be tested. In a few cases it appears to be correct.

The question arises whether all solid solutions in metallic alloys have as their basis, or solvent lattice, either the normal lattice of a pure metal or of a definite compound. While it seems probable that this is so in most cases there is no reason to suppose that other possibilities do not exist. From our consideration of the behavior of atoms arranged on a space lattice we arrived at the general view that when a lattice becomes distorted or expanded beyond a certain definite point, it tends to break up, with the formation of a second phase—this may be a liquid, as in melting, an amorphous solid, or congealed liquid, as in plastic deformation, or a second solid phase, when the limit of solid solubility is passed. But it does not follow that the second solid phase must be either a saturated solid solution of the second metal or an intermetallic compound. It is conceivable that the atoms of the solvent metal may be capable of assuming an arrangement on a lattice slightly different from that which is normal to the pure metal. In that case the alternative lattice is probably less stable, *i.e.*, contains more potential energy, than the usual lattice when the metal is pure. It is not difficult to imagine that in the presence of a number of solute atoms sufficient to strain the usual lattice to its limit, the second or alternative lattice becomes more stable, that is, capable of accommodating a larger proportion of solute atoms before disruption occurs. If this is the case, the second phase will still be a solid solution of the solute in the original solvent metal but based on a different space lattice. The solvent metal may be regarded as taking up this rather less stable arrangement by the inducement due to the presence of the solute. In a sense, this alternative lattice may be regarded as representing an allotropic form of the pure metal and cases are known where an allotropic modification that normally undergoes transformation at a definite temperature is maintained in a stable condition beyond that temperature by the presence of solute. The effect of nickel in lowering the transformation temperature of gamma into alpha iron is a case in point. In other cases, the allotropic modification may not exist independently, but merely constitutes a latent possibility, which is brought into action by the solute. A case of this kind may be found in the beta body of the copper-zinc alloys and in the beta body of the zinc-aluminum alloys.

If the second or, as it is usually termed, the beta phase in an alloy system is based not on any definite compound but on an alternative, or allotropic, space lattice of the solvent metal, it may be expected that such a beta solid solution will not be as brittle as one based on a compound. This anticipation is borne out by the properties of the beta solid solution in the two best known cases of this kind, which occur in the copper-zinc and the zinc-aluminum systems. Here, again, we are not in position to seek proof among known facts and, therefore, having pointed out the possibilities, or even probabilities of the case, we may leave the matter for future investigation.

I have left to the end one of the most difficult and most important points: the change of solubility with change of temperature. The importance of these differences of solubility has only recently been recognized, but it is now to be regarded as the determining factor in regard to the power that an alloy may possess of undergoing hardening as the result of sudden cooling followed by tempering. It is, therefore, a matter requiring careful consideration from the point of view of any theory that claims to explain the phenomena of solid solutions. Looking at the question from the point of view adopted here, it would seem that the thermal expansions of the two metals concerned should prove decisive. Thus with a solvent A and a solute B, if B is a smaller atom than A but has a greater rate of expansion than A, the two atoms would become increasingly similar and solid solubility should increase with rising temperature. But it does not seem justifiable to suppose that the thermal expansion of the atoms of B, isolated among atoms of A, will necessarily be the same as that of a crystal of B. Further, the matter does not depend solely on the dimensions of the atom, and change of temperature may and probably does bring with it changes in the atom other than those of the lattice constant.

The intensity of the interatomic attraction or bond, the ease with which it can be deflected from its normal position, and even the range over which it extends may be appreciably changed. That changes of this kind actually occur in some metals is suggested by the occurrence of allotropic transformations at definite temperatures, usually accompanied by large and abrupt changes of solid solubility. It is not surprising, therefore, to find that comparison of coefficients of thermal expansion alone does not always indicate correctly the manner in which solid solubility varies with changing temperature. A change in the stiffness of the solvent lattice with changing temperature is likely to have a determining influence in this matter, for it will affect the energy content of the solid solution for a given concentration of solute. Another effect must arise from the nature of the second solid phase formed when a previously saturated solid solution becomes supersaturated and then breaks down; the stiffness of a second type of lattice, often that of an intermetallic compound, enters into the question in this way. It may be hoped that when this whole matter of solid solubilities and their changes with temperature has been as fully investigated as it deserves, a satisfactory explanation may be forthcoming.

If we try to sum up the general result of the consideration in regard to solid solutions which I have presented here, it may appear that there is much that is speculative about the various suggestions and arguments put forward. On the other hand, there is a definite, although limited, amount of fact that bears out the fundamental assumption of the substitution theory. Further, the resulting inferences in several instances lead us to recognize groupings of known facts that almost deserve the name of laws and these fit in with extreme clearness and simplicity with the available data. What is still more important, new aspects of known facts have been brought to light and a large number of lines of new inquiry have been suggested. This furnishes ample justification for putting this theory before you. Further research alone can establish it on a firm foundation. Certainly something more than a *prima facie* case has been made out for the theory and it will, I hope, stimulate thought on whole ranges of phenomena that we have hitherto been content to study in an empiric manner.

Occurrence, Chemistry, and Uses of Selenium and Tellurium

By Victor Lenher,* Ph. D., Madison, Wis.

(New York Meeting, February, 1923)

The interest shown recently in selenium and tellurium has brought many requests for an assembling of the analytical chemistry of these two elements. The intention of this paper is to discuss all of the chemical principles utilized in the detection and quantitative determination of these elements, as well as to describe the works methods actually followed by producers of tellurium and selenium in the United States. These methods have been furnished by S. Skowronski of the Raritan Copper Works, H. D. Greenwood of the United States Metals Refining Co., F. F. Colcord of the U. S. S. Lead Refinery, and E. W. Rouse and A. E. Hall of the American Smelting and Refining Company.

Occurrence

Selenium is found associated with sulfur, to which in many respects it is quite similar. Occasionally deposits of native sulfur, notably those in the Lipari Islands, Kilauea, and Japan, show a red-brown coloration, which is readily distinguished from that of other sulfur; this red-brown sulfur contains a small amount of selenium. Not infrequently, the sulfides of the heavy metals are accompanied by selenium and tellurium, presumably as selenides and tellurides. The Scandinavian pyrite, which is one of the large sources of European sulfuric acid, when roasted has furnished a large part of the selenium that has come to the United States; it was in flue dust from the roasting of Scandinavian pyrites that the element was discovered by Berzelius in 1817. Greek and Spanish pyrites also carry a small amount of selenium. The element has been reported in Yorkshire coke;[1] it is possible that it existed in the pyrites of the original coal. In the United States, pyrites and metallic sulfides other than those of copper have not been carefully studied for their selenium content, the copper-sulfide ores being the only source from which American selenium is obtained. In American copper-smelting practice, selenium follows copper and silver into the blister copper, and is ultimately recovered by treatment of the refinery anode slime.

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* Professor of Chemistry, University of Wisconsin; Chairman of Committee of National Research Council on Uses of Selenium and Tellurium.

Little information is available regarding the association of selenium with the oxidized compounds of sulfur. The geologic origin of some of the deposits of native sulfur is traceable to calcium sulfate; it would seem, therefore, that the selenium in the selenium-bearing sulfur might have its origin in the same source. Our knowledge of the selenium content of the immense deposits of gypsum, barite, sodium sulfate, the Stassfurt minerals, as well as of other sulfates, is limited. A considerable number of selenium compounds of the heavy metals have been identified as minerals but they can hardly be considered as ores of selenium.

Tellurium is rather widely distributed in combination with the precious metals, the tellurides of gold and silver forming valuable ores at Nagyag, Transylvania; Offenbanya, Hungary; Cripple Creek, Colo.; King's Mountain mine, N. C.; Fredericksburg, Va.; Tararu Creek, New Zealand; and the Great Boulder reef in Australia. We have considerable information on the association of gold and tellurium in nature; in general, all tellurium deposits are gold bearing, while, on the other hand, gold is usually accompanied by at least small quantities of tellurium. A large number of tellurides of the heavy metals have been identified, which frequently carry gold.

Like selenium, tellurium is found in small amounts in numerous copper ores; it follows the precious metals into the blister copper and appears eventually in the anode slime. It occurs also in some lead ores, in the smelting of which it follows the precious metals into the work lead; when this is refined by usual methods, the tellurium is removed as telluride of lead, but when it is refined electrolytically, the tellurium is recovered in the anode slimes. The occurrence of tellurium in pyrites and other natural sulfides deserves further study. As to the oxidized compounds, tellurium ocher, native TeO_2, and a few tellurites are known, but only as comparatively rare mineral specimens. Native tellurium is well known. The occurrence of tellurium in the sulfates has not been seriously considered, doubtless because the tellurates are not closely related to the sulfates, either chemically or in respect to their solubilities.

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[1] J. F. Smith: *Jnl.* Soc. Chem. Ind. (1903) **22,** 201.

QUALITATIVE DETECTION OF SELENIUM AND TELLURIUM

Sulfuric Acid Test

Concentrated sulfuric acid gently warmed, or fuming sulfuric acid, in the cold, gives a green color with selenium, the intensity of which varies from a light green to an almost opaque greenish black, depending on the amount of selenium present. This color is ascribed by Weber[2] to the formation of the compound $SeSO_3$. When the green solution is added to water, bright red elemental selenium is precipitated; and on boiling in the diluted acid, this red amorphous selenium turns to the gray crystalline form. The green color in the strong sulfuric acid is destroyed by warming the solution for a few minutes, when sulfur dioxide is evolved and colorless selenium dioxide is produced which is soluble in the sulfuric acid. The above test is applicable to elemental selenium or a selenide, but not to an oxidized selenium compound.

The sulfuric acid test can also be applied to tellurium and tellurides, but not to oxidized compounds of tellurium. The material is treated in the cold with fuming sulfuric acid or with warm concentrated sulfuric acid; if tellurium is present, a red color appears which is due, according to Weber,[3] to the formation of $TeSO_3$. When the red solution is poured into water, black elemental tellurium is precipitated in a finely divided condition. When the red solution is warmed, sulfur dioxide is evolved, the red color disappears, and, if much tellurium is present, white crystals of basic tellurium sulfate separate.

The sulfuric acid test is usually of no value if the two elements occur together, since the intense red of the tellurium may obscure the green of the selenium.

Blowpipe Tests

Selenium compounds, heated in the reducing flame, give characteristic fumes that are commonly described as having a putrid horseradish odor; in the oxidizing or the bunsen flame, selenium compounds give a characteristic blue color. Tellurium, with the blowpipe, gives white fumes but no characteristic odor; in the bunsen flame, tellurium compounds give a pale blue-green color. The fumes, when breathed, after a few hours give the characteristic garlic breath. When roasted in the air, both selenium and tellurium ores give white volatile oxides which, when condensed on a cold surface, deposit as white solids. Selenium dioxide volatilizes without melting, while tellurium dioxide melts to a clear liquid that gives off white fumes.

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[2] *Pogg. Ann.* (1875) **156**, 531.

[3] *J. Pr. Ch.* (1882) **133**, 218.

OXIDIZED COMPOUNDS OF SELENIUM AND TELLURIUM

The oxidized types of selenium fall into two classes; the selenites, corresponding to the oxide SeO_2 or the acid H_2SeO_3, and the selenates, derivatives of the acid H_2SeO_4. Similarly, the oxidized compounds of tellurium are the tellurites, corresponding to TeO_2 and H_2TeO_3, and the tellurates, derived from H_2TeO_4.

Selenium dioxide is a white crystalline solid that dissolves readily in water to form selenious acid, H_2SeO_3. The solubility of the selenites in water follows, in general, that of the sulfites. The selenites are not reducing agents like the sulfites.

Selenic acid physically, and in many respects chemically, strongly resembles sulfuric acid; its salts, the selenates, closely resemble the sulfates in their solubilities. Selenic acid, or the selenates, warmed with strong hydrochloric acid evolve chlorine and are, in turn, reduced to selenites. On the other hand, when in solution, considerable care must be used to oxidize selenites to selenates. Fusion with an alkaline oxidizing mixture converts at once all selenium compounds, including elemental selenium, to selenate.

Tellurium dioxide is a white solid commonly obtained by dehydrating H_2TeO_3, or by roasting elemental tellurium or a volatile tellurium containing compound. Tellurous acid H_2TeO_3 is a white, very sparingly soluble solid. Most of the tellurites are insoluble in water. Tellurous acid or TeO_2, when treated with hydrochloric acid, dissolve to form a solution of tellurium tetrachloride. If this hydrochloric acid solution is diluted with water, hydrated TeO_2 or H_2TeO_3 is precipitated, the precipitation being similar to that of the action of water on antimony trichloride.

The *tellurates* are derived from telluric acid, $H_2TeO_4.2H_2O$, which is a white crystalline solid, soluble in water, and a very weak acid. The tellurates are nearly all insoluble in water; they are formed with ease from the tellurites or from any tellurium compound by fusion with an alkaline oxidizing mixture. When boiled with hydrochloric acid, the tellurates evolve chlorine, being in turn reduced to the tellurous type. Tellurous acid and the tellurites can be oxidized, when sufficient care is taken, to telluric acid or to the tellurates.

PRECIPITATION TESTS FOR SELENIUM AND TELLURIUM

Sulfur dioxide, when passed into a hydrochloric acid solution of selenium, precipitates the amorphous red variety which, on warming, goes to the gray crystalline form.

Hydrogen sulfide gives, with *selenious acid* solutions, a precipitate that is at first lemon yellow, but on standing quickly becomes red, due to the dissociation of the yellow sulfide of selenium into sulfur and amorphous red selenium. The precipitate is soluble in ammonium sulfide. With *tellurous* solutions, hydrogen sulfide gives a precipitate which is red-brown at first but rapidly darkens, due to dissociation into elemental tellurium and sulfur. The precipitate is soluble in ammonium sulfide.

Stannous chloride precipitates from selenium solutions, in the cold, red elemental selenium which passes over into the gray crystalline variety on warming. From tellurium solutions, it precipitates black elemental tellurium.

Ferrous sulfate, hydroxylamine hydrochloride, phosphorous acid, or hypophosphorous acid, added in the cold to a selenious solution, gives red selenium, which goes over into the gray crystalline variety on warming. Potassium iodide added in excess to a hydrochloric acid solution of either a selenite or selenate gives, in the cold, red selenium together with iodine. On warming the solution, the iodine distils and the red selenium goes over into the gray form.

A tellurium solution precipitates black elemental tellurium when treated with stannous chloride, hypophosphorous acid, hydrazine hydrochloride, or metals like zinc, aluminum, and magnesium.

Barium chloride added to a neutral selenious solution precipitates barium selenite, which is soluble in hydrochloric acid. Barium selenate is insoluble in water, like barium sulfate, but is decomposed by hydrochloric acid, as are all selenates, with the evolution of chlorine and subsequent reduction to the selenite, which passes into solution in the hydrochloric acid.

The few alkaline tellurites that are soluble yield, with barium chloride, a white precipitate of barium tellurite soluble in hydrochloric acid. Barium tellurate is precipitated when a tellurate solution is treated with barium chloride It is decomposed by hydrochloric acid, yielding chlorine and forming the tellurite of barium, which dissolves in the hydrochloric acid.

QUALITATIVE DETECTION OF SELENIUM AND TELLURIUM IN COMPLEX MIXTURES

First Method

A portion of the substance is attacked with either aqua regia or hydrochloric acid and potassium chlorate, and the free chlorine is expelled by warming at a temperature below boiling in order to avoid loss of volatile chlorides; the solution is then diluted and filtered to remove insoluble matter. When the solution is diluted, tellurous acid frequently precipitates but is redissolved by further addition of hydrochloric acid. The acid solution is treated with sulfur dioxide gas, or sodium acid sulfite can be added. The formation of a precipitate indicates the possible presence of selenium, tellurium, or gold.

1. If the precipitate is allowed to settle, the liquid poured off, and the precipitate warmed with strong nitric acid, selenium and tellurium will dissolve, leaving the gold insoluble. This nitric acid solution of selenium and tellurium, when evaporated with concentrated hydrochloric acid to expel nitric acid and then treated in concentrated hydrochloric acid solution with sulfur dioxide gas, gives a precipitate of red elemental selenium, which on warming goes to the gray crystalline. This selenium precipitate can be filtered off through an asbestos filter and the solution, when diluted with water and treated with more sulfur dioxide, gives a precipitate of black elemental tellurium.

2. The sulfur dioxide precipitate, containing possibly selenium, tellurium and gold, can be washed with water and dried, after which, treatment with warm strong sulfuric acid or with cold fuming sulfuric acid gives the characteristic selenium or tellurium colors.

Second Method

Crude selenium- or tellurium-bearing material from any source, whether oxidized or non-oxidized, is fused with five to six times its weight of potassium cyanide. Tellurium forms potassium telluride, while the selenium and sulfur, which are invariably present, form the selenocyanate or sulfocyanate, as the case may be. Extraction of the fused mass with water gives a purple solution of potassium telluride, the selenocyanate and sulfocyanate dissolving to colorless solutions. The heavy metals remain insoluble. When a current of air is bubbled through the solution, the purple color of the telluride is discharged, elemental tellurium precipitating out in granular form. This tellurium can be filtered off and tested for by the sulfuric acid test.

The selenocyanate and sulfocyanate filtrate from the air oxidation of the potassium telluride solution can be treated, under a good hood, with hydrochloric acid, when hydrocyanic acid is set free and red selenium precipitated. The latter can be confirmed by conversion to the black variety by heat, or the sulfuric acid test can be applied.

QUANTITATIVE METHODS FOR SELENIUM AND TELLURIUM

Selenium

Selenium may be precipitated in the elemental form and weighed as such on a Gooch crucible after drying at 105°. The precipitating agents most used are either hydroxylamine hydrochloride or sulfur dioxide.

When hydroxylamine hydrochloride is the precipitating agent, the material is usually most conveniently brought into hydrochloric acid solution and brought into the selenious state. From this selenious solution, which may be either acid or neutral, but preferably ammoniacal, hydroxylamine hydrochloride on boiling precipitates elemental selenium which can be filtered on to a Gooch crucible, washed with hot water, dried at 105°, and weighed.

The addition of sulfur dioxide to a solution of selenious acid or a selenite which is strongly acid with hydrochloric acid is one of the oldest and best methods of precipitating elemental selenium. A selenate or selenic acid must first be reduced to a selenite or selenious acid by warming with hydrochloric acid, after which sulfur dioxide can be introduced.

It is sometimes convenient to produce the sulfur dioxide in the hydrochloric acid solution of selenious acid by the addition of a solution of sodium acid sulfite or of sodium sulfite. This procedure is satisfactory but should be accompanied by a blank test on the sulfite with hydrochloric acid, since the sulfites, after standing, not uncommonly give a precipitate of sulfur on acidulation.

Potassium iodide added to a selenious solution containing free hydrochloric acid gives a precipitate of elemental selenium, iodine being liberated simultaneously,

$$SeO_2 + 4\,HI = Se + 2H_2O + 2I_2$$

While in this procedure the iodine can be removed by boiling and the selenium brought on a filter and weighed, Gooch and Pierce[4] prefer the use of sodium arsenite and iodine solutions in carrying out the method.

The thiosulfate method of Norris and Fay[5] consists in treating a hydrochloric acid solution of selenious acid with a measured excess of standard sodium thiosulfate and then titrating the excess of thiosulfate with an iodine solution. The reaction between thiosulfate and selenious acid is expressed

$$H_2SeO_3 + 4Na_2S_2O_3 + 4HCl = 4NaCl + Na_2S_4SeO_6 + Na_2S_4O_6 + 3H_2O$$

Selenates, or selenic acid, can be analyzed by boiling with hydrochloric acid,[6] when chlorine is evolved which can be estimated iodometrically. According to the procedure of Gooch and Reynolds,[7] boiling a selenate or selenic acid with potassium iodide and hydrochloric acid causes the following reaction:

$$H_2SeO_4 + 6HI = Se + 3I_2 + 4H_2O$$

The iodine can be collected and titrated.

The principle of oxidizing selenious compounds to selenic by means of permanganate has been made the basis of a volumetric method, the details of which have been studied by Gooch and Clemons;[8] the fading end point, which occurs when permanganate is added directly to the selenious acid, is avoided by adding a measured excess of the standard permanganate to the acidified solution of the selenious acid, followed by an excess of a standard oxalic acid solution, the surplus of which is titrated with permanganate after warming.

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[4] *Am. Jnl. Sci.* [4] (1896) **1**, 31.

[5] *Am. Chem. Jnl.* (1900) **23**, 119.

[6] Petterson, *Fres. Zeit.* (1873) **12**, 287.

[7] *Am. Jnl. Sci.* [3] (1895) **50**, 258.

[8] *Am. Jnl. Sci.* [3] (1895) **50**, 51.

Selenium can be separated with ease from the metals of which the chlorides are non-volatile. If the selenium exists as selenide, heating in a current of chlorine gas will cause volatilization of selenium tetrachloride; this can be collected in water and reduced by sulfur dioxide or hydroxylamine hydrochloride to elemental selenium which can be weighed. In case of a selenite, heating in hydrochloric acid gas produces $SeO_2.2HCl$, which is volatile and can be similarly reduced to elemental selenium and estimated. The selenates heated in hydrochloric acid gas give chlorine together with the volatile $SeO_2.2HCl$.

Tellurium

Tellurium can be determined gravimetrically and separated from most of the elements, except selenium and gold, by a number of reducing agents. The oldest method, originally proposed by Berzelius, is the use of sulfur dioxide in slightly acid solution. Black elemental tellurium is precipitated either cold or hot, but complete precipitation is much delayed even when warm; the hydrochloric acid solution of tellurium, after treatment with sulfur dioxide, should always be allowed to stand for at least 24 hr. The tellurium is then conveniently brought on to a Gooch crucible and the filtrate further digested after the introduction of more sulfur dioxide. Very frequently, more tellurium settles out on standing 24 hr. longer. After all of the tellurium is collected on a Gooch crucible, it is washed and dried at 105° as quickly as possible, in order to avoid the slight superficial oxidation that takes place and gives slightly high results, as has been shown by Schwetter,[9] Brauner,[10] Norris and Fay,[11] Crane,[12] Frerichs,[13] and Lenher and Homberger.[14]

Gutbier[15] has suggested the use of hydrazine hydrochloride as a reducing agent for the precipitation of elemental tellurium from solution. It gives fairly good results, but complete precipitation, as with sulfur dioxide, is somewhat delayed.

The use of sulfur dioxide and hydrazine together has been found, by Lenher and Homberger,[16] to be widely applicable to both tellurites and tellurates, as well as to the free acids. The tellurium, either as a derivative of the dioxide or as a tellurate, should be present in a solution which has an acidity of approximately 10 per cent. of free hydrochloric acid, and it is preferable that the solution be concentrated, for otherwise the precipitate will be so finely divided as to be difficult to wash. The solution is heated to boiling, 15 cc. of a saturated solution of sulfur dioxide is added, then 10 cc. of a 15 per cent. solution of hydrazine hydrochloride, and again 25 cc. of the sulfur dioxide solution. The boiling is continued until the precipitate settles in such a way that it can be easily washed, which should not take more than 5 min. The precipitated tellurium is washed with hot water on a Gooch filter until all the chlorides are removed, after which the water is displaced by alcohol, and the crucible and contents dried at 105°.

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[9] *Chem. News* (1903) **87,** 17.

[10] *Jnl. Chem. Soc.* (1889) **55,** 392.

[11] *Am. Chem. Jnl.* (1898) **20,** 278.

[12] *Am. Chem. Jnl.* (1900) **23,** 408.

[13] *Jnl. Prak. Chem.* **66,** 261.

[14] *Jnl.* Am. Chem. Soc. (1908) **30,** 387.

[15] *Berichte* (1901) **34,** 2724.

[6] *Loc. cit.*

The precipitation of elemental tellurium from alkaline solution was proposed by Stolba[17] and later studied by Kastner.[18] The precipitation is satisfactory but the tellurium is difficult to wash.

Hydrosulfurous acid as a reducing agent has been studied by McIvor[19] and by Donath.[20] The precipitated metal is contaminated by sulfur, hence the method has no particular advantage.

The precipitation of tellurium by hypophosphorous acid or by stannous chloride has not apparently offered any material advantages for the gravimetric determination of tellurium, although Brauner[21] has suggested that the stannous chloride procedure can be made the basis of a volumetric method. Tellurium is precipitated by a measured excess of a standard solution of stannous chloride, the surplus of which is titrated with an iodine solution.

The several volumetric schemes for tellurium are generally based on either the permanganic or the iodometric principles.

Brauner[22] has suggested that if permanganate be added to tellurous acid or to a tellurite acidified with sulfuric acid, oxidation to the telluric state will occur. As a definite pink end point cannot be obtained directly by this procedure, various modifications have been proposed. One by Brauner consists in adding an excess of standard permanganate followed by an excess of standard oxalic acid, warming, and then titrating the excess of oxalic acid with standard permanganate. The details of this procedure have received considerable attention by Gooch and his coworkers[23] who have proposed to effect the oxidation by permanganate in alkaline solution, acidification with sulfuric acid, and applying the oxalic acid and permanganate titration. Norris and Fay[24] add an excess of sodium hydroxide to the tellurite and follow with an excess of tenth-normal permanganate. The liquid is then diluted with ice water, and treated with potassium iodide followed by sulfuric acid. The liberated iodine is titrated with tenth-normal thiosulfate by Norris and Fay, while Gooch and Peters,[25] following the same general procedure, modify the final titration by the use of sodium arsenite to obtain the iodine liberated.

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[17] *Fres. Zeit.* (1872) **11**, 437.

[18] *Fres. Zeit.* (1874) **13**, 142.

[19] *Chem. News* (1903) **87**, 163.

[20] *Zeit. Angew. Chem.* (1890) **5**, 214.

[21] *Fres. Zeit.* (1891) **30**, 707.

[22] *Monatshefte* (1891) **12**, 34.

[23] *Am. Jnl. Sci.* [3] (1892) **44**, 301; *Ibid.* [4] (1899) **8**, 125.

A volumetric method for tellurous acid, devised by Gooch and Morgan,[26] is based on the principle that, in the presence of 25 per cent. sulfuric acid, the compound TeI_4 is precipitated as a quite insoluble substance. The end point of the titration is the complete precipitation of the iodide by a potassium iodide solution.

Telluric acid may be reduced to tellurous by boiling with potassium bromide and sulfuric acid, bromine being evolved and absorbed in potassium iodide from which solution it can be estimated by a standard thiosulfate solution, according to Gooch and Howland.[27]

Either selenic acid, telluric acid, or their salts, when boiled with hydrochloric acid, give chlorine with reduction to the selenious or tellurous state.

$$H_2SeO_4 + 2HCl = H_2SeO_3 + H_2O + Cl_2$$
$$H_2TeO_4 + 2HCl = H_2TeO_3 + H_2O + Cl_2$$

The chlorine evolved can be passed into potassium iodide solution and the liberated iodine subsequently estimated. This procedure is one of the most convenient laboratory methods for the analysis of selenates and tellurates.

Tellurium, as telluride, in a mixture can be separated from the elements that form non-volatile chlorides by heating in chlorine gas. Similarly, the oxidized compounds of tellurium can be heated in a current of hydrochloric acid gas and the volatile compound $TeO_2.2HCl$ obtained.

SEPARATIONS OF SELENIUM AND TELLURIUM

Both of these elements can be separated from most of the more common other elements by the general principle of reducing their compounds and precipitating them in elemental form by passing sulfur dioxide into their hydrochloric acid solution. Gold is precipitated at the same time but can be removed by treating the well washed precipitate with nitric acid (sp. gr. 1.25) which will dissolve the selenium and tellurium but not the gold. The nitric acid solution can then be carefully evaporated with hydrochloric acid to destroy the nitric, and convert to chlorides.

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[24] *Am. Chem. Jnl.* (1898) **20**, 278.

[25] *Am. Jnl. Sci.* [4] (1899) **8**, 125.

[26] *Am. Jnl. Sci.* [4] (1896) **2**, 271.

[27] *Am. Jnl. Sci.* [3] (1894) **48**, 375.

Fusion of any selenium- or tellurium-bearing material with potassium cyanide yields selenocyanate or telluride of potassium, which are soluble in water, while at the same time metals will be reduced from many of the common heavy-metal mixtures.

Heating of the selenide and telluride combinations or mixtures in chlorine gas affords a separation from the metals the chlorides of which are non-volatile.

Heating of selenites or tellurites in a current of hydrochloric acid gas forms volatile $SeO_2.2HCl$ or $TeO_2.2HCl$, while the selenates or tellurates give chlorine in addition. This treatment with hydrochloric acid gas, when applied to the oxidized selenium and tellurium compounds, is an excellent means of separation from the metals of which the chlorides are non-volatile.

The sulfur dioxide separation of selenium and tellurium, suggested by Shimose[28] and later studied by Keller,[29] has been the method most used for a number of years by the chemists of the electrolytic copper-refining plants for separating the small amounts of selenium and tellurium that are present in copper and copper products. The principle of the method is that tellurium is not precipitated from a solution strongly acid with hydrochloric acid, while selenium is completely precipitated

Keller's procedure is to separate the selenium and tellurium from a nitric acid solution (of 100 gm. of copper) to which has been added 2 to 4 gm. of ferrous sulfate, by the addition of sufficient excess of ammonia; this gives a precipitate of ferric hydroxide containing the selenium and tellurium, and leaves the copper soluble. This ferric hydroxide is dissolved in the minimum amount of dilute hydrochloric acid, and hydrogen sulfide is passed into the solution until it is saturated. The sulfides of selenium and tellurium are filtered, washed, and digested with a solution of sodium sulfide to dissolve them. The solution, after filtering, is acidified with nitric acid and carefully evaporated to dryness. To this residue is added 200 cc. of strong hydrochloric acid, which is then boiled to destroy the nitric, after which the solution is filtered through a Gooch filter, and the solution saturated with sulfur dioxide gas. After settling, the selenium is collected on a Gooch filter, washed with hydrochloric acid (sp. gr. 1.175), pure water, and then alcohol, being weighed after drying. The filtrate containing the tellurium is diluted to twice its volume with water. Owing to the small amount of tellurium usually present in copper, there is generally enough sulfur dioxide in the solution to precipitate the tellurium completely on boiling, but Keller considers it safer to boil the solution for some minutes and while doing so to conduct more sulfur dioxide into it. The tellurium is brought on to a Gooch filter, dried and weighed. The use of hydrazine hydrochloride[30] together with sulfur dioxide as a means of precipitating tellurium is a great improvement in accuracy as well as in speed.

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[28] Divers and Shimose. *Chem. News* (1884) **49,** 26; *Chem. News* (1885) **51,** 199.

[29] *Jnl.* Am. Chem. Soc. (1897) **19,** 771; *Ibid.* (1900) **22,** 241.

Lenher[31] has shown that, in strong solutions, the selenium precipitate contains tellurium, which has been verified by Keller,[32] who later recommended that, in the separation of selenium and tellurium by sulfur dioxide in strong hydrochloric acid solution, the quantity of acid should be about 100 times that of the tellurium.

The separation of tellurium from selenium and sulfur by means of potassium cyanide is one of the older methods. The reactions involved were known to Berzelius, but were first applied as a means of separation probably by Oppenheim,[33] although Crookes studied the selenocyanates in 1851.[34] The method gives fair results as a means for quantitative separation and for the preparation of pure tellurium. The cyanide separation can be conducted either by treatment of the precipitated elements with a solution of potassium cyanide or by fusing any selenium- or tellurium-containing mixture with an excess of potassium cyanide.

The elements, or their compounds, can be fused with potassium cyanide; as the latter is a strong reducing agent in melted form, the oxidized compounds of selenium and tellurium are reduced. The products of the fusion are potassium telluride, potassium selenocyanate and, if sulfur is present, sulfocyanate. The sulfocyanate and selenocyanate are readily dissolved by water to a colorless solution, while the potassium telluride forms a permanganate colored solution, which is decomposed by bubbling a current of air through it, elemental tellurium being precipitated thus:

$$K_2Te + H_2O + O = 2KOH + Te$$

The tellurium can be collected on a Gooch crucible and weighed. The tellurium results from this procedure are usually low, because of a slight volatilization at the fusion temperature and because of the oxidation of a small amount of tellurium to potassium tellurite. The latter can be recovered, but requires considerable manipulation. To avoid this formation of alkaline tellurite, it is sometimes recommended that the cyanide fusion be conducted in an atmosphere of hydrogen.

The solution of selenocyanate, after air precipitation of the tellurium and treating with hydrochloric acid, yields red elemental selenium and large quantities of hydrocyanic acid; the operation must therefore be conducted under a hood with a strong draft. The sulfocyanate does not interfere at any place in the method; on the contrary, the procedure affords an excellent separation of sulfur from selenium and tellurium. A small amount of tellurium is oxidized in the fusion and appears as tellurite in the final solution, from which it can be recovered by sulfur dioxide or by the sulfur dioxide and hydrazine hydrochloride method.

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[30] Lenher and Homberger: *Jnl.* Am. Chem. Soc. (1908) **30,** 387.
Gutbier: *Fres. Zeit.* (1914) **53,** 430.

[31] *Jnl.* Am. Chem. Soc. (1899) **21,** 347.

[32] *Ibid.* (1900) **22,** 243.

[33] *Jnl. prak. chem.* (1857) **71,** 266; *Ibid.* (1860) **81,** 308.

[34] *Jnl. prak. chem.* (1851) **53,** 161.

The separation can also be effected by boiling the precipitated elements with a solution of potassium cyanide, which dissolves the selenium while only traces of tellurium pass into solution.

The Stolba[35] method of precipitating tellurium in elemental form by heating the alkaline solution with a reducing sugar has not worked out as an accurate method of separation; tellurium precipitates first, but selenium in small amounts separates on continued boiling or on standing.

Ferrous sulfate or phosphorous acid precipitates selenium but not tellurium from selenious or tellurous solutions containing hydrochloric acid, but these reactions have not been studied in sufficient detail to be recommended as quantitative methods for the separation of the two elements.

Browning and Flint[36] have utilized the insolubility of tellurium dioxide as a means of separation from the readily soluble selenious acid. Their recommendation is to treat dilute hydrochloric acid solutions of the oxides with 200 cc. of boiling water and add dilute ammonia in slight excess, followed by the faintest excess of acetic acid. Crystalline tellurium dioxide is precipitated while the selenium remains in the filtrate. The dioxide of tellurium can be dried at 105°.

The differential volatility of the bromides and chlorides has been made the basis for separation of the two elements. Gooch and Pierce[37] dissolve selenium dioxide and tellurium dioxide in potassium hydroxide, treat with excess of phosphoric acid, add potassium bromide, and distil in a current of CO_2. Tellurium bromide does not volatilize, while selenium bromide distils and liberates iodine, which is subsequently titrated with sodium thiosulfate.

Scott[38] recommends, for the analysis of alloys containing selenium and tellurium, that the alloy be introduced into a special distilling apparatus and heated with concentrated sulfuric acid to 300° while a current of hydrochloric acid gas is passed through the apparatus. Tellurium remains non-volatile, while the selenium-containing distillate is treated with sulfur dioxide and the precipitated selenium weighed. The tellurium is recovered from the non-volatile portion.

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[35] *Fres. Zeit.* (1872) **11,** 437.

[36] *Am. Jnl. Sci.* [4] (1909) **28,** 112.

[37] *Ibid.* [4] (1896) **1,** 181.

[38] "Standard Methods of Chemical Analysis," 2d ed., 361.

METHODS AT REFINERIES

Commercial Selenium.[39]—A 0.5 gm. portion of the material ground to 100 mesh is placed in a 150 cc. beaker, and 10 cc. of water are added, followed by 15 cc. conc. HNO_3. After the sample has dissolved in the beaker, which is covered with a watch glass, it is evaporated to dryness on the water bath, and taken up in 10 cc. HCl and 20 cc. of water in the cold. The insoluble is filtered off and the solution received in a 400 cc. beaker. Sufficient conc. HCl is added to make the solution 70 per cent. conc. HCl by volume.

The selenium is precipitated at room temperature by passing a slow current of SO_2 gas through the solution at the rate of two small bubbles per second stirring frequently to granulate the selenium. It is recommended that the temperature of the solution be maintained at 60° to 70° F by placing the beaker in a vessel of running water.

When complete precipitation has been effected and the solution smells strongly of SO_2, the beaker is removed and allowed to settle for ½ hr. The supernatant liquor is decanted through a tared Gooch filter; the precipitated selenium in the beaker is washed three times with conc. HCl and once with cold water, decanting each time through the crucible. To the precipitate in the beaker, 25 cc. of cold water are added, followed by hot water and vigorous stirring until the selenium turns black and granular. It is then filtered, washed with hot water followed by alcohol, and dried at 105° to constant weight. After weighing, the crucible can be gently heated to expel the selenium, in order to obtain a check on its purity. A residue may consist of silica or gold.

Tellurium in the filtrate is recovered by adding 3 gm. of powdered tartaric acid, diluting with four times its bulk of hot water, then adding 25 cc. NH_4OH and saturating with SO_2 gas. After the SO_2 treatment, which takes but a few minutes, the solution is brought to boiling and allowed to stand for 2 hr. on a hot plate. The granular elemental tellurium is brought on a tared Gooch filter, washed with hot water, dried at 115° and weighed.

The important laboratory details suggested by Greenwood are:

1. Evaporation of the selenious acid should be made on the water-bath rather than on a hot stove in order to avoid loss by volatilization. There is an appreciable loss of selenium dioxide when heated above 100° C. even in the presence of sodium and potassium chlorides. This fact has been verified in Lenher's laboratory.

2. If the temperature of the solution during the precipitation of selenium is above 70° F., the selenium agglomerates and occludes impurities, which cannot be washed out. If the temperature is below 60° F., the precipitation is either incomplete or very much delayed.

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[39] H. D. Greenwood: *Eng. & Min. Jnl.* (1915) **100,** 1012.

3. If the precipitated selenium is not granular, it cakes during the drying and retains moisture even at 110°.

Lead, copper and iron in selenium are determined in a sample of from 10 to 25 gm. which is dissolved in 50 to 75 cc. conc. HNO_3 in a 375 cc. casserole and evaporated to dryness. The volatile SeO_2 is expelled by carefully raising the temperature. The non-volatile residue is dissolved in 10 cc. conc. HNO_3 and 5 cc. conc. HCl, evaporated to 5 cc., when 5 cc. conc. H_2SO_4 are added and evaporated to fumes; it is allowed to cool, 75 cc. of water are added and allowed to stand over night, when the lead sulfate is filtered off and weighed as usual. The filtrate from the lead is treated with H_2S, the precipitate is ignited to burn off selenium, tellurium and arsenic, the residue dissolved in HNO_3 and the copper determined volumetrically. The filtrate from the precipitation of the copper is boiled to expel the H_2S, oxidized by a few crystals of potassium chlorate, and the iron precipitated in the usual manner by ammonia.

An alternate method for the determination of *iron in selenium* is to weigh 10 gm. of the sample into a porcelain dish, ignite at a red heat until the selenium appears to be completely driven off. The residue is sometimes weighed and reported as "non-volatile matter." This residue is fused with sodium carbonate, treated with dilute sulfuric acid, the solution reduced with zinc, and the iron titrated with a weak solution of permanganate.

Insoluble in Commercial Selenium.—Dissolve 10 to 25 gm. in a 375-cc. casserole, in conc. HNO_3, and evaporate to dryness; add conc. HCl and dehydrate the silica. The insoluble is taken up in HCl and water, filtered off, ignited and weighed.

Selenium and Tellurium in Metallic Tellurium.—A 0.5 gm. sample of the finely powdered metal is treated with 10 cc. of conc. H_2SO_4 and fumed until all the tellurium has dissolved; after which it is cooled and 20 cc. water and 50 cc. conc. HCl are added.

Selenium is precipitated from the cold solution by passing in SO_2 gas for 3 to 4 min., after which it is filtered through a Gooch crucible, washed three times with HCl (2 parts acid to 1 water) then with hot water and finally with alcohol, dried and weighed. To obtain a check on the purity of the selenium the Gooch crucible is ignited and reweighed.

The tellurium-containing filtrate is diluted to about 700 cc., heated to nearly boiling, and a rapid current of SO_2 gas is passed in for 15 min., or until the tellurium separates readily; it is then brought on to a Gooch filter, washed with hot water, finally with alcohol. The elemental tellurium is dried for 1 hr. at 110° and weighed.

A selenium and tellurium procedure used for a number of years in one of the refineries consists in dissolving in nitric acid, adding a pinch of salt, evaporating to dryness, taking up in 25 cc. HCl (1:1) and bringing to boiling. The insoluble matter is filtered off, the solution diluted to 150 cc. and SO_2 passed through the boiling solution. A few grams of hydrazine hydrochloride are added and SO_2 again passed in after which the solution is boiled a few minutes. The precipitated selenium and tellurium are filtered off, dissolved in nitric acid with the addition of a pinch of salt, and the solution evaporated. The residue is taken up in about 200 cc. conc. HCl, the solution boiled about 10 min. and the selenium brought on a Gooch crucible where it is washed with hot water, then alcohol, dried and weighed.

The tellurium-containing filtrate is diluted with water to three times its volume and saturated with SO_2; a few grams of hydrazine hydrochloride are added, boiled a few minutes, the tellurium is brought on a Gooch filter, washed with hot water, then with alcohol, dried and weighed.

Quick Method for Tellurium and Iron in Selenium.—According to Kann,[40] 5 gm. of finely powdered selenium are dissolved in 10 cc. conc. HNO_3 in a 25 cc. tared porcelain dish. After solution has taken place, two drops of conc. H_2SO_4 are added and the solution evaporated to dryness on the hot plate. The SeO_2 is then volatilized over a free flame and the residue of tellurium dioxide and iron oxide is weighed. The residue is dissolved in conc. HCl, diluted with water and treated with H_2S gas. The precipitated tellurium sulfide is filtered off. After boiling and oxidizing with a few drops of conc. HNO_3, iron is precipitated by ammonia as ferric hydroxide which can be weighed as oxide, or it can be dissolved in dilute sulfuric acid, reduced with zinc and titrated with permanganate. The weight of iron oxide deducted from the weight of residue in the porcelain dish gives the weight of the tellurium.

Selenium and tellurium in blister or pig copper are commonly determined by the method of Keller, using samples of 50 gm. or less.

Selenium and tellurium in electrolytic copper slimes are determined by treating a 0.5 gm. sample of the slime in a 250 cc. beaker with 10 cc. conc. H_2SO_4, heating until the sample is decomposed and nothing remains but a white residue. After cooling, 20 cc. of water are added, followed by 2 cc. conc. HCl; the solution is agitated to coagulate the silver chloride which is then filtered off. The acidity of the filtrate is brought up to about 80 per cent. by adding conc. HCl and the selenium and tellurium separated by SO_2, following the customary procedure.

Selenium in Lead Slimes.—A 2 gm. sample is fused with a mixture of 8 gm. sodium carbonate and 2 gm. nitrate in a nickel crucible. The cold fusion is extracted with water and filtered. The filtrate is acidified with HCl and heated until chlorine is expelled. To the solution is added an equal volume of HCl, and SO_2 gas is added until the red precipitate becomes granular. The selenium can be brought on an asbestos filter, washed with HCl, redissolved in HCl and $HClO_3$, and reprecipitated by SO_2.

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[40] *Chem. Engr.* (1908) **8**, 32.

Tellurium in Lead Slimes.—1 gm. of slimes is treated with a mixture of 10 cc. conc. H_2SO_4, 10 cc. conc. HNO_3, and 20 cc. water, and evaporated to fumes. After cooling, 40 cc. water and about 2 gm. tartaric acid are added. The solution is boiled and filtered. The residue is washed back into the original beaker, 5 to 10 cc. conc. H_2SO_4 are added, and again evaporated to fumes; 40 cc. of water are added and the solution boiled and filtered. The two filtrates are united and treated with H_2S gas. The sulfides are filtered on paper washed with H_2S water, washed back into the beaker, and a little sodium bicarbonate added followed by about 4 cc. of 10 per cent. sodium sulfide solution. The solution is brought to boiling, digested for 12 hr., and filtered through the filter previously used. The sulfide-containing filtrate is acidified with dilute sulfuric acid and treated with H_2S gas, to render it granular. The sulfide precipitate, after filtration, is dissolved in nitric acid with the addition of 10 cc. conc. H_2SO_4 and evaporated to fumes of H_2SO_4. It is then diluted with water and boiled after adding about 2 gm. of tartaric acid. The solution is cooled, diluted to about 60 cc. with water, and after bringing up to about 100 cc. with conc. HCl, is again treated with H_2S. The precipitate is filtered off, washed with 1:1 HCl, dissolved in HCl and $KClO_3$, warmed gently to expel the excess of chlorine, tartaric acid is again added, any residue filtered off, and the filtrate made strongly acid with HCl. The selenium is then precipitated by SO_2.

The filtrate containing the tellurium is diluted with warm water, and the tellurium precipitated and weighed as usual.

Tellurium in Lead Bullion Containing Antimony and Arsenic.—A 10 gm. sample is dissolved in 20 cc. HNO_3, diluted to 100 cc., 10 cc. H_2SO_4 are added and, after settling, the clear portion is decanted and washed; 10 cc. H_2SO_4 are added to the filtrate, which is then evaporated to fumes. After cooling, 50 cc. of water and 5 to 10 gm. of tartaric acid are added. The solution is boiled and allowed to settle. The lead sulfate is filtered off on a Gooch filter and the filtrate treated with H_2S until precipitation is complete. The sulfide precipitate is filtered off, washed with H_2S water, transferred back to the same beaker, a little sodium bicarbonate is added, followed by about 4 cc. of a 10 per cent. solution of sodium sulfide, after which the solution is filtered.

The sulfide filtrate containing selenium and tellurium is acidified with dilute H_2SO_4, and H_2S is passed through to cause the precipitate to granulate. The sulfide precipitate is filtered off, dissolved in HNO_3 and 10 cc. H_2SO_4, and evaporated to fumes. After cooling, it is diluted to about 60 cc., 2 gm. of tartaric acid are added, followed by 40 cc. conc. HCl, and the solution is again treated with H_2S. The precipitate is dissolved in HCl and $KClO_3$, the excess of chlorine being removed by gentle warming;

a little tartaric acid is added, followed by excess of conc. HCl, after which the selenium is precipitated by SO_2 and weighed.

The filtrate, after dilution with water, is treated with SO_2 and the tellurium precipitated and weighed.

Selenium and Tellurium in Flue Dust.—2 gm. are dissolved in HCl and $HClO_3$ on a water bath, and chlorine expelled by gentle heat. The insoluble matter is filtered off and washed with concentrated HCl, and and the filtrate is treated with SO_2 gas. The selenium and tellurium are are filtered on to a Gooch crucible, washed, then dissolved in HCl and $KClO_3$; about 2 gm. of tartaric acid are added, followed by conc. HCl, and the selenium is precipitated by SO_3, filtered off and weighed. The filtrate is diluted to three times its volume with warm water, and the tellurium is precipitated by SO_2, collected and weighed.

Flue dust or niter slag from Dore furnaces can be analyzed for the water-soluble selenium and tellurium by boiling a 1 gm. sample with water, filtering and washing with hot water, keeping the volume down to 20 cc. To this filtrate are added 200 cc. conc. HCl; the solution is chilled with ice water, and SO_2 is passed in, the selenium and tellurium being separated by the method of Keller.

The insoluble selenium and tellurium are transferred from the filter to a 50 cc. beaker, conc. HNO_3 and conc. HCl are added, and the solution evaporated at 50° C. or below. It is recommended to evaporate twice more with HCl, keeping the temperature at 50° C. or below. The residue is taken up in HCl (1:2) and filtered, after which 100 cc. conc. HCl are added and the selenium and tellurium in the filtrate are separated by the Keller method.

Commercial Sodium Selenite.—To 1 gm. of the sample in a 50 cc. beaker, are added 10 cc. of water and 5 drops HCl and shaken gently until solution is complete. After filtering out the insoluble matter, a large excess of conc. HCl is added to the filtrate and selenium is precipitated by SO_2. In the filtrate from the selenium, the tellurium can be recovered by diluting with warm water and passing in more SO_2.

Selenic Acid.—20 gm. of the sample are quickly transferred to a tightly stoppered weighing bottle. After weighing, the acid is put into a liter flask, dissolved in water, and made up to the mark. A 25 cc. portion is measured out and treated with an excess of conc. HCl and the selenium precipitated by SO_2. If tellurium is present, it can be recovered in the filtrate.

Selenium and Tellurium in Copper Bullion.—25 gm. of the sample are dissolved in a 1000 cc. beaker in 100 cc. HNO_3 (sp. gr. 1.42) and 50 cc. water. When solution is complete, 10 cc. of ferric nitration solution (10 cc. = 2g.) are added and the solution is diluted to 500 cc. The iron is precipitated either by carefully neutralizing with ammonia or sodium carbonate, or by adding ammonia before the addition of ferric nitrate until a decided precipitate of copper hydrate is formed, then adding the ferric nitrate, depending on the iron to replace the copper precipitate. The solution is diluted to 800 cc. and boiled for 5 min. The iron precipitate found should be flocculent and settle rapidly. A slight precipitate is an indication of insufficient iron for the impurities present in the copper. The determination should be repeated either with a 10 gm. sample of the bullion or a larger amount of ferric nitrate.

The solution is allowed to settle until perfectly clear, after which it is decanted, 500 cc. boiling water added, again settled, and followed by decantation; 20 cc. of sulfuric acid (concentrated) are added and the whole evaporated to fumes of SO_3. On cooling, the residue is boiled with 30 cc. of water, 3 drops of HCl added to precipitate the silver and the whole stirred well to coagulate the precipitate which is removed by filtration.

To the filtrate an equal volume of concentrated HCl is added, the solution is cooled, and the selenium precipitated by passing in sulfur-dioxide gas. The selenium is weighed in the usual manner. To the filtrate from the selenium, 10 gm. of tartaric acid are added and nearly neutralized with ammonia, after which the tellurium is precipitated from the hot solution by passing in sulfur-dioxide gas. The tellurium is weighed in the usual way.

USES OF SELENIUM, AND ITS MARKET

The principal uses of selenium are in the manufacture of ruby glass and as a decolorizer of white glass. In the latter case, the selenium is introduced into the glass in the elemental form or as a selenium compound. As the red color that selenium imparts to the glass is not exactly the complement of the green of ferrous iron, cobalt oxide is usually added with the selenium. The objections to the more extended use of selenium as a decolorizer are its volatility at glass-making temperatures, and its higher cost as compared with manganese dioxide.

The selenium cell has received considerable attention for many years. Crystalline selenium is an electrical insulator in the dark, but when exposed to light it becomes a conductor, the conductivity being proportional to the intensity of the light. The crystalline modification is the form in which it is used in the selenium cell, which is the essential element in a number of mechanical appliances, such as the optophone, an apparatus invented by D'Albe by which the blind may read ordinary type by ear; the self-lighting buoy; control apparatus for chimney draft, control of progress in sulfuric acid manufacture by contact processes; and various other minor uses. The selenium cell requires considerable study, as it can hardly be said to have reached the stage of dependable service.

Some well directed effort has been applied to the manufacture and marketing of selenium oxychloride, which possesses certain remarkable properties as a solvent for a wide range of organic substances.

Other possible uses for selenium,[41] suggested by its chemical similarity to sulfur, are: the preparation of colored lithopones after the same manner that solutions of antimony, cadmium and arsenic can be used; in medicine, for which some attention has already been given to the simpler chemical combinations; the vulcanization of rubber, for which a few experiments have indicated that selenium acts much like sulfur; and in dyestuffs, for which purpose selenium can hardly be expected to replace sulfur unless investigation should disclose colors of particularly striking character.

The market price for elemental selenium ranges from $2 to $2.25 per lb., with an annual production in the United States of about 100,000 lb. The copper refineries are in a position to produce considerably more than this quantity, should there be sufficient demand for the element. The customary method of preparing selenium for market is described by Lawrence Addicks.[42]

TELLURIUM, ITS NATURE, USES, AND MARKET

In appearance, elemental tellurium closely resembles antimony, being white, crystalline, and very brittle. Density, 6.20; melting point, 452° C.; boils at red heat, giving off a golden yellow vapor if air is excluded. Toward acids, it is as refactory as antimony, and is resistant to alkaline solutions except concentrated KOH; it does not corrode appreciably in water or moist air.

The last-mentioned property of tellurium suggests its use in electrolytic plating, like antimony; the available electrolytes have not been examined.

Metallic tellurium is used as a crystal detector in the wireless industry. A small amount is used in marking glass in the interior of the electric-light bulb. It has been used in a small way in high-resistance and other alloys, in organic dyes, for staining silver, in medicine, and as a reagent in chemical laboratories. As a coloring agent in glass or porcelain, blue, brown, and red may be produced by tellurium, some of the best ceramics being colored in this way.[43] Tellurium dissolved in sodium-sulfide solution is used in toning baths for photographic prints.

The use of 0.2 per cent. diethyl telluride in gasoline as an antiknock compound has been reliably reported. It is said to eliminate carbon deposits and to produce greatly increased efficiency when used in motors designed to operate on very high compression. A special type of engine is said to be required to produce these results, hence its general use in motors would not be feasible unless the motor industry should conform to the required type. This step, in turn, would be dependent on a supply of tellurium adequate to treat all the motor fuel. For this purpose, 1500 tons of tellurium per year would be required, which, of course, is far beyond the present or possible supply. Later investigations have shown that lead methyl and similar compounds, cheaper than the diethyl telluride, are almost equally satisfactory as addition agents for gasoline.

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[41] *Jnl. Ind. & Eng. Chem.* (1920) **12**, 597.

[42] "Copper Refining," 113. N. Y., 1921. McGraw-Hill Book Co.

[43] *Chem. Zeit.* (1914) **38**, 873.

The production of tellurium and its compounds is very small, but the copper refineries of the United States could produce annually between 100,000 and 150,000 lb. of the element, should uses be found. Owing to its small consumption, no market standards for tellurium have been developed.

DISCUSSION

D. M. Liddell, New York, N. Y.—The investigator should be warned against getting selenium acid under the fingernails; its reduction to metallic selenium a few hours later causes the most acute pain. A 5 per cent. solution of cocaine affords some relief.

The statement that "selenium is ultimately recovered by treatment of the copper refinery anode slime," while true, is incomplete. It is most easily recovered from the flue dust or alkaline slags from the slimes-smelting furnaces, not from direct treatment of the slimes. There is no commercial recovery by direct treatment.

The statement "fusion with an alkaline oxidizing mixture converts at once all selenium compounds, including elemental selenium, to selenate" is true but this process may be exceedingly dangerous on a large scale. I have had violent explosions from this cause due to gunpowder-like mixtures of an oxidizer with flue-dusts carrying elemental selenium and finely divided carbon.

On the same page, under "precipitation tests," might be added the statement that carbon monoxide reduces selenious-acid solutions; of course, it is a dangerous reagent to work with.

Under certain conditions of precipitation, a selenium is produced that may be separated from tellurium by solution in carbon disulfide; but I have had no chance recently to check this by experiment. The metallic and vitreous modifications of selenium are, however, completely insoluble in carbon disulfide.

The instability of the alkaline-telluride solutions in air should be emphasized to make this separation under optimum conditions, the solutions must be filtered as far as possible without access of air, in case any insoluble matter remains and must be removed after the cyanide fusion.

There is no mention of the Knorr distillation separation of selenium and tellurium. Knorr separates arsenic and antimony by distillation of the chlorides in hydrochloric-acid solution with repeated addition of acid. Tin comes over partly with the arsenic, partly with the antimony, selenium comes with the antimony, leaving the tellurium behind.

Arsenic, tin, antimony, selenium and tellurium are thrown down in the basic ferric sulfate or nitrate precipitations as described by Knorr. It is disputed whether selenium and tellurium will come down completely with ferric hydroxide as described by Keller; later workers, such as Skowronski and Greenwood, say they do not. Keller, basing the statement on work with weighed quantities of selenium, says that if the quantity of iron bears a sufficiently large ratio to the selenium and tellurium, the precipitation is qualitative.

For convenience, it may here be noted that of the common elements bismuth, lead, gold, and platinum will also be found in this precipitate under the conditions described by Doctor Lenher. This is practically the method advised in the last paragraph on p. 1047, attributed to Scott, though I think Knorr devised it, but without anything being said as to tin and antimony interfering with the selenium determination.

Without time to recheck Kann's method on page 1050 for the quick determination of tellurium and iron in selenium, I should imagine PbO, SiO_2, SnO, and Au might be found in the original residence and be eventually calculated as tellurium oxide, in the method as given. I had always considered it necessary to reprecipitate the tellurium for accurate results.

C. S. Witherell, New York, N. Y.—Has anything been done with these metals for electroplating?

S. Skowronski, Perth Amboy, N. J.—Selenium cannot be used for electroplating, for the elemental red selenium is deposited. Tellurium, however, is easily electroplated from a sulfate solution, and this method may be used as a means of refining or recovering tellurium from solutions. Deposits ¼ in. thick have been obtained; the metal itself, however, is so brittle that it is questionable whether the deposit would have any merit for electroplating.

C. S. Witherell.—Can these metals be used for flashing metals in electroplating?

S. Skowronski.—As a strictly "flashing" proposition, selenium and tellurium have not been used. Both metals in solution act like sulfides, forming selenides and tellurides with copper, silver, and alloys, and as such may be used as stripping agents, to prevent the deposit from adhering fast to the cathode, thus insuring easy stripping of the deposit. A patent for the use of selenium for this purpose was granted to T. A. Edison in 1920.

Solutions of tellurium are finding increasing uses in the electroplating industry as a dip for staining silverware.

D. M. Liddell New York, N. Y.—Selenium is unique among the metals in that it deposits in elemental form on both anode and cathode from a seleno-cyanide solution. By plating selenium on gold, it is possible to get an adherent deposit and to make beautiful interference bands.

Victor Lenher (author's reply to discussion).—Relative to the comments of Mr. Liddell, it is suggested that all of the selenium recovered in the United States today, and for many years, is from copper-refinery anode slimes. Several types of processes, all more or less complicated, are in use; there is no known recovery of selenium by direct treatment. The statement that selenium "produced under certain conditions of precipitation" can be separated from tellurium by solution in carbon disulfide as an analytical procedure requires experimental verification.

Regarding the ignoring of Knorr's distillation method, Mr. Liddell doubtless is unaware that Scott's distillation method probably owes its origin to the Knorr method. The Keller method of separating selenium from tellurium has been described as given by the refineries using this method.

Bright Annealing of Copper Wire in an Atmosphere of Natural Gas

By P. E. Demmler,* East Pittsburgh, Pa.

(New York Meeting, February, 1923)

The apparatus in which the process of bright annealing of copper wire was carried out consisted of a section of iron pipe, 6 ft. long and 3 ft. in diameter. The pipe was provided with flanges to which were bolted the end plates for sealing, also with inlet and outlet tubes to allow for the displacement of the air in the pipe by natural gas. After charging with copper wire, sealing and displacing the air, the pipe was heated to 350° C. (662° F.). It was expected that the natural gas would furnish a neutral atmosphere which would prevent oxidation and give a satisfactory bright-annealed product. It was found, however, that the outer layers of wire were discolored and not suited for the use for which they were intended.

To obtain an idea of what was taking place in the annealing furnace, skeins of the hard-drawn wire were placed in a glass tube so that all changes in appearance could be observed as they were heated to different temperatures and under varying conditions. The temperature in each case was determined by means of a thermocouple placed in the tube close to the wire. Following is a summary of the results of these experiments:

1. When the wire was heated for 1 hr. in a current of natural gas at a temperature of 350°, 400°, or 500° C. (662°, 752°, or 932° F., respectively), there was a marked darkening of the wire throughout the skein.

2. When heated for 1 hr. in natural gas at a temperature of 600° C. (1112° F.), there was no coloration, the wire remaining clean and bright.

3. When heated in natural gas to 350° C. until darkened and the treatment continued with the temperature raised to 600° C., the coloration disappeared and the wire became clean and bright.

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* Chemical Engineer, Research Laboratory, Westinghouse Electric & Mfg. Co.

4. When heated to 350° C. in natural gas that had been passed over copper heated to 600° C. before entering the annealing chamber, no coloration was produced, the wire remaining clean and bright.

5. When heated in air to 350° C. for 15 min., a heavy scale of oxide was produced. On displacing the air by natural gas and then heating to 600° C., the dark color disappeared and the wire became clean and bright.

Oxygen in Natural Gas

The coloring of the copper wire during the bright annealing process was evidently the result of oxidation caused by a small amount of oxygen contained in the natural gas. The presence of oxygen in the gas was demonstrated by alkaline pyrogallate solution, which became dark in color after passing the gas through it for several hours.

The source of the oxygen in the gas was not determined. According to G. A. Burrell[1] the traces of oxygen shown on analysis are not necessarily present in the gas at the wells, but are probably introduced as the gas traverses the pipe lines. Burrell, Seibert and Oberfell[2] state that oxygen is not a constituent of natural gas as it occurs in the earth, while F. C. Phillips[3] states that traces of oxygen are indicated with great positiveness in gas as it flows directly from the wells and under high pressure. In some places, the amount of natural gas, as supplied to the consumer is, at times, increased by the addition of manufactured gases, as shown by the presence of considerable amounts of carbon monoxide, carbon dioxide, hydrogen, oxygen, and nitrogen. The presence of oxygen at any time presents a factor that must be considered and dealt with in the process under consideration. While the facts here presented refer to the use of natural gas for producing a neutral atmosphere, similar points would require attention in the case of coal gas and other manufactured gases.

Treatment of Natural Gas

The experiments outlined above demonstrate that the bright annealing of copper wire in natural gas containing small amounts of oxygen is possible by either of two procedures: (1) Heat the wire to 600° C.; or (2) pass the natural gas over copper heated to 600° C. before passing it into the annealing chamber, containing the wire to be annealed, heated to 350° C.

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[1] Characteristics of the Natural Gas used at Pittsburgh: Bureau of Mines, *Bull.* 15, 64.

[2] The Condensation of Gasoline from Natural Gas: Bureau of Mines, *Bull.* 88, 68.

[3] The Chemical Composition of Natural Gas: Second Geol. Surv. of Pennsylvania, Part 1, Oil and Gas, (1886); reprinted in W. Va. Geol. Surv. (1904), I*a*, 507.

Practical considerations indicate that the second method is the more desirable. There is a considerable saving in heating the large annealing chamber to 350° C., rather than to 600° C. Furthermore, it is not practical to anneal wire on spools at 600° C., as at this temperature the expansion causes the wire to weld so that it may become impossible to unwind it from the spools. The small amount of copper wire necessary for treating the gas in the preheating furnace lasts indefinitely, as the action is catalytic and there is no noticeable change in the copper after use.

The explanation of the action is, briefly: Natural gas is composed largely of hydrocarbons of the methane series; it may contain small amounts of other hydrocarbons, also hydrogen, nitrogen, and other gases. The amount of oxygen may vary from none to several per cent. of the gas. When gas containing small amounts of oxygen is passed over copper heated to 350° C., as in the bright annealing process, the oxygen will combine with the copper and form a dark skin of copper oxide; the hydrocarbons will not combine with the oxygen in the gas nor will they reduce the copper oxide. At 600° C., the hydrocarbons will reduce the copper oxide to copper, an equivalent amount of hydrocarbon being oxidized to carbon dioxide and water. We may thus consider that, at 600° C., the copper is oxidized to copper oxide, which, however, is immediately reduced by the hydrocarbons present, the copper acting as a catalytic agent, causing the reaction between oxygen and hydrocarbon at a temperature lower than that at which they would react in the absence of copper. The ignition temperature[4] of an explosive mixture of methane and oxygen is given as 650–750° C. The final reaction may be represented by the equation

$$CH_4 + 2O_2 = CO_2 + 2H_2O$$

but there is the intermediate reaction between copper and oxygen, so that the complete reaction may be represented by the equations

$$4\,Cu + 2O_2 = 4\,CuO$$
$$4\,CuO + CH_4 = CO_2 + 2H_2O + 4\,Cu$$

In this case, the steps of the reaction can be demonstrated and it may be classed as one of those certain cases of contact catalysis where it is known that certain definite intermediate compounds are formed.[5]

C. Sandonnini[6] found that in using copper as a catalyst in a mixture of ethylene, hydrogen and oxygen, the oxidation of the ethylene is not much increased, although the copper is oxidized, an action analogous to the oxidation of copper in natural gas as noted above.

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[4] G. A. Burrell. Bureau of Mines. *Bull.* **15.**

[5] First Report of Committee on Contact Catalysis, W. D. Bancroft, Chairman: *Jnl.* Ind. & Eng. Chem. (1922) **16,** 326.

[6] *Gazz. Chem. Ital.* (1922) **52,** 1, 394-408. *Chem. Abst.* (1922) **16,** 3571.

Pretreatment of natural gas by passing it over copper heated to 600° C. thus furnishes a simple means of getting a satisfactory bright annealed copper wire, where annealing at 350° C. in natural gas produces a dark color. No change in either the annealing oven or the process is necessary; the apparatus for heating the gas can readily be connected outside the oven and the treated gas may be used in the regular process.

DISCUSSION

T. S. Fuller, Schenectady, N. Y. (written discussion).—The author's experiments should be considered with the results of previous experiments on the embrittlement at elevated temperatures in reducing gases of copper containing small amounts of oxygen; otherwise the conclusions may be dangerous.

Copper, containing the usual amount of oxide (0.5 per cent $Cu_2O \pm$), when heated in hydrogen or carbon monoxide is made brittle, the degree of brittleness depending on the temperature and time. Hydrogen and carbon monoxide diffuse rather rapidly through heated copper, reducing the oxide in their path, thus forming steam or carbon dioxide as the case may be. The rate of diffusion of steam and carbon dioxide in the heated metal is less than the rate of hydrogen and carbon monoxide. Because of this difference in diffusion rates, a pressure sufficient to produce intergranular cracks is built up and the copper is correspondingly weakened. Oxygen-free copper, of course, is not made brittle in this manner. Inasmuch as natural gas contains small amounts of both hydrogen and carbon monoxide and inasmuch as 600° C. for 1 hr. is sufficient to produce embrittlement, if the author has failed to observe such an action the logical assumption is that the copper used by him in his experiments was oxygen free, or nearly so. It is always imperative in the annealing of copper in the presence of even minute amounts of reducing gases to be on the lookout for this embrittlement.

Otto Lellep, New York, N. Y. (written discussion).—In the manufacture of wire, bar castings are rolled into a thick wire which is drawn, in successive operations, through a series of dies to the desired diameter. After a number of drawing operations, the wire, which was soft at the beginning, becomes hard and less ductile because of a structural deformation of the metal crystals. To make the wire soft and suitable for further drawing operations, it is annealed at a temperature anywhere between 350° to 700° C. in sealed retorts, usually filled with some gas that does not discolor the bright surface of metal.

As a standard furnace for a continuous large-scale annealing of copper wire, the Bates & Peard type could be mentioned. It is essentially a horizontal iron retort about 11 ft. long, 4 ft. wide, and 2 ft. high heated externally by fire gases. To each end of this large pipe or retort are bolted 30° ells of the same cross section. The ends of the ells plunge into a basin of water so the contents of the retort are sealed against the outside air. An endless chain conveyor moves through the water-sealed retort and returns below the retort under the water. To this conveyor the spools of copper wire are fastened and thus carried through the furnace during about half an hour at a temperature of 600° C. The retort fills with water vapor generated from the materials.

A large copper-wire manufacturer tried to replace the externally oil-flame heated retort furnace by a simpler construction using city gas as fuel, as some experiments had demonstrated that fire gases resulting from a complete combustion of city gas were non-oxidizing for copper. The furnace worked over a week under this new arrangement and about 100 tons of large-size copper wire were annealed. When the temperature of the wire was kept high, say 650° C., the surface of the wire remained unusually bright. But at the temperatures much below 600° C., which were required for annealing thin wire, the surface was discolored, gray, and occasionally black.

It was thought that the wire was coated by copper oxide due to free oxygen in the furnace atmosphere. Repeated analyses of the furnace gases by Orsat apparatus, however, did not reveal any noticeable amounts of oxygen. It was then thought that the city gas might change periodically so that temporarily an excess of air might occur in the furnace; the burners, therefore, were supplied with a homogeneous mixture of gas and air automatically well proportioned and a slight excess of gas against theoretical mixture was fed into the burners. A number of companies are marketing devices that proportion gas and air automatically in a settled ratio. A special flame indicator was erected on the mixture manifold. It consisted of a back-fire preventing grid to be used on the top of Meker burners, only the voids between the grid plates were made narrower.

A 1/8-in. copper wire kept on the flame above this grid is a very sensitive indicator of the mixture supplied into the furnace. With a slight excess of gas, the indicator wire remains bright; as soon as any free oxygen occurs the indicator wire becomes covered with a dull layer of copper oxide. This device makes it possible to keep a constant control of the furnace atmosphere. To be safely on the reducing side, the fire gases were regulated so that the Orsat apparatus showed anywhere from 0.5 to 3 per cent. of carbon monoxide and no noticeable amounts of free oxygen. Contrary to all expectations, the copper often came from the furnace with a black coating although the indicator wire remained bright.

As this black coating did not dissolve in hydrochloric acid, while an oxide or sulfide coating dissolved quickly, the coating was assumed to be carbon. This assumption was confirmed by the following test: A 1½-in.

steel pipe, 4 ft. long, was filled with loose wire and inserted through the furnace wall and left there over night. The furnace gases were kept under a slight positive pressure so that the fire gases moved out through the pipe, preventing the air from coming in contact with the wire. The inside end of the pipe, located near the burners, was heated to about 600°–700° C.; the outside end was 500° C. and lower. The hard thin copper wire filling the pipe remained golden bright in the hot part of the pipe but it became entirely black in the colder end. Carbon was identified positively by an analysis of the black sample.

This test revealed two things: that the black substance was not copper oxide and that the same gaseous atmosphere can leave the wire bright or tarnished by a layer of carbon, depending solely on the temperature. It is known that carbon monoxide is decomposed catalytically by metal surfaces or other solids according to the reversible reaction $2CO \leftrightarrows C + CO_2$. At a lower temperature, say from 350° to 500° C., the reaction proceeds from the left to the right; at 650° and at higher temperatures, the reaction is reversed and the carbon is oxidized by carbon dioxide. The reversed reaction explains why a carbon-tarnished wire gets bright when kept in slightly reducing fire gases at 650° or higher. It was impossible to prevent the black carbon coating, particularly when annealing thinner wire on spools at a temperature around 400° F. so the retort furnace was again used.

A neutral or reducing gas for annealing copper may contain a small percentage of free oxygen and yet remain reducing for copper oxide. This follows, from the physico-chemical laws of reduction and oxidation of metals. Very probably at 500° C. copper oxide has a very small but measurable oxygen pressure. Water vapor filling the retort in standard copper annealing furnaces undoubtedly contains a small fraction of 1 per cent. free oxygen which was dissolved originally in the water of the water seal and carried over by evaporation; however, the copper remains bright. If the natural gas contains a small fraction of 1 per cent. of free oxygen and the copper gets black when annealed, it does not necessarily follow that the black coating is oxide.

I have not had an opportunity to experiment with natural gas but did make a small test with city gas. A strand of fine copper wire and a thread of pure asbestos cord free from organic matter were placed in a glass tube and city gas run through at a temperature of 350° to 500° C. In 15 min., the copper became black and in ½ hr. the asbestos too. The black substance on both was carbon which did not dissolve in hydrochloric acid.

Natural gas reveals a small percentage of carbon monoxide, especially if it contains admixed manufactured gas. Hydrocarbon gases are liable to thermal decomposition liberating free carbon. Unless there is proof to the contrary probably the discoloration described in the first three paragraphs might have been due to a carbon deposit and not to copper oxide; in those cases a nearer identification of the substance of the black coating is desirable.

When copper wire must be annealed at 350° C. in an atmosphere of a fuel gas containing carbon monoxide, a preliminary preheating of this gas in contact with copper at 600° C. might not prevent blackening of wire, because carbon monoxide is not eliminated.

The actual oxidation of copper wire annealed in open-fire gases is easily preventable by keeping the furnace atmosphere slightly reducing for copper oxide. The carbon deposit following catalytic decomposition of carbon monoxide may be prevented by using higher temperatures, which is not practicable if thin wire is annealed on spools.

There is another obstacle to be considered in open-flame annealing. In an open-flame furnace, the gases might be much hotter than the furnace walls and than the temperature indicated by large diameter pyrometer tubes; consequently, thin loose wire gets bright-red hot while the compact wire on the spools remains dark. The first will be overannealed, coarse crystalline and brittle, while the latter remains hard and underannealed. This result may be overcome by pulling the wire in single strands, through a reducing fire gas, at a temperature where no carbon deposit is formed; the entrance and outlet of the furnace should remain water-sealed. Thus the construction of the furnace would be simplified and cheapened as the complicated and costly conveying mechanism and the retort would be obsolete. Thin wire should be drawn through the furnace at higher speed and thicker wire at a lower speed and the temperature of the furnace should be kept above the decomposition point of carbon monoxide. A direct annealing would result also in a cheaper fuel cost and a uniform bright anneal could be obtained in a simple, cheap, retortless furnace at a reduced cost.

Thermal Conductivity of Some Industrial Alloys

By H. M. Williams* and V. W. Bihlman,† Dayton, Ohio

(New York Meeting, February, 1923)

In the construction of internal-combustion engines, the cooling of the combustion chamber and pistons is of great importance. In certain types of engines, the adequate dissipation of heat from the bearings is also important. One of the important factors affecting heat dissipation from these parts is the thermal conductivity of the material used. A search of the technical literature shows that, in general, the thermal conductivity of the elements[1] has been determined accurately by a number of investigators. The data on the industrial alloys are, however, meager. For an investigation of these alloys, the thermal-conductivity apparatus described in this paper has been constructed at the laboratory of the General Motors Research Corporation.

It is desirable that an apparatus for work of this nature should give fair accuracy of results with a convenient form of specimen and a reasonable rapidity of operation. A summary of the different methods of determining thermal conductivity is given in the first volume of the "Dictionary of Applied Physics," pages 429 and 470. The method selected as best satisfying these conditions is that of Griffith, on page 443, with the addition of a guard ring similar in principle to the method used by Berget (p. 445).

Briefly, the method consists of heating one end of a bar *T*, Fig. 1, electrically, measuring the amount of heat transmitted through the bar by a flow calorimeter *J* fastened to the other end, and measuring the temperature gradient along the bar. To compensate for the loss of heat laterally, the test bar is surrounded by a guard ring *E*, in the form of a steel pipe, which is electrically heated at the bottom and cooled at the top with a

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* Chief Metallurgist, General Motors Research Corp.

† Physicist, General Motors Research Corp.

[1] "Smithsonian Physical Tables." 7th ed., 213.
"Glazebrook: "Dictionary of Applied Physics," 443.
Lucke: "Engineering Thermo-Dynamics," 639.

water jacket *K*. The test bar is supported and located at one end by the stainless-steel socket *O* upon which the 500-watt electrical heater *D* is wound.

A convenient form of heater for this purpose is the sheath wire heating

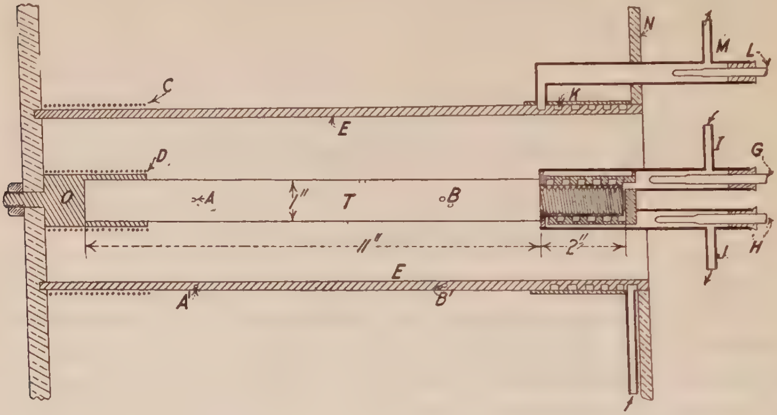

FIG 1.

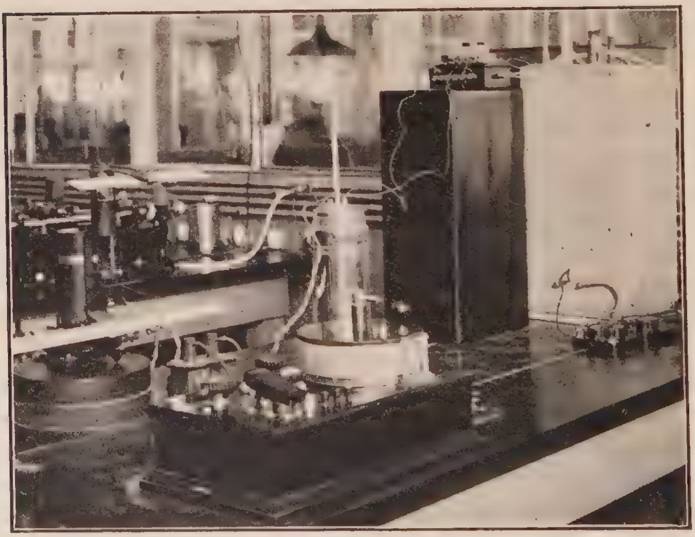

FIG. 2.

unit manufactured by the General Electric Co.; as the resistance wire is insulated from the steel sheath, the unit can be brazed directly to the socket. The continuous flow calorimeter is fastened to the other end of the test bar. The calorimeter thermometers *G* and *H* can be read directly to tenths of a degree Fahrenheit, the hundredth being estimated. The design of this particular form of flow calorimeter is due to Gordon B. Wilkes, professor of industrial physics at the Massachusetts Institute of Technology. The rate of flow of water to the calorimeter is maintained constant by the usual constant-level tank. Iron-constantan thermocouples (No. 28 B. & S. gage wire) for measuring the temperature gradient are inserted in small holes A, B in the test bar, 6 in. apart. The temperature of these points can be readily measured to ½° C. by the Leeds & Northrup type K potentiometer.

The lower end of the guard ring E is heated by the heater C and the upper end is cooled by the water jacket K. Two thermocouples are brazed to the pipe at A' and B' so that A' is in the same plane as thermocouple A, and B' is in the same plane as B. If the current in the inner and outer heating coils and the flow of water through the calorimeter and the jacket K are so adjusted that the temperature A is equal to A', and B is equal to B', the lateral flow of heat from the test bar will be zero; under these conditions, all of the heat is transmitted by the bar to the calorimeter. Because of the change of thermal conductivity with temperature, this is strictly true only if the test bar and the guard ring are of the same material or have the same temperature coefficient of thermal conductivity. This means, if any other bar but steel is being tested, even if A is at the same temperature as A', and B as B', that parts of the test bar will gain heat from the guard ring and other parts will lose heat to the guard ring. To minimize this effect, the annular space between the test specimen and the guard ring is filled with sil-o-cel packing; this packing also minimizes the interchange of heat between the calorimeter and the water jacket. The thermometer L, indicating the outlet-water temperature of the water jacket K, is used as a check on the possible interchange of heat between it and the calorimeter.

The entire apparatus is supported in a vertical position and enclosed in sil-o-cel packing. The external appearance of the apparatus is shown in Fig. 2 which includes the calorimeter, thermometers, the thermocouple wires A, B, A', B', the ammeters and rheostat for controlling the heating effect of the inner and outer coil. The potentiometer equipment is shown at the right.

Under the condition of no lateral flow of heat, the coefficient of thermal conductivity is calculated by the following formula:

$$K = \frac{M \times (t_1 - t_2) \times l}{(T_1 - T_2) \times a}$$

in which

K = quantity of heat, in calories, transmitted per second through a plate 1 cm. thick per square centimeter of its surface, when the difference of temperature between the two faces of the plate is 1° C.

TABLE 1.—*Mean Coefficient of Thermal Conductivity of Various Metals and Alloys*

	Temperature ° C.	K
Silver, pure	100	0.998
Aluminum, No. 1, commercial, hard drawn (Zinc, 0.07 per cent.; copper, 0.07 per cent.; iron, 0.30 per cent.; silicon, 0.30 per cent.; aluminum, 99.49, by difference)	79–193	0.47
No. 12 Aluminum alloy, all pig, sand cast (Copper, 7.61 per cent.; zinc, trace; manganese, 0.35 per cent.; iron, 0.56 per cent.; silicon, 0.31 per cent.; aluminum, 91.17, by difference)	94–232	0.33
No. 12 Aluminum alloy, all scrap, sand cast (Copper, 6.78 per cent.; zinc, trace; manganese, 0.36 per cent.; iron, 0.58 per cent.; silicon, 0.23 per cent.; aluminum, 92.05, by difference)	97–250	0.34
No. 12 Aluminum alloy, 50 per cent. pig, 50 per cent. scrap, sand cast (Copper, 7.30 per cent.; zinc, 0.03 per cent.; manganese, 0.41 per cent.; iron, 0.59 per cent.; silicon, 0.23 per cent.; aluminum, 91.44, by difference)	95–249	0.33
No. 12 Aluminum alloy, all pig, chill cast (Copper, 8.04 per cent.; zinc, nil; iron, 0.63 per cent.; aluminum, 91.33, by difference)	85–224	0.35
S. A. E. bearing alloy, No. 10 (Tin, 92.49 per cent.; copper, 3.58 per cent.; lead, 0.19 per cent.; antimony, 3.74 per cent.)	50–100	0.092
S. A. E. bearing alloy No. 11 (Tin, 86.92 per cent.; copper, 5.16 per cent.; lead, 0.12 per cent.; antimony, 7.90 per cent.)	50–100	0.062
S. A. E. bearing alloy No. 12 (Copper, 7.07 per cent.; lead, 63.94 per cent.; antimony, 28.84 per cent.)	50–100	0.076
S. A. E. bearing alloy No. 40 (Tin, 5.14 per cent.; copper, 84.93 per cent.; lead, 5.01 per cent.; zinc, 4.92 per cent.)	140–325	0.23
S. A. E. bearing alloy No. 62 (Tin, 10.55 per cent.; copper, 86.60 per cent.; lead, 0.04 per cent.; zinc, 2.81 per cent.)	155–340	0.142
S. A. E. bearing alloy No. 64 (Tin, 10.83 per cent.; copper, 79.04 per cent.; lead, 9.55 per cent.; phosphorus, 0.30 per cent.)	150–340	0.109
S. A. E. bearing alloy No. 66 (Tin, 5.56 per cent.; copper, 85.29 per cent.; lead, 8.26 per cent.; zinc, 0.89 per cent.)	140–320	0.177
Aluminum bronze (Copper, 89.87 per cent.; aluminum, 9.09 per cent.; tin, 0.47 per cent.)	130–350	0.174
Copper, hard drawn, electrolytic (Copper, 99.97 per cent.)	90–210	0.88
Copper, annealed, electrolytic (Copper, 99.98 per cent.)	90–210	0.90
Copper, cast electrolytic (Copper, 99.98 per cent.)	90–220	0.77
Copper, cast Lake (Copper, 99.76 per cent.)	90–225	0.76
Copper, hard-drawn (Copper, 99.44 per cent.; nickel, 0.20 per cent.; iron, 0.036 per cent.; arsenic, 0.231 per cent.)	100–230	0.52
Copper, annealed (Copper, 99.44 per cent.; nickel, 0.20 per cent.; iron, 0.036 per cent.; arsenic, 0.231 per cent.)	105–245	0.50

M = rate of water flow in calorimeter, in grams per second;
t_1 = temperature outgoing water from calorimeter, degrees C.;
t_2 = temperature of incoming water to calorimeter, degrees C.;
T_1 = temperature at point A, degrees C.;
T_2 = temperature at point B, degrees C.;
l = distance between A and B, in centimeters;
a = cross-sectional area of bar, in square centimeters.

A complete determination takes from 5 to 6 hr., a large part of the time being consumed in obtaining the heat balance and the establishment of thermal equilibrium. The readings used in the calculation extend over a period of about 5 min., the average value of $t_1 - t_2$ being used, the other factors remaining practically constant. Determinations made on electrolytic copper, the thermal conductivity of which is known, indicate that the results obtained with the above apparatus are accurate within about 2 per cent.

Table 1 gives the value of the mean coefficient of thermal conductivity (K) in c. g. s. units, between the temperatures noted, for a number of metals and alloys. All of the values given, except of that for silver, which was taken from Table 229 of the Smithsonian Physical Tables, were determined with the apparatus described.

This table shows that the conductivity of No. 12 aluminum alloy is not varied greatly by the method of casting or the proportions of new metal and scrap metal.

The thermal conductivity of the bearing alloys are in general low, varying from a maximum of $K = 0.23$ for S. A. E. alloy No. 40, to a minimum of $K = 0.062$ for S. A. E. alloy No. 11.

The determinations made on electrolytic copper and arsenical copper show that the thermal conductivity of arsenical copper is much lower than that of pure copper, although the arsenic content is but 0.23 per cent. This is similar to the effect of arsenic on the electrical conductivity of copper and could be anticipated from the generalization of Wiedemann and Franz, which states that the ratio between thermal and electrical conductivities is the same for all metals.

Density of Magnesium from 20° to 700° C.

BY JUNIUS D. EDWARDS AND CYRIL S. TAYLOR,* PITTSBURGH, PA.

(Cleveland Meeting, April, 1923)

MAGNESIUM is the lightest metal used for structural purposes, for which reason perhaps more than usual interest is attached to measurements of its density. Although the density of solid magnesium has been determined frequently, no measurements of the density of the liquid metal are available. Because of unusual experimental difficulties in measuring the density of the liquid, only a limited number of measurements of good precision were secured. The results, especially the value for the crystallization shrinkage, are of great interest in connection with the casting of the metal.

A number of samples for these measurements were made available through the courtesy of the American Magnesium Corp. They were in the form of extruded bars ⅜ in. in diameter and several were of exceptional purity, as indicated by careful analyses. Table 1 gives the density values (at 20° C.) for magnesium of varying degrees of purity.

The density was measured at room temperature by comparison of the weight in air, and when immersed in distilled water. The necessary buoyancy corrections were applied, so that the density could be expressed in absolute measure—grams per milliliter. The effect of the buoyancy of the immersed suspension and the surface-tension pull of the water upon the suspension were eliminated from the results by a preliminary weighing of the suspension immersed to the working depth. Great care was necessary to prevent the introduction of a significant error by the formation of gas bubbles upon the clean magnesium surfaces; these are sometimes very minute and easily escape observation. They may be detached by raising the specimen from the water and again immersing; this should be done just prior to completing the weighing. Their formation may be almost entirely prevented during a measurement by previously immersing the specimen in a solution of paraffin in petroleum ether. The thin film of paraffin remaining after the evaporation of the solvent prevents wetting of the magnesium surface by the water. In one such determination, the paraffin coating on a 22-gm. specimen weighed only 0.0007 gm. This is one part in 30,000 and could be neglected for most measurements. Experiments made with and without the film of paraffin showed that our results were free from any significant error resulting from the formation and adherence of gas to the test specimen. The measurements were made at the prevailing room temperature and were corrected to a value at 20° C. by use of the expansivity values of K. Scheel.[1]

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* Assistant Director of Research and Physical Chemist, respectively, Research Bureau, Aluminum Co. of America.

TABLE 1.—*Density of Magnesium at 20° C.*

Purity of Test Pieces, Per Cent. Magnesium	Density at 20° C. Grams per Milliliter	Remarks
99.99	1.7388	As extruded
99.99	1.7388	Same, annealed 2.5 hr. at 400° C.
99.95	1.7381	As extruded
99.94	1.7386	As extruded
99.90	1.7381	As extruded

The purity of the samples tested covers a range of one part in 1000; the variation in density is less than one part in 2000, and shows no regular relation to the purity. No change in density was observed as the result of annealing the purest metal for 2.5 hr. at 400° C. The density is appreciably lower than the value 1.75 frequently found in the handbooks.

In the measurement of the density of liquid magnesium, use was made of the densimeter devised by Frary and Edwards.[2] It is a kind of high-temperature pycnometer that automatically adjusts the volume of contained metal at the desired temperature. The apparatus first used was made of graphite, and is shown in section in Fig. 1. It consists of a small graphite crucible of approximately 40 ml. capacity which is closed by a graphite cover that is screwed tightly in place. A groove, about 0.5 mm. in diameter, in the top face of the crucible wall, closed on top by the crucible cover, forms a channel through which liquid metal can pass. The small crucible is placed in the center of a larger graphite container and the annular space is filled with molten tin to give a uniform temperature distribution.

The magnesium sample, in the form of a tapered cylinder similar to the inner crucible, but with the appropriate clearance to allow for expansion, is placed in the crucible and the cover screwed on. The apparatus is then heated in an electric furnace to the desired temperature and held at that temperature for 5 min. to insure a constant and uniform temperature throughout the sample. The temperature is measured by means of a calibrated platinum platinum-rhodium thermocouple suitably protected and immersed in the molten tin. The magnesium expands during the heating and melting; that in excess of the amount required to fill the crucible is expelled through the small channel, and usually appears at the opening in the form of an adherent globule; this is detached to prevent its being drawn back into the crucible upon cooling. When cool, the contained ingot is removed and weighed.

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[1] Holborn, Scheel, and Henning, "Warmetabellen der Physikalisch Technischen Reichsanstalt" (1919).

[2] J. D. Edwards, *Chem. & Met. Eng.* (1921) **24,** 61.

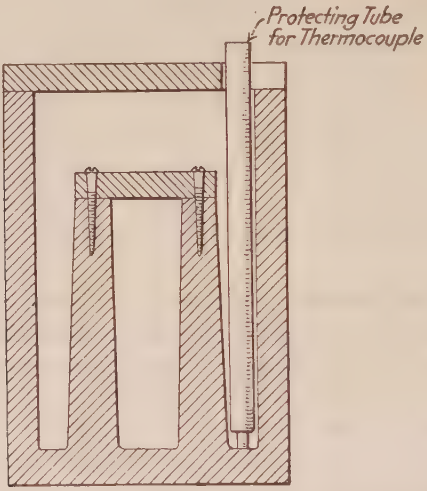

FIG. 1.—DENSIMETER FOR MEASURING DENSITY OF LIQUID METAL.

The volume of the crucible is measured at room temperature by means of the mass of mercury required to fill it; its volume at the higher temperatures is calculated from the expansivity coefficient of graphite. From the volume and weight of the ingot, the density is readily calculated.

Two concordant measurements were successfully made at temperatures 20° C. above the melting point of magnesium. When, however, measurements above 700° C. were attempted, some not unexpected difficulties were experienced. The magnesium seemed to evaporate into the graphite crucible; at least the graphite crucible was so firmly attached to the magnesium cylinder that the magnesium could be removed only by breaking off the crucible, and the last traces had to be scraped from the magnesium. Furthermore, there appeared to be evidence of some superficial reaction of the magnesium with the small amounts of gas in and penetrating the graphite walls. When pure magnesium is heated much in excess of its melting point in air, it burns quite readily.

Three measurements made at temperatures above 700° C. were sufficiently erratic to lead us to seek another material for the crucible. Chrome steel seemed the most satisfactory from the standpoint of resistance to any but surface oxidation at the temperatures to be used, so a crucible to hold magnesium was constructed from a "stainless steel" containing about 13 per cent. chromium. Its weight and volume were such that it sank in molten tin up to the level of the opening into the crucible. It could therefore be readily set in and removed from a graphite crucible containing a heating bath of molten tin. It differed from the graphite densimeter in that the latter was made in one piece while in the former only the inner crucible was made of stainless steel.

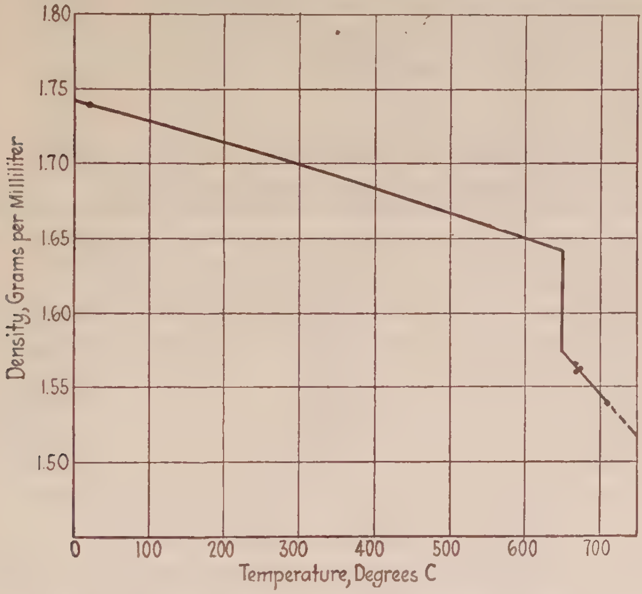

FIG. 2.—DENSITY OF MAGNESIUM FROM 20° TO 700° C.

The volume of the crucible was obtained by calibrating it with molten tin. The tin used was a standard sample from the Bureau of Standards, analysis of which showed its purity to be 99.99 per cent. The density of tin has been determined by Day, Sosman, and Hostetter, at temperatures up to 1600° C.[3] One determination of the density of liquid magnesium was made at approximately the same temperature as the first two measurements with the graphite densimeter; this showed very satisfactory agreement. A measurement at about 800° C. appeared to be perfectly successful, but the ingot stuck so tightly to the crucible that it could not be removed intact. A number of expedients for preventing the sticking by coating the inner surface of the cylinder have been suggested and some of them tested, but lack of time has prevented us from carrying the experiments further. The problem apparently is capable of solution, even though it presents many unusual experimental difficulties.

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[3] Day, Sosman, and Hostetter, *Am. Jnl. Sci.* (1914) **37,** 1.

The results of the tests are reported in Table 2. Only the measurements below 700° C. were completely satisfactory; no great reliance is placed upon the remaining tests. As first approximations to the most probable values, however, they do not appear to be very much in error.

The density values secured with the graphite densimeter are based on the density of mercury and the expansivity of graphite; the value secured with the chrome-steel densimeter is based upon the density values of tin at elevated temperatures, as determined by Day, Sosman and Hostetter.

TABLE 2.—*Density of Magnesium from* 20° *C. to* 700° *C.*

(Purity of magnesium, 99.99 per cent.)

Temperature, Degrees C.	Density, Grams per Milliliter	Kind of Densimeter Used
20	1.7388	
667	1.560	Graphite
667	1.565	Graphite
673	1.562	Steel
711	[1.538?]	Graphite
768	[1.445?]	Graphite
822	[1.478?]	Graphite

In Fig. 2, the complete density curve of magnesium is given up to about 700° C. The densities of the solid metal up to the melting point were calculated with the aid of the expansion formula of Scheel, which appears to be the best expansivity data available. The equation for thermal expansivity (20° to 500° C.) is $L_t\ L_o[(1 + 25.07t + 0.00936t^2)\ 10^{-6}]$. The crystallization shrinkage of magnesium, or the percentage change in specific volume in passing from a liquid of density 1.572 at the melting point to a solid of density 1.642 at the melting point, is approximately 4.2 per cent. The density of the liquid at the melting point is obtained by extrapolation over a very small temperature range, so that it should be reasonably accurate. The crystallization shrinkage of magnesium is somewhat smaller than that of aluminum, which is about 6.6 per cent.

The rate of expansion of the liquid magnesium appears to be somewhat greater than that of the solid—whether it is as great as the doubtful density measurements at temperatures about 700° C. would indicate, we have no means of knowing as yet. Because of the relatively small interval between the melting point (about 650° C.) and the boiling point, it would be reasonable to expect a high rate of expansion of the liquid.

DISCUSSION

Jesse Jones, Pittsburgh, Pa.—In the determination of the gravity of the solid magnesium, does surface oxidation offer any difficulties?

Junius D. Edwards.—The high purity metal can be immersed in water for a considerable period without tarnishing, and it is hardly necessary to coat it with paraffin. When molten, it seems to react superficially with the gases retained in the graphite, and there is a dark colored skin on the ingot which is removed from the crucible.

Jesse L. Jones.—Is the dark color the result of impurities in the oxidation product, or the way the light is reflected? It has been stated that the dark color is really due to impurities oxidizing out. Do you think it is due to the impurities or to the physical state of the oxide?

Junius D. Edwards.—In the case of magnesium, there was more or less the same colored skin outside as with aluminum. This bluish-gray film was probably a combination of oxide and nitride and was somewhat dirty, due to contact with the graphite. I do not think the impurities were sufficient in amount to play any very important part in coloring the metal oxidation products.

Action of Mud-laden Fluids in Wells

By Arthur Knapp, M.E., Philadelphia, Pa.

(New York Meeting, February, 1923)

The practical application of mud-laden fluids in wells has been the subject of many papers.[1] However, there seems to have been little investigation of what actually happens when mud-laden fluids are applied to a loose porous sand. The assumption is that the fluids penetrate the sands and that there is some plastering action. The degree of penetration and the relation of plastering action to wall stability is not generally known. This paper describes a series of experiments made in an attempt to observe, in laboratory apparatus, the action of mud-laden fluids under conditions simulating those to be found in a well. For the purposes of this discussion all fluids containing solid matter in suspension are mud-laden fluids.

Series I: Penetration of the Pores of Uncemented Sands

The subject of the penetration of porous formations by water is discussed in detail in Water-Supply Paper No. 67 of the U. S. Geological Survey. No attempt was made to repeat the experiments described therein. The results of the first experiment of this series are given simply to show the capacity of the apparatus used. The conditions in this first experiment were maintained throughout the remainder of this and the second series, except where noted. The experiments of Mills, McCoy, and others on subsurface flows, adjustments, migrations, etc., deal with the oil, water or gas within the porous beds.[2] The experiments described herein deal with the conditions immediately surrounding the bore hole.

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[1] I. N. Knapp: Use of Mud-laden Water. *Trans.* (1915) **51,** 571.

Lewis and McMurray: Use of Mud-laden Fluids. Bureau of Mines. *Bull.* 134.

Stroud: Mud-laden Fluids. Louisiana Dept. of Conserv. *Tech. Paper,* No. 1.

Tough: Methods of Shutting Off Water. Bureau of Mines. *Bull.* 163.

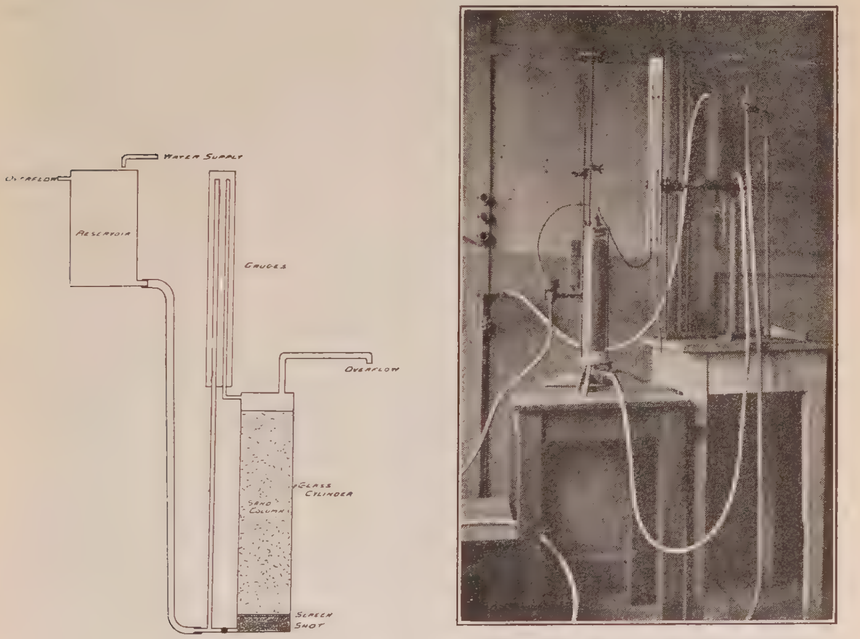
[2] Mills: Subsurface Relationships in Oil and Gas Fields. *Econ. Geol.* (1920), **15,** 398.

Mills: Relation of Texture and Bedding to Movements of Oil and Water through Sands. *Econ. Geol.* (1921) **16,** pt. 2.

McCoy: Principles of Oil Accumulation. *Jnl. Geol.* (1919) **27,** 255.

Water

The apparatus, shown in Figs. 1 and 2, consists of a reservoir in which a constant head may be maintained if desired. This reservoir is connected to the bottom of a glass cylinder 18 in. high and 2¾ in. in diameter. This cylinder contained the sand to be used in the observations. The bottom and top of the cylinder were connected to manometers so that the head of fluids on each could be observed. The bottom of the apparatus contained BB shot so that the bottom of the sand column could be seen; a piece of 40-mesh screen separated the sand and shot. The sand column filled the cylinder to within 2 in. of the top.



FIGS. 1 AND 2.—ARRANGEMENT OF APPARATUS.

The sand used was moderately rounded white quartz sand that passed through a 20-mesh sieve but was held on a 40-mesh; the grains ranged between 0.034 and 0.014 in. in diameter.

Sand column, height	15½ in.
Sand column, diameter	2¾ in.
Sand porosity	40 per cent.
Flow of water through pores	0.121 gal. per min.
Effective head on sand	13.2 in.
Loss of head through sand	12.3 in.

When wet sand was introduced into the cylinder, it was impossible to flow water through the sand column. The water, rising from the bottom as it was introduced from the reservoir, would drive the air, entrapped in the sand, ahead of it until sufficient air had collected to form a series of bubbles entirely across the cylinder. Enough bubbles would usually collect about 4 in. from the bottom. The surface tension of the bubbles was sufficient to raise the remainder of the sand column to the top of the cylinder and the flow of water would be shut off completely. Increasing the head to 10 ft. was not sufficient to start the flow and it was necessary to agitate the sand by stirring or reversing the flow suddenly, before all the air could be worked out of the sand. When most of the air had been worked out of the column, the water would flow freely through the pores. Exactly the same results were obtained when sand wet with oil was introduced and an attempt was made to flow oil through the pores.

These observations lead to the conclusion that the failure of wells to produce oil or gas may sometimes be explained by exactly such conditions. With weak pressures, it would be possible for gas bubbles to shut off the oil flow or for even a small quantity of water bubbles in a gas sand to shut off a gas flow.

Clay Fluid

The clay first used was yellow in color and mixed readily; it was being used for making a high grade brick in a nearby brickyard. This clay would settle rapidly for a few hours, afterwards there was no perceptible settling after several weeks. The settled mud was still liquid enough to flow readily.

The first mixture used weighed 11.4 lb. per gal. (1.34 kg. per liter). Water weighs 8.33 lb. per U. S. gal. An attempt was made to circulate this mud through the pores of the sand with an effective head of 13.2 in.; after the mud had penetrated about 3⁄16 in. sufficient solid matter had collected on the bottom surface of the sand column to stop further flow. The head was increased to 10 ft. without any further penetration. The formation of the film of mud across the bottom of the sand column caused the fluid to push the entire body of sand to the top of the cylinder. Therefore the bottom of the sand presented a surface unobstructed by any screen.

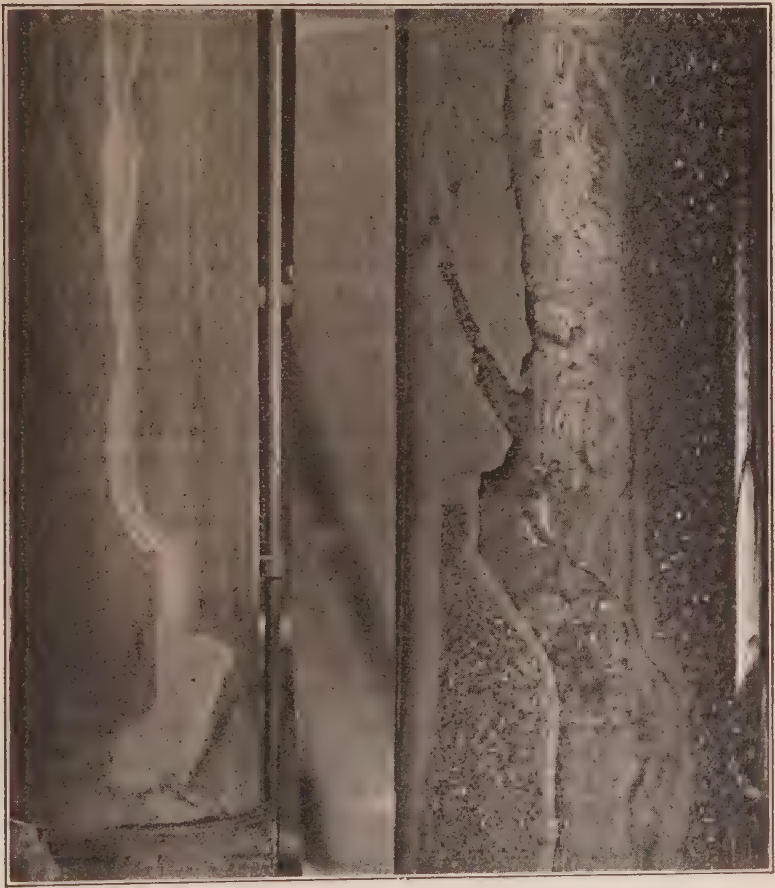
The same clay was then made into a mixture weighing only 9.65 lb. per gal. (1.6 kg. per liter). With a head of 13.2 in. this mud penetrated into the pores of the sand to a depth of 1¼ in. before there was sufficient accumulation to stop the flow.

In this instance, the flow was reversed and a water pressure sufficient to burst the glass cylinder was applied to the top of the sand but the clay could not be forced out of the pores and the flow started backward.

SERIES II: WALL-BUILDING ACTION

Clay Fluid

A second reservoir was added to the apparatus so that mud-laden fluid or water could be introduced at will by manipulating cocks. If the water was introduced at a pressure greater than about 14 in. head, the velocity caused eddies in the sand so that the water flowed through irregular channels as well as through the pores of the sand. When any mud-laden fluid was introduced at a velocity great enough to form eddies, the solid matter would be deposited on the edges of the channel and a wall would be built up until the upper surface of the sand column was reached. Fig. 3 shows a channel made by mud-laden fluid being forced in from the bottom. The first rush produced a rather large cavity for



FIGS. 3 AND 4.—CHANNELS IN SAND FORMED BY MUD-LADEN FLUIDS.

the first 2 in. From this point the mud rather quickly worked its way to the top, depositing a wall as it went. The distance that the fluid traveled into the pores is clearly seen. At the top, a small quantity of sand is inside the channel. The amount of sand not caught and walled up was directly proportional to the wall-forming qualities of the mud, as will be noted later.

In Fig. 3, the wall was first built up with a mixture of mud-laden fluid after which lime was run through to whiten the walls, so that they would photograph readily. Water was being circulated through the channel when the photograph was taken.

With clay mud in the channel, the walls were permanent whether the mud was circulated or not. When water was circulated slowly through the clay channel, a certain amount of the plastered clay would be washed away but a stable condition would soon be reached, after which there was no more erosion and water could be circulated indefinitely and remain clear. If the water circulation was stopped, however, the walls were not very stable and usually they would commence to cave almost immediately. Once caving started, practically the entire wall would disappear. The clay mud did not readily build a wall in a sand saturated with clay; it was necessary to wash the sand column clear with water before the clay mud would form another wall.

When water was introduced at a high velocity, it would slowly wash the deposit from the walls until the sand grains were uncovered, after which the wall would be eaten away immediately. Once the water made a hole and got behind the wall the whole channel above the hole would break and be washed away at once.

Cement Fluid

The fluid used in the first part of the experiment weighed 10.1 lb. per gal. (1.2 kg. per liter). When an attempt was made to circulate this cement mud through the pores of the sand, it would penetrate only ¼ in. before there was sufficient accumulation of solid matter to shut off further flow. If the pressure was immediately reversed, the cement showed some tendency to be forced out of the pores. There was no such tendency if the cement was allowed to remain in the sand even a few minutes.

When introduced into the cylinder at a velocity sufficient to move the sand particles, a channel with cement-lined walls was built up in a manner similar to the clay walls. Several well-marked differences were noted between clay and cement muds. The cement formed a wall much more quickly but with less penetration; this wall was stronger than the clay wall a few minutes after it had been formed. There was no plastering action. In the case of clay, a break in the wall of the channel, except when mud was being circulated, meant the destruction of the channel; in the case of cement, it was possible to collapse the wall and rebuild it with a clear water circulation. The cement washed out of the sand by the water was sufficient to form a wall. This shows the great wall-forming ability of cement. If the cement wall caved slightly when water was standing in the channel, the wall would often be repaired by the slight amount of cement present.

Fig. 4 shows a cement-walled channel. There is no plastering action as the print of the sand grains may be seen clearly in places in the channel wall. The channel is rough compared with the clay-walled channel.

When the wall is newly formed, a large amount of loose sand in the channel is quite likely to erode the cement wall because of the absence of plastered solid matter but the wall usually rebuilds itself. After the wall has stood for an hour or so it is stronger than any other wall made in these experiments.

Oxide-of-iron Fluid

Stroud[3] suggests the use of oxide of iron on account of its weight. It was found that a fluid composed of oxide of iron and water behaved exactly like cement, with the exception that the oxide does not set or settle until it is as hard as cement although it does settle until it is almost non-fluid. There was very little plastering action, as in the case of cement.

The author considers that it is out of the question to use iron oxide except for the purpose of drilling in. A very large percentage of the so-called corrosion in oil-well casing is the direct result of electrolytic action. The presence of iron oxide back of a string of casing or between two strings would produce about almost ideal conditions for corrosion. The iron oxide must be used only where it will ultimately be entirely expelled from the well.

Clay Fluid, Sand Wet with Oil

The same procedure was followed as in the previous experiments, except that sand was wet with oil in place of water and oil was circulated through the cylinder before the mud-laden fluid was introduced. If the oil was introduced at a velocity sufficient to cause the sand to move, the sand did not settle as tightly as in the case of water circulation. Oil of 30° Bé. and low viscosity was used.

It was necessary to introduce mud into the cylinder until the greater part of the oil was washed out and oil, sand, and mud in the cylinder were thoroughly mixed before a channel would be formed. The mud would not form a channel on the oil-wet grains until there was no excess oil in the pore spaces. It is not possible to form a wall in the usual manner, that is by sticking the grains together; there is no cohesion between the mud and the oil-wet grains and it is necessary to surround the grains with mud before there is any tendency toward wall building. Once the wall is formed, the plastering action takes place as before.

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[3] *Op. cit.*

Clear water could be circulated through the channel, once it was well-formed, but the introduction of oil into the channel caused its immediate total collapse. If the wall collapsed at any point when water was being circulated, the entire wall would collapse due to the slight amount of oil introduced through the break. Gas introduced into the channel also caused it to collapse, this was due to the agitation of the bubbles as they passed. The resistance of the walls to the passage of gas bubbles was a good indication of the strength of the walls in all of these experiments.

In order further to simulate well conditions, the oil-filled sand column was washed with clay mud. This removed all the free oil but the oil film upon the sand grains cannot be washed off with mud or water circulation. If oil was now introduced into the bottom of the cylinder, it would quite readily work its way through the sand pores even though the pressure on the oil was not sufficient to cause eddies.

In this series of experiments, this is the only case where there was found any tendency on the part of the oil to destroy the cohesive property of the mud-laden fluid, thus working through the pore spaces to the surface. In other words, the attraction of the clay mud and oil seems to be negative. This action did not occur when the sand was wet with water, showing that the oil film around the grains assists materially in the penetration of the oil and the displacement of the mud.

Cement Fluid, Sand Wet with Oil

Cement, even when very thin, will form a wall in sand wet with oil much more readily than will clay mud; in fact, the tendency was to form several small channels rather than one large one. These channels had the characteristics previously noted for cement. Clear water could be circulated through them and there was a tendency to build up places that were on the verge of collapsing, which was entirely lacking in clay mud. It was also possible to circulate oil and gas through cement-walled channels for some time before they collapsed. Fig. 4 shows a channel formed by cement in oil-wet sand. At the time the photograph was taken, clear water was being circulated. This channel had partly collapsed and had been rebuilt with clear water, using the cement washed from the collapsed wall. This accounts for the closed branch, which was the original channel.

Acid and Alkaline Muds

In the course of the previous experiments, sufficient acid was added to the various mud-laden fluids to give a good acid indication on litmus paper but no difference in the action could be observed. Both clay and cement were mixed with a saturated solution of salt but no change was noted in the action.

Mixture of Clay and Cement

A mixture of clay and cement showed strong cement properties as wall-building material even though the proportion of cement to clay was small. This would indicate that when drilling in badly caving material, the addition of even a small quantity of cement to the mud is beneficial.

Chemical Precipitates

The suggestion has been made that if one chemical were pumped into a sand and then a second chemical, which on mixing with the first would form a precipitate, the wall of a well could be cemented. When either ferric or cupric chloride were circulated through the sand pores and ammonium hydroxide was introduced, the circulation was stopped after about 1/16 in. of precipitate had been formed in the pores.

Iron hydroxide, resulting from the reaction of ammonium hydroxide and ferric chloride, was used as a mud-laden fluid and showed wall-building tendencies somewhat less than clay mud but much greater than lime mud.

Hydrated Lime

Lime has been suggested for use as a mud-laden fluid, but it was not possible to form a walled channel, in the apparatus described, with it. The lime would plug up the pores when the procedure of the second experiment of Series I was followed. With a different apparatus, which will be described later, it was possible to build up a wall if great care was taken but this wall was extremely fragile.

Infusorial Earth

The commercial name of the material used is kieselguhr. The fluid from this substance had remarkable wall-building properties although the weight of the mixture was little greater than that of water. The liquid after settling is very flocculent. When this mud was introduced, it instantly caused all the particles of sand with which it came in contact to adhere together. The channel was formed as quickly and as readily as though a glass tube had been thrust through the sand.

Kaolin

The kaolin used in this experiment was a commercial product made for use as fireclay. It was uniformly ground and practically all of it would go into suspension. There were no large grains or other material, as were present in all other materials used, so that it was not necessary to screen the fluid to remove particles that might clog the apparatus. Kaolin is a mineral having the formula $Al_2O_3.2SiO_2.2H_2O$. It is one of the most common mineralogical constituents of clay.

The kaolin mud showed marked penetration properties under low pressure; it was the only material examined that would migrate through the pores for any distance. When introduced into the cylinder, as in Series I, to test the penetration, this fluid would stop the pores and prevent further flow after entering the sand from ⅛ to ½ in. depending on the thickness of the fluid. But the material would continue to migrate another ⅛ in., spreading out until it appeared like a white fungus growth. The total penetration was never over ¾ in. for heads up to 10 ft. and fluids weighing over 9.5 lb.

The observed wall-forming tendencies of kaolin are nearly that of cement. There was a tendency to form branched channels in place of the usual single cylindrical channel. There was less tendency to reform a broken wall than was possessed by cement but much greater than was possessed by clay. The wall-plastering action was equal to that of clay. The resistance to the washing action of clear water on the plastered material was greater than that of clay; it was not affected by the passage of oil or gas through the channel.

Fig. 3 shows a channel formed by kaolin in an oil-wet sand. It would form a channel as readily in an oil-wet sand as in a water-wet sand. The channel would allow oil or gas to flow through without being eroded to the breaking point. The negative affinity for oil did not appear, as was the case of clay. This fluid was the only one that formed a channel for any distance inside of the sand column under normal conditions; all other fluids followed the glass from bottom to top. Kaolin would form a channel even though the sand column had previously been saturated with the kaolin fluid.

Mixture with Kaolin

Kaolin materially increased the ease with which clay would form a wall; the migratory penetration was still present to some extent. A mixture with cement showed a predominence of the properties of cement unless the amount of cement was very small.

Mud-laden Fluids with Oil Base

A mud-laden fluid mixture made with kaolin and oil was found to work as well in an oil-wet sand as the kaolin-water mixture. The channel would pass clear water or clear oil without being disturbed.

Water in Oil-wet Sands

Passages or channels are readily formed in oil-wet sands when clear water is circulated. If the cylinder is filled with dry sand and oil is circulated to wet the sand and then water is introduced, it will not wash the oil out of the sand but will form a channel having the appearance of the channel in Fig. 4.

This channel is irregular and there is considerable circulation of the sand near the channel for a short time. A film of emulsion appears on the surface of the channel after the water has flowed for about a half minute. This film grows thicker until the channel wall becomes permanent.

If such a wall was built up with a very thin mud, that is water just colored with clay or cement, the emulsion wall was elastic but very strong and could not be broken by ordinary means. A water pressure sufficient to break the glass cylinder did not give velocity enough to erode the channel walls. Water hot enough to crack the glass did not materially alter the channel. Thick cement or kaolin did not adhere to the walls. Thick clay mud would gradually wash away the emulsion and finally cause the channel to collapse.

Steam-mixed Mud

Mud is often mixed by a steam jet. In these experiments, steam-mixed mud was superior in every way to mechanically mixed mud. A batch of clay mud was mixed with a steam jet and run through the experiments of this series; the wall-forming properties of the clay were greatly improved. The mud had the appearance of a jelly after standing and no water appeared on the top after the mud had stood for two weeks.

General Remarks Concerning Series II

After noting the wide variation in the wall-forming properties of the various muds, an effort was made to find some criterion whereby field observations could be made as to the relative value, for mud-making purposes, of the materials found at any particular location.

The experiments of Series II were repeated and verified in the apparatus shown in Fig. 5, which is the simplest method the author has been able to devise. This is not a practical piece of apparatus to have in the field but further study may reveal some simple method that can be applied by the driller. It was possible to build up a walled channel in this apparatus by using a thick lime mud, although this was not possible in the original apparatus. This channel was very fragile and was eroded immediately if clear water was introduced.

At the top of Fig. 3, there is a quantity of free sand inside of the channel. As the mud-laden fluid arises in the cylinder and the wall is mudded off, there is always some sand in the center of the jet of mud that is moving too fast to be caught in the mudding-off action. When the jet reaches the top of the same column, a small quantity of sand continues to eddy as long as there is a circulation within the channel. The form of the top of the channel and the amount of sand depends on the wall-building properties of the mud-laden fluid and is one of the things to be observed in determining the relative value of this property. Fig. 6 is a diagram of the upper end of the typical channel with various muds. The dotted lines are the channel walls for each mud and the cross lines show the average amount of free sand that remains at the top of the channel.

The velocity of water that will move sand grains is astonishingly low. Sand grains ranging from 0.034 to 0.014 in. in diameter, which were used in these experiments, will rearrange themselves with a velocity as low as 4½ ft. per min. The grains would all float in water having a velocity of 1 ft. in 4 sec. and the smaller or lighter grains with a velocity of 1 ft. in 10 sec. The velocity of sand grains carried by mud-laden fluid would be proportional to the fluidity of the mud. For purposes of calculating the time required for cuttings to reach the surface, no allowance for lag is necessary. Practically, the cuttings would begin to reach the surface at the same time as the mud present when the cutting was done.

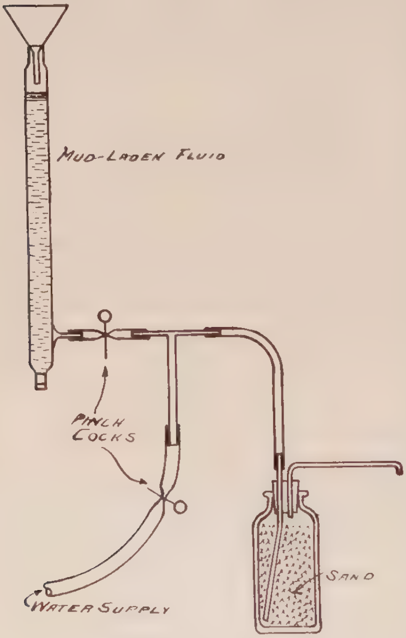

Fig. 5.

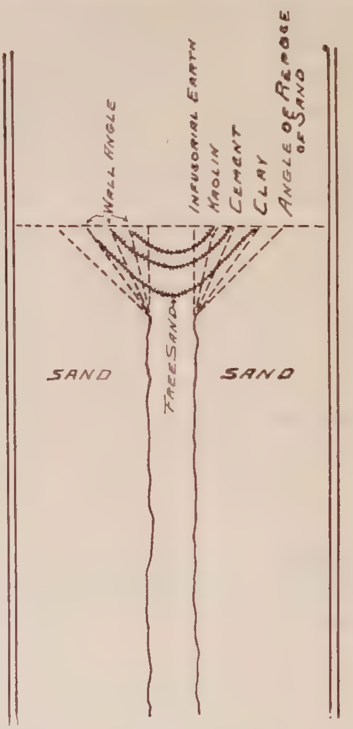

Fig. 6.

Series III: Penetration under Pressures

The experiments of Series I having indicated that the penetration of mud-laden fluids into the pores of uncemented sands was very small under the low pressures used, it was necessary to build a new apparatus to determine the action under higher pressures. Fig. 7 is a diagram of the apparatus and Fig. 8 a photograph of the apparatus as constructed. The cylinder for holding the sand was made of 4-in. pipe. The top of the cylinder was connected to a second cylinder, also of 4-in. pipe; this cylinder was connected to a pressure air line. The sand used was the same as in the previous experiments, except where noted.

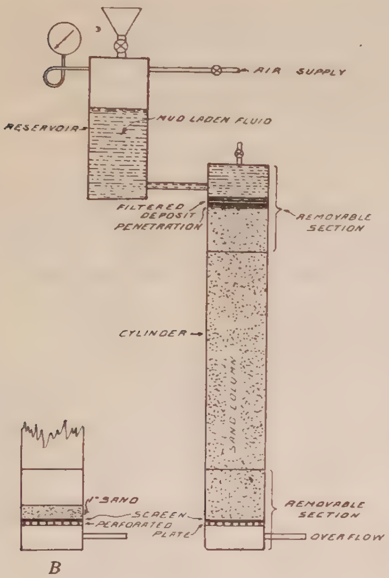

Fig. 7.

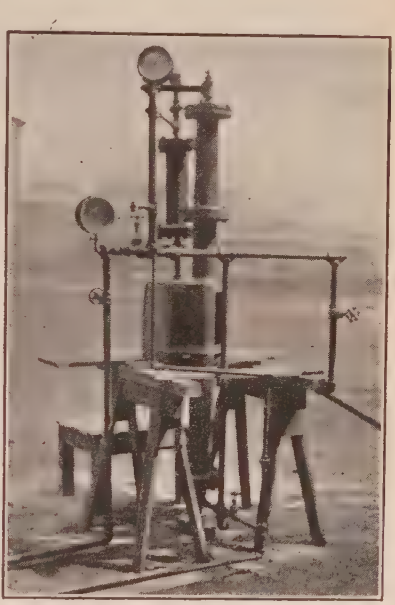

Fig. 8.

Clay Mud

Water was introduced into the reservoir to make sure that the sand column was unobstructed. It was found that the water would flow slowly through the sand by gravity and a full 1-in. stream would come from the overflow when a pressure of 7 lb. of air was applied to the water in the reservoir.

A clay mud weighing approximately 11 lb. per gal. was next introduced into the cylinder and a pressure of 80 lb. per sq. in. was applied for 15 min.; it was found that the mud had penetrated about ¾ in. The clay collected above the sand for about 1 in. in a non-fluid condition as the water had been filtered out of it.

The cylinder was jointed about 14 in. from the top, to permit examination of the sand column from within. In determining the penetration, a glass tube was inserted from the top and a core sample removed. Also, in order to be sure that there was not some slight penetration that was not visible, the tube was inserted from the bottom of the upper section of the cylinder and samples were taken every inch; these samples were dropped into clear water. Samples taken to within 1 in. of the top of the sand column showed no clay present.

Thin Clay Mud

A gallon of the 11-lb. mud used in the previous experiment was mixed with 1 gal. of water, making a mud which was much thinner than would ordinarily be used in drilling. The penetration of this very thin liquid was very little greater than the penetration of the thicker mud.

Kaolin

Kaolin showed the greatest penetration in the first series of experiments. In this experiment, a very thin mixture penetrated only $\frac{1}{2}$-in. with the formation of $\frac{3}{16}$-in. of semi-solid kaolin on the sand surface.

Migration under Pressure

The top 6 in. of the sand column was removed and 3 in. of sand was replaced with sand saturated with a thin mixture of kaolin; 3 in. of sand saturated with water was put in on top of this. The reservoir was filled with water and a pressure of 75 lb. applied; there was no movement of the kaolin through the sand. This could be determined as the overflow was so arranged as to be always full of water and any flow within the sand would produce a corresponding flow from the overflow. The sand column, in all of these experiments, was saturated with water so that any flow within the sand could be noted by the flow from the overflow.

Lime

A thin lime mud was applied to the sand under a pressure of 80 lb. but there was no appreciable penetration. As the lime became hard under the filtering action under pressure, the accumulation could be lifted from the top of the sand column and the top of the sand examined closely; the penetration was not more than the diameter of 3 or 4 sand grains.

Cement

Thin cement showed a penetration of about $\frac{1}{16}$ inch.

SERIES IV: FILTER PACKING

The apparatus shown in Fig. 8 was modified so that a short section of the bottom of the cylinder could be removed in place of the top section, as shown at *B*, Fig. 7. In conducting the third series of experiments, it was noted that the amount of solid matter above the top of the sand column was different for different muds. This material is deposited from the mud-laden fluids as the result of a filtering action of the sand plus the filtering action of this deposit itself as each successive layer is formed. The term "filter packing" has been adopted here to describe this action.

Filter Packing

In this experiment, 1 in. of sand was placed above the screen at the bottom of the cylinder. None of the muds investigated passed through this inch of sand even when only moderately thin mixtures were used; that is, muds weighing 9.5 lb. per gal. or more. Very thin clay and lime showed a slight color the instant the pressure was applied but no penetration after that. Each mud-laden fluid was introduced into the cylinder and a pressure of 80 lb. was applied until water no longer dropped from the bottom of the apparatus, showing that the filtering action had ceased. Fig. 9 gives the average results of this test. Some grades of cement gave as much as 14 in. of deposit. Clay after boiling gave only ¾ in. of deposit.

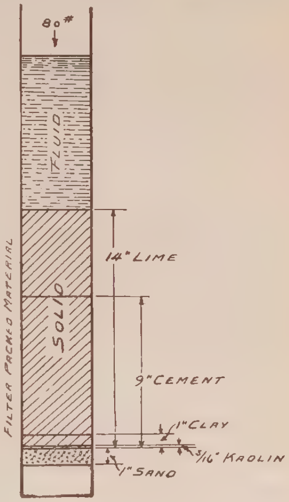

Fig. 9.

The amount of water that passed through the sand was nearly proportional to the volume of the filtered deposit. The deposit of mixtures of two materials varied directly as the proportions of the materials in the mixture. Equal parts of cement and kaolin gave 4½ in. of deposit and equal parts of lime and clay gave approximately 7 in. of deposit. The filter-packed material was no longer fluid but in every case the slug formed could be forced out in one piece by reversing the pressure and could be handled without breaking; it was about the consistency of soft putty.

The apparatus *B* (Fig. 7), was rebuilt to withstand a pressure of 500 lb. per sq. in., hydraulic pipe and special fittings being used. In the first run, clay mud was introduced and subjected to 80-lb. pressure; the water ceased to flow after 1 in. of filter deposit had been formed. The pressure was then increased to 500 lb. and this pressure held for 15 min. No fluid was forced out by the additional pressure and no more filter deposit was formed. The deposit still had the consistency of soft putty and the mud above the deposit was fluid and showed no change.

A cement mud was then tried; there was no flow from the high pressure and no additional packing of the material.

As a check to the cement experiment, a small quantity of the filter-packed material was removed from the top of the deposit and water was poured in on top. This was then subjected to 80-lb. pressure and the water filtered readily through 14 in. of wet unset cement that had previously been under 500-lb. pressure.

Oil Penetration of Filter Deposit

A clay mixture was subjected to pressure until the filter deposit was so thick as to be at the point of shutting off the penetration of water completely. Oil was then introduced in the place of the fluid mud on top of the deposit. It was found that the oil would penetrate readily. At the time the pressure was removed from the clay mud, the water was dropping at the rate of 20 drops per minute. The oil under the same pressure flowed at the rate of 100 drops per minute.

General Remarks

There is a wide variation in the materials that may be classed as clays for mud-making purposes so that observations may be made on some clay muds that differ from the foregoing. As many different clays were used as were available, including mud from a river bottom, yellow clay, black loam, and red earth containing iron; no appreciable difference in the action of these various clays was noted. An alkali earth to correspond to the surface material of some of the California fields was not available. Wherever there was any doubt as to the action, several weights of fluid were used. Sands of different fineness of grain were tried. All the conclusions given are for sands ordinarily encountered in drilling and for all ranges in the weights of muds commonly employed.

In connection with the penetration of very porous material by mud-laden fluids, the following may be of interest. Suppose that, in the course of drilling, a gravel deposit is encountered and that mud is lost in mudding off this formation; assume that the conditions are:

Reduction in the volume of mud in the slush pit	1 cu. yd.
Size of bit	9⅞ in.
Amount of hole drilled	20 ft.
Porosity of gravel	40 per cent.

Also, assume that penetration was equal in all directions from the bore hole; in other words, that the mud formed a cylinder in the gravel concentric with the hole.

Under the foregoing conditions, the diameter of this cylinder would be 22 in., or the penetration would be 6 in. If the slush pit is 8 by 16 ft., a 1-in. loss in drilling 20 ft. would be just sufficient to fill the hole drilled. A second inch loss would mean a penetration of 5 in.; an additional foot loss means 2 ft. 4 in. penetration.

It should be remembered that this penetration would be the same for 40 per cent. porosity, no matter what the size of the pores might be; greater porosity would mean less penetration for any given loss of mud. It is further to be borne in mind that, in many cases, a loss of mud does not mean that the mud has penetrated the sand but that the hole has caved and an extra quantity of mud is necessary to fill the enlarged hole.

The loss of 1 cu. yd. of mud in drilling 20 ft., in the case above, may mean that the average diameter is 16 in. in place of 9⅝ in., the size of the bit. The loss of the foot, or 13 in. in all, from a 8 by 16 ft. slush pit, may mean the average diameter is 2 ft. 10 in.

CONCLUSIONS

The results of these experiences leads to the following conclusions:

1. The penetration of the mud-laden fluids into the pores of the wall of a well is very slight, not over 1 or 2 inches.

It is not possible to pump mud back into the ordinary porous formation. The addition of pump pressure to the hydrostatic head merely adds a small amount to the filtered deposit and a small quantity of water is introduced into the sand. When the circulated fluid appears in wells at a distance from the drilling well, it is because natural crevices have been encountered or channels have been formed by the fluid in a loose sand.

2. The stability of the wall of a well depends on:

(*a*) The cohesion imparted to the sand grains by the mud-laden fluid; (*b*) the formation upon the surface of the hole of a plastic solid deposit of material from the mud-laden fluid by filtration; (*c*) the plastering action where filtering is not possible; (*d*) the static head of the fluid column.

Of these, the first is important for such time as the drill is penetrating a formation. After the bit has passed below any given point, the filtering action or plastering action quickly provides a deposit of sufficient strength to hold the wall. The plastering action is important where there is no filtering action. Filtering action is only possible when there is sufficient head to overcome the pressure in the formation and it is necessary for the fluid in the pores to be able to move back. In the case of some, but by no means all, formations it is possible for the fluid in the pores to move back, there being an outlet to the surface at some distant point. In the case of sealed formations, such as lenticular sands and others, where there is no outlet, the fluid cannot move back into the formation and there is no filtering action. Static head has no effect on wall stability except when filtering action is possible, in other words only porous formations under favorable conditions.

3. The resistance offered by the wall-forming material to erosion has a great deal to do with its utility as a circulating medium.

Pure cement is not a good mud-laden fluid as its plastering action is very slight and non-porous but fragile materials are not protected from erosion. Cement in a well very soon becomes mixed with enough clay to increase the plastering action greatly so that under ordinary circumstances the mixture that will be used in a well is an ideal fluid. Lime, on the other hand, gives a good plastering action but, even when mixed with large quantities of clay, the plastered deposit is so soft as to be subject to excessive erosion. Filter-packed lime is easily eroded.

4. In bringing in a well in which the productive horizon has been mudded off, it is necessary to destroy mechanically the well wall to the depth of several inches, as the mud-laden fluid cannot be forced out of the pores and the filter-packed material forms a continuous arch that is strong enough to resist considerable pressure, and does not erode or wash away readily.

In this case the well wall includes the filter-packed or plastered material. When this part of the wall is broken or removed, the amount of the original formation it is necessary to disturb is practically the penetration, which is very small, in most cases probably an inch or less.

5. The fact that a mud-laden fluid will settle rapidly does not condemn it as a circulating fluid, provided its other wall properties are good. It may condemn the fluid for drilling purposes if the settling is so rapid as to tend to stick to the tools or casing.

6. It might greatly facilitate the bringing in of a well if a few barrels of light oil were pumped into a well as a final flush just before the screen is unhooked and also if a few barrels were pumped on the inside of the screen as a wash pipe is pulled.

The oil appears to have an active cutting action on the clay mud so that heavy mud could be more easily washed out if cut by oil.

7. While there is evidence that wells have been drilled through productive horizons and the oil shut off completely by the mud-laden fluid used in drilling, it is probable that some of these wells would have been commercial producers if washed and bailed for a sufficient length of time or if they had been bailed down and allowed to stand for a sufficient length of time to allow the oil to work its way through the mud.

If the productive sand is very porous and the pressure within the sand is very low, the filter-packed material may be very hard. When this material is behind a liner, considerable washing is necessary to break and remove it.

8. Uncemented oil sands should be drilled into with a thick well-mixed mud.

When the mud is thinned just before entering the oil sand there is an excess of water, which may be forced into the sand before there is sufficient filter-packed material to shut it off. The thin muds are more liable to form emulsions with the oil. The absence of oil showings in the overflow, while drilling through a productive horizon, may be due to the formation of an emulsion that adheres to the surface of the oil sand or the casing or the wall of the well.

9. The failure of an oil horizon to produce oil may sometimes be explained by the fact that water or thin mud will form a tenaceous coating or emulsion on the oil sand. This wall is particularly hard to break.

10. The washing of a well with heavy, well-mixed clay mud will tend to destroy any emulsion that has formed on the oil sand and obstructs the flow of oil to the well and may assist in bringing in the well.

Clear water will remove the mud, while it will have no effect upon the emulsion.

11. When the walls of a well have caved badly, for any reason, it would be much better to allow the well to stand for a few hours than to try to redrill the hole and stop the cave immediately.

There would be more chances of getting a stable wall than there would be if the caved material were redrilled at once. Also, less sand would be removed, because it is difficult to build up a wall in a loose sand which is saturated with clay mud. The caved material, being to some extent saturated with mud, would be washed out and the wall finally formed behind. If the sand is allowed to settle, all of the caved material will probably remain in place and be walled off.

12. The hole drilled in uncemented oil sands is greater in diameter than the average diameter of the rest of the hole, because of the difficulty which clay mud has in building up a wall on oil-wet sands.

13. Even a small quantity of cement added to clay mud increases the stability of the wall considerably and has no detrimental effects.

Caving may be stopped more quickly by the addition of a small quantity of cement to the mud than by attempting to thicken the mud from the storage pit.

14. The resistance of well-established walls to the erosion by water leads to the belief that muds may be safely thinned much more than is usually considered practical when drilling clay, gumbo, or gypsum.

15. In general, the finer the solid particles in suspension in a mud-laden fluid the better are its wall-building properties.

16. If materials other than clay are available, such as oxide of iron or kaolin, they may be used to great advantage in "lubricating" wells making oil, as they are not affected by the oil to the same extent as clay.

17. It is possible for water to form channels through loose uncemented

sands through which the water will reach the well from long distances, thus bypassing large quantities of oil.

The velocity of water through a sand which will move the sand grains is extremely small. Every one is familiar with the boiling of the sand grains in the bottom of a spring even though there is little flow. It is possible that the "coning" of oil-field waters may not in every case be a symmetrical cone caused by capillary resistance or other means. If channels were formed, the difference in friction between the travel through the pores and through channels would account for a difference in water level.

In the case of the Kern River field, the conditions described by Stevens in the Summary of Operations of the California State Mining Bureau, October, 1921, may be coning in the accepted sense; but as large quantities of sand have been removed from parts of this field, there is room for extensive movement of the sand and for the formation of channels by the water. In the third ideal cross-section on page 13 of Stevens' article, there are conditions that could readily lead to channeling.

18. The total deposit on the wall of a well is independent of the thickness of the circulating fluid, within the limits of the muds ordinarily used for drilling or mudding off.

19. The thickness of the wall deposit is independent of the depth.

20. The thickness of the deposit is different for different mud-laden fluids.

21. Although the wall of a well may be so sealed with filter-packed material as to prevent further infiltration of water, it may not be sealed against the filtration of oil under the same pressure.

22. Steam mixing improves the quality of clay muds.

Steam mixing may be used to advantage where the raw product is poor; also, where it is desirable to increase the quality of the mud as when large bodies of loose sands are expected or in mudding-off processes when the pressure of gas or underground waters is excessive.

PRACTICAL SUGGESTIONS

1. Oil added to the circulating fluid might aid in loosening a drill pipe stuck in the hole. It sometimes happens that the rotary drill pipe is stuck even though a perfect circulation is still possible. It is suggested that, as a last resort, the drill pipe be filled with oil and the slush pit ed with a mixture of mud and cement. As the oil is pumped into the hole it will tend to destroy the wall of the well. It is probable that the in portion of the wall that has caved and stuck the pipe will be destroyed first. The cement following the oil will tend to rebuild the wall and counteract the action of the oil. In this manner, it might be possible to

recave the well at the point where it is holding the drill pipe and thus free it.

2. F. B. Tough[4] cites the case where hydraulic lime and manure were used with mud to restore circulation lost in gravel, crevices or fissures. It is possible to pump a high concentration of sand and mud or sand, mud and cement. It is suggested that, before going to the expense of hauling lime and manure, the cuttings from the drill hole or other sand or surface material that will settle rapidly, be mixed with a batch of mud and a small quantity of cement and pumped down the hole. After such a batch has been pumped in, the bit should be pulled well off the bottom and the pump slowed or stopped for 10 min., if the well is not caving. This would give the sand and cement a chance to settle and it is probable that further pumping would not produce sufficient velocity within the crevices to move the settled sand. Several such batches should materially aid in closing ordinary gravel or other excessively porous formations.

3. Washing a well before cementation is beneficial. The washing should immediately precede the cementation and should not be prolonged. If water is used, it should be introduced very slowly. The washing will not remove the filter-packed portion of the wall and will not erode the plastered portion unless the velocity is excessive and the washing prolonged. The casing should be slowly rotated by hand during the entire period of washing and cementing. After washing and before cementing, the casing should not be raised or lowered unless absolutely necessary.

The benefits derived from the clearing of the hole of all excess mud by using clear water are off-set by the advisability of having mud behind the casing to seal off strata above the cement. A thin mud will clear a large portion of the heavy mud from the well and it is suggested that thin mud is better than clear water when all factors are taken into consideration.

Relative Value of Various Materials; Clay = 1

The following values are from the observations made during the experiments.

	Penetration	Cohesive Wall-forming Property	Plastering Action	Wall Resistance to Clear-water Erosion	Wall Resistance to Gas Flow
Clay	1.0	1.0	1.0	1.0	1.0
Lime	0.9	None	0.1	0.0	0.0
Cement	0.5	2.5	0.0	10.0	10.0
Kaolin	2.5	5.0	1.2	8.0	7.5
Infusorial earth	0.2	25.0	0.8	2.0	2.5
Iron oxide	1.0	2.0	0.8	1.0	[a]
Flocculent chemical precipitates	0.5	0.5	0.5	0.1	[a]
Steam mixed clay	1.0	5.0	1.2	4.0	[a]

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[4] Bureau of Mines. *Bull.* 163.

[a] Not determined.

	Cohesive Wall-forming Property in Oil-wet Sand	Limit of Weight of Mud for Practical Use	Fluid Properties of Mud after Long Settling	Wall Reforming with Clear Water	Thickness of Filtered Deposit
Clay	1.0	1.0	1.0	0.0	1.0
Lime	0.0	0.7	8.0	0.0	14.0
Cement	2.5	1.5	Hard	10.0	9.0
Kaolin	10.0	1.5	2.0	1.0	0.2
Infusorial earth	[a]	1.5	10.0	8.0	[a]
Iron oxide	[a]	2.0	Very viscous	1.0	[a]
Flocculent chemical precipitates	[a]	[a]	[a]	[a]	[a]
Steam mixed clay	2.0	1.0	2.0	0.0	0.8

[a] Not determined.

DISCUSSION

Ben K. Stroud,* Shreveport, La. (written discussion).—The author has brought out, through his experiments, some important conclusions; particularly the experiments as to penetration under pressures. These tests are fairly closely approximated in actual practice. A mud-laden fluid used in oil-field work has little penetration in the solid strata, as in limestones, shales, and clays. There are cases, of course, where crevices or fissures or a loose water sand may admit the pumping of large quantities of mud, *i.e.*, lost circulation. In Louisiana some water sands are so fine that they are carried along like the water itself and an angle of repose is never reached. They invariably give trouble in mudding off, particularly where a stray gas pressure might be found. The latter condition now obtains in portions of the Smackover, Arkansas, field and is causing a great deal of trouble.

By weighing many samples of oil-field muds, we were surprised to find in Louisiana that the drillers were all overestimating the weights and that the average oil-field mud, even when very thick, only weighed a little over 10 lb. per gal. This showed that much of our so-called clays are unfit for making heavy muds, where such are necessary to control heavy gas pressures; after some experimenting we tried iron oxide, the use of which has been fully justified by the extremely favorable results reported by operators in Texas, Arkansas, and Louisiana.

A few weeks ago I received a message from an operator stating that a blow-out was threatening in a well which he was drilling in the Monroe field and which was an offset to a gasser that had just been completed. His message stated that the blow-out came at a depth of 150 ft. and that he had shut down pending an investigation by the Department of Conservation. On investigation, we found that, in some manner, a large quantity of air had collected in the hole and that it partly expelled the column of mud from the well. From this case the question would arise: how much air have we in our drilling muds under average conditions and what will be its action in aiding or hastening corrosion of casing? It is well known that the formations of Texas, Arkansas, and Louisiana contain certain amounts, and often solid beds, of iron pyrites. Might not the oxygen in the air that has been trapped by the mud start a chemical action in connection with the iron pyrites and in the presence of bituminous shales which would result in sulfuric acid? If so, it might be the source of some of our corrosion troubles.

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* Superior, Division of Minerals, Department of Conservation.

F. N. SPELLER, Pittsburgh, Pa. (written discussion).—By electrolytic action (page 1081, paragraph 4) I assume that the author refers to the influence of contact of two dissimilar materials, such as mill scale and iron. This undoubtedly accelerates local corrosion in water when the scale is fused on to the metal in the welding furnace. The voltage generated between such materials is extremely small so that unless there is very intimate contact, as in the case of mill scale on steel, the resistance prevents appreciable current from flowing, and of course, therefore, corrosion is not accelerated. Furthermore, ferric oxide is not nearly so electronegative to iron as is the magnetic sesquioxide of iron. It is doubtful whether ferric oxide in the fluid would be in direct contact with the casing. All things considered, therefore, I do not see how the presence of some ferric oxide in the mud would materially influence corrosion. This would probably have less effect than an increase of a few grains per gallon of soluble salts or a sudden change of wet strata in contact with the casing, or many other variable factors that affect corrosion. It is common to find iron pipe embedded in soil that has become strongly impregnated with ferric hydroxide from the pipe. This forms a more or less impermeable protective coating so that under these conditions the ferric hydroxide is largely responsible for preserving the metal from further corrosion.

G. M. PONTON, Spring Hill, La. (written discussion).—This paper cannot be too highly recommended. The author's attempt to duplicate actual field practice with laboratory equipment has been most successful and his conclusions are cleverly drawn. Much more work can be done along the same lines, and I trust others will extend the investigation, if not with laboratory work, by the recording of results obtained in special cases with mud-laden fluids.

In North Webster Parish, Louisiana, the producing sand (Blossom Sand) is found at about 2700 ft. From the surface to 1150 ft., little mud is made and weighs (weights are given for average heavy drilling fluids) 9½ to 10 lb. per gal., but is poorer in other respects, having much fine sand, lignite, and gypsum. From 1150 to 1650 ft., much mud is formed and a big mud reserve can be built up. The mud weighs 10½ to 11 lb.

per gal. and is clean gumbo. From 1650 to 2700 ft., except for a few thin strata, the muds do not weigh over 9 lb., and because of the presence of chalk, sand, etc., is very poor. Besides, it usually has much salt water, from which the above good mud is free.

Drillers generally have called muds heavy because of their appearance, but the practice of weighing samples is being adopted. This change is being effected by wells blowing in unexpectedly, causing craters in some cases, in others preventing the proper finishing of the wells with liner, etc., and in others necessitating lubrication so as to drill deeper to the oil sand.

The use of iron oxide as a control fluid has proved a great boon to operators dealing with high pressures in wells, such as are found in North Webster Parish where the pressure is about 1200 lb. per sq. in. The author suggests that iron oxide might set up excessive corrosion, if left behind casing; personally, I see no reason for leaving the iron oxide behind casing. Much study should be given to corrosion of pipe, especially when an examination of the fluids used in North Louisiana and South Arkansas shows a great diversity of combinations of pyrites, gypsum, lignite, and other bituminous material, glauconite and waters with various mineral contents and with air always present in the fluids.

In North Webster Parish, the muds formed between 1150 and 1650 ft. are set aside in a reserve pit. When setting the last string, the well is thoroughly flushed, then the mud is steamed to expel the air (bituminous material, thread dope and most of the pyrites are expelled, the last by flotation), and after six sacks of cement are mixed with the slush pit mud, the fluid is run behind the casing. The cement has been added merely to increase the weight, but the author has shown that the cement has other important uses.

The value of steaming and adding cement to the fluid was shown in the drilling of Pine Woods No. 4 well. From the surface to 200 ft., sometimes to 400 ft., there is a semi-quicksand. In this well, the surface casing could not be set until the mud was steamed and mixed with cement.

In the North Webster field, at least eight wells have blown in unexpectedly, and practically all these disasters could have been prevented with a proper knowledge of mud-laden fluids. With the high pressure obtaining here, only two methods of safety exist: first, the use of a heavy control fluid, such as iron oxide; second, keeping a heavy mud in circulation at all times.

Pine Woods No. 2 well blew in after a shutdown of three days, with very heavy mud in the hole. It was first controlled by a 6-in. casing set at 2595 ft., which collapsed at 2200 ft. It was then controlled by the 8-in. set at 1740 ft. which, in turn, collapsed. This 8-in. casing was destroyed almost entirely from 1000 ft. to the surface by sanding. Eventually the well was controlled by running a 3-in. drill stem to the bottom and mudding down with high pressure pumps. Later, this mud was replaced with pure cement, which came to the surface inside and outside the 6-in. casing, the drill stem being left in the hole. At first there was no sign of gas; but as the cement hardened, gas showed through the cement inside and outside the 6-in. casing and through the 3-in. drill stem.

The following wells have blown in unexpectedly but it has been difficult to get much information as to the general conditions:

Gleason No. 1, after a short shutdown, with medium mud.

Gleason No. 2, after a short shutdown, with heavy mud.

Raymond No. 1, while shutdown for luncheon, with light mud.

Pine Woods No. 2, after three days shutdown, with heavy mud.

Morefield and Tanner No. 1, after 2 hr. shutdown, with medium mud.

Boone No. 1, while drilling with light mud.

Webb No. 1, while drilling with light mud.

Pine Woods No. 3, heavy mud was used; the drill stem was never out of hole more than ½ hr., and was rotated with the pumps running slowly; all operations, including taking of cores, were made, but the well blew in 12 hr. after the liner was set in.

The Raymond No. 2 was drilled through the high-pressure sand with the use of iron oxide, shutting down at will with no blowout.

The Munn well, perhaps the largest well in the field, was bailed dry with no sign of pressure and was only brought in after washing with clear water. In this case I believe the liner held the walls mudded off.

The paper shows the difficulty of mudding off an oil-wet sand. In Webster Parish, even the gas sand has more or less oil. It may be the presence of this oil that makes blowouts prevalent in the field. The use of kaolin might correct this situation.

The author, on page 1086, says: "Practically, the cuttings should begin to reach the surface at the same time as the mud present when the cutting was done;" also, "No allowance for lag is necessary." While this is generally true for sand, especially fine sands, it is not true for cuttings other than sand, which in well drilling are just as important as sand. Shale, lime, and similar cuttings cannot be relied on to show the formation being cut. This I have demonstrated many times by cores and after cores were taken. The time that elapsed for mud to be pumped to the bottom and returned was carefully checked by coloring some of the mud with indigo. Even the returns with respect to sand cuttings give most unreliable results, in that near the sands often are found sandy shales. The lag of cuttings are dependent on well conditions, amount of casing above the cutting, amount of open hole, thinness of the mud, speed of the mud, size of drill stem, etc.

Ralph Arnold, Los Angeles, Calif.—One California field has three sands, the deeper sand being the more prolific. The oil companies are eager to go into the third sand, where possible, so the Oil and Gas Inspection Service has permitted drilling through the first and second sands, with the rotary, and mudding off these and the water sands. Many oil men question whether this will protect the two upper sands from the waters that lie above them and possibly are interpolated with these sands.

Arthur Knapp, Philadelphia, Pa.—One thing that I did not discuss was the mixture of these mud-laden fluids. When clay is mixed with kaolin, for instance, the clay features are predominant; when mixed with cement, the cement is very much in evidence. For this reason, where the mud must be relied upon for other than mere wall stability, it should contain some cement—it is then heavier, less susceptible to the absorption of gas or air, is less liquid, makes a little thicker filter deposit, and settles to a harder condition. Everything is in favor of a mixture of clay and cement.

W. E. Wrather, Dallas, Tex.—Is the walling-up of the well purely a matter of the physical constitution of the materials used, or is there any chemical action traceable to the use of lime, for instance, or siliceous material, granting that they are of equal degree of fineness?

Arthur Knapp.—The suggestion has been made (I think it was patented) that one chemical be pumped into a sand and then a second chemical, which would combine with the first and form a cement. I tried that plan but found that the cement would penetrate the sand only a few grains before it would effectively shut off the flow. So that if there should be any reactions, I do not believe the reactions would be effective in the active wall; that is, the wall that is sustaining the well before the casing is set.

In making these experiments, I had only the material that was available at Philadelphia. I used all the clays I could obtain; even some mud that was dredged from the bottom of the river. This was a red clay which is evidently full of iron. I also tried some very fine clays that are used for a fine grade of brick. Some of the muds used in the field, however, I could not get, particularly the alkaline found on the surface in California. I would be glad to receive some dry samples of the mud being used in the wells, for next summer I intend to make another series of experiments, when I shall use these muds.

Samuel S. Arentz, Simpson, Nev.—Have you carried out any experiments with Bentonite?

Arthur Knapp.—I recently received a large sample of Bentonite which I am planning to use next summer.

The Sunburst Oil and Gas Field, Montana

By Dorsey Hager, Great Falls, Mont.

(New York Meeting, February, 1923)

One of the most interesting fields in North America was discovered, early in 1922, near the towns of Sunburst and Kevin, Toole County, Mont. This field produces oil from the Jurassic beds, which heretofore have been of but little importance as oil producers, and promises to be the

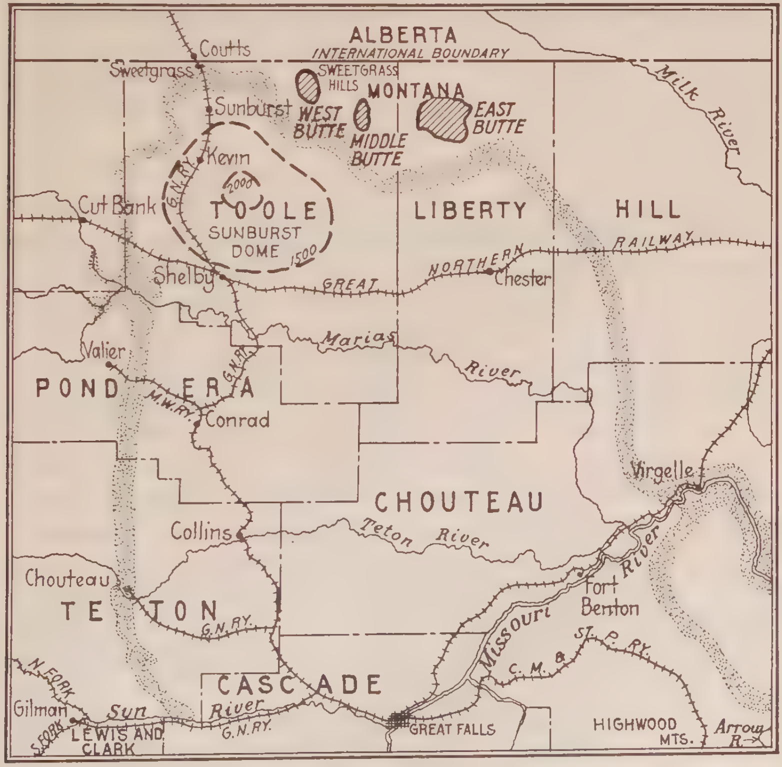

Fig. 1.—Base map, after Bulletin 641-C, U. S. Geol. Surv. (Stebinger). Stippled belt shows outcrop of Eagle sandstone.

largest single oil pool, in areal extent, opened on this continent. Attention was called to this field early in 1922, but the developments of the summer and fall show its possibilities. Sunburst, the main town at the north end of the field, is growing into a substantial center for oil-field workers. Shelby, the county seat of Toole County, is the center for the field offices of most of the companies operating in the field. Great Falls, the largest town near the field, has a population of 25,000, and is the central point for the oil companies operating in the northern part of Montana.

The Sunburst field, see Fig. 1, is located in the northern part of Toole County. As the producing area is just 11 mi. south of the Canadian boundary this field is the farthest north in the United States. The towns of Sunburst and Kevin, on a branch line of the Great Northern railway, are 3 to 6 mi. from the producing area, and the town of Shelby, on the main line of the Great Northern railway, is 5 mi. southwest of the southernmost well in the field, the Ohio-Berg in sec. 6, 32N,R1W.

HISTORY OF FIELD

The first well in the Sunburst field was drilled in sec. 16, T.35N,R3W, by the Gordon Campbell-Kevin Syndicate No. 1, Mar. 14, 1922. Oil was found at 1770 ft. in the basal Ellis formation of Jurassic age. The well was reported as good for 100 to 1000 bbl. but developments prove it good for 5 to 10 bbl. of 30° gravity oil.

The second well, in sec. 34, T.36N,R2W, was completed June 5, 1922, by the Sunburst Oil and Gas Co. This well produced 100 bbl. of 36° oil, at a depth of 1545 ft., in the basal Kootenai formation of Cretaceous age, and augmented the boom in the field. The active entrance of the Ohio Oil Co. into the field, June, 1922, started a vigorous development campaign. One of the important wells is the California-Morton[1] well in sec. 13, T.34,R2W, completed in the Ellis sand at a depth of 1373 ft., which extended the field 9 mi. south of the Sunburst discovery well. Completion of the Ohio-Berg well in sec. 6, T.32,R1W, 9 mi. south of the Morton well, as a 7,000,000-ft. gasser, has opened possibilities 18 mi. south of the northernmost wells.

The rapid development of the field is shown by the number of companies, and their holdings. The best part of the field is now largely in the hands of a few concerns, which are holding the field for economical operations.

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[1] The California Co. is a subsidiary of the Standard of California.

Companies Operating in the Field

Company	Class	Acres
Black Magic Co.	Independent	320
California Oil Co.	(Standard of California)	15,000
Carter Oil Co.	(Standard of New Jersey)	3,000
Campbell Syndicates	Independent	15,000
Franklin Oil Co.		160
Kalispell-Kevin		4,000
Hager-Stevenson Oil Co.	Independent	12,500
Mid-Northern Oil Co.	Standard	3,000
Ohio-Oil Co.	Standard	20,000
Gypsy Oil Co.	Independent	20,000
Gladys Belle Oil Co.	Independent	640
Homestake Co.	Independent	3,500
Hogan, and Associates	Independent	2,000
Shaffer Oil Co.	Independent	320
Sunburst Oil and Gas Co.	Independent	45,000
Troy-Sweetgrass Oil Co.	Independent	20,000
Three-in-one Syndicate	Independent	1,000
Texas Oil Co.	Standard	2,000
Waite Oil Co.	Independent	160
Western Petroleum Exploration Co.	Independent	3,000
Individuals and other small companies	Independent	40,000
		169, 320

The development of the field has been rapid. Over $2,500,000 has been expended in leases and in development, which to Jan. 20, 1923, was:[2]

Producing wells	29
Commercial gas wells	6
Acknowledged dry holes	6
Holes that may be shot into production	4
Idle wells	2
New locations	33
Drilling wells	30
Total	110

STATUS OF LANDS

There are three classes of lands in the area: Patented lands without mineral reservations; unappropriated government lands without minerals reserved to government; state lands. Titles to the patented lands are more or less complicated by mortgages and back taxes; however, the purchase of royalty interests has enabled many of the patentees to clear their lands of obligations. The bulk of the land is patented. Patented lands with mineral reservations are obtained under the Federal Leasing Act of February, 1920; all this land, about 30,000 acres, has been taken up.

The Montana state lands comprise sections 16 and 36 in each township, and sites selected for state institutions. These lands are subject to lease from the state; all of them have been secured. As a whole, the patented land titles are good except for the heavy encumbrances, which will be cleared by the landowners or by oil companies in protecting their rights.

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[2] Brought up to date when proof was revised.

Untested Areas

Not a single test hole has been completed in townships 35N,R1E, 33N,R2W, 33N,R1W, 33N,R1E, and 32N,R1W. Two holes have been drilled in 34N,R1W. One of these was a 1½ in. diamond-drill test, the Troy-Sweetgrass well, which showed oil but was never put on the pump. These townships are structurally as well located as other producing area and there is no reason why production should be absent from them. In at least three full townships no adequate test hole has been completed to date. These townships are just as well located, structurally and stratigraphically, as other areas on the dome that are now producing oil.

Commercial Wells

The oil wells produce from two horizons, the Kootenai and the Ellis Kootenai wells that are producing oil are the original Sunburst discovery well. in sec. 34, T.36N,R2W, the four Davey wells, and two Swazey wells in the same section (see Fig. 5). The Homestake-Reibe No. 1 and the Davey No. 2, in sec. 3, T.35N,R2W, range from 35 to 70 bbl. at a depth of 1540 to 1560 ft. The California-Newman well, in sec. 23, T.35N,R2W, is pumping at 1250 ft., and produces 30 bbl. per day.

One mile south of the Campbell well, in sec. 16, T.35N,R3W, the Black Magic has recently encountered commercial production in the Ellis horizon, although the well is about 40 ft. higher structurally than the Campbell but about 30 ft. lower than the Lewis gas well in sec. 14, T.35N,R3W, which did not obtain commercial oil. In sec. 18, 2 mi. east of the Lewis well, is the Gladys Belle, which had an initial production of 200 bbl. After 60 days' run, that well is producing 65 bbl. per day; it is 6 ft. in the Ellis oil horizon. The other wells in the Ellis horizon are average. The California-Morton well is near the top of the dome in sec. 13-34-1. This well showed dry sands in the Kootenai and had a production in the Ellis horizon of 25 bbl.; however, a new well west of the California-Morton and one east were both seemingly dry in the Kootenai horizon, as well as in the Ellis horizon.

Some of the wells in the northern end of the field, in sec. 34 and 35, T.36N,R2W, and sec. 2 and 3, T.35N,R2W, commonly known as the Sunhio group, produce from both the Kootenai and the Ellis horizons. The oil in the Kootenai horizon is of better grade than that in the Ellis, but the quantity does not justify separate tank facilities.

SHOOTING OF WELLS

Most wells on this dome should be shot in both the Kootenai and the Ellis horizons; shooting has benefited all the wells in which it has been tried. The first shooting was with 20-qt. charges, which were too light. The Hannon and Zimmerman wells, however, showed increased production with these light charges. Davey Nos. 3, 4, and 6, and the Swazey No. 2, drilled in the Kootenai and the Ellis sands, were shot into good wells with 40-qt. charges. Davey No. 5 had a 40-qt. charge for each sand. The Homestake-Riebe No. 1, which had only a showing, was shot with 100 qt. in the Kootenai sand, and is a 40-bbl. well. The Thornton No. 2, in sec. 17, T.35N,R2W, was shot in the Ellis sands with an 80-qt. charge and now produces 50 bbl. or more. Charges of 80 to 100 qt. in all new wells producing less than 50 bbl. are justified by the results so far obtained by shooting.

EXPECTANCY OF PRODUCTION FROM WELLS

From present operations, it is difficult to say what the wells should produce. Wells in the Kootenai horizon, with a body of sand 15 to 20 ft. thick, should produce at least 4000 bbl. per acre, or 32,000 bbl. per well, with one well to each 8 acres. This estimate is conservative.

It is hard to estimate the production for wells in the Ellis horizon, like the Gladys-Belle, Mid-Northern, and Hogan, which started at 200 to 450 bbl. each. The oil horizon is from 5 to 20 ft. thick. Such wells should produce not less than 20,000 bbl. the first year, and be good for 60,000 bbl. for their first ten years of life. This means 6000 bbl. per acre, where one well is drilled per 8 acres, which will be good practice.

These estimates are largely assumptive, especially as only one well has produced over 5 month. That well, the Sunburst discovery well, produced initially 100 bbl. per day and is now producing 45 bbl. per day from the Kootenai sand. The Gladys-Belle No. 1 has produced for 60 days from the Ellis sand and produces 65 bbl. at present. The Mid-Northern well has produced over 225 bbl. per day for 45 days and shows no diminution in output. After 35 days, the Hogan well is producing 450 bbl., which was its initial flow. These wells have not been affected by offsets and so do not give a true index of conditions.

The results of drilling and the geological evidence lead to the belief that a full township of oil-producing land has been fairly well blocked out in the northern end of the field. A township comprises 23,040 acres; with 8 acres per well; this means room for 2880 wells. If these wells average 30,000 bbl. apiece, there are 86,400,000 bbl. of recoverable oil.

DISCUSSION OF DRY HOLES

Dry holes in the field fall into three classes: Those dry holes naturally expected on the edge of every structure: dry holes caused by the lenticular nature of sands; dry holes caused by difference in porosity as a result of cementation, or lack of it, in the oil horizon.

The dry holes found in the first class limit the field and are: The Sunburst Boyce in sec. 23, T.36N,R2W; the Maley well in sec. 31, T.36N,R1W; the Rainbow well in sec. 7, T.35N,R3W; the California-Wilson well in sec. 10, T.35N,R3W. The Montana-Sweetgrass test well in sec. 1, T.32N,R1W is dry at 1577 ft. This well showed no oil, gas, nor water in the Kootenai horizon and an absence of sand in the Ellis. It is more likely to go in the second class but at present is a boundary well; these wells definitely mark productive bounds in both the Kootenai and in the Ellis horizons.

Dry holes in the second class are the Fulton well in sec. 1, T.34N,R3W, and the Sunburst-Suhr well in sec. 10, T.34N,R2W. These wells showed from 1 to 2 ft. of sand in the Ellis, but not enough oil to pay. The Sunburst producing sand, in the southern part of the area, is entirely absent. The Berg gas horizon, sec. 6, T.32N,R1W. is a new horizon in the Kootenai.

Dry holes in the third class are the Ohio-Reibe No. 1 in sec. 3, T.35N, R2W; the Shaffer well in sec. 33, T.36N,R2W; the Hollingsworth well in sec. 34, T.34N,R2W.; these wells may be greatly benefited by shooting. Two wells—the Homestake-Reibe in sec. 3, T.35N,R2W, and the Thornton No. 2, in sec. 17, T.35N,R2W—were classed as dry holes, until shot with 100- and 80-qt. charges, respectively. Other wells of similar type will be shot into production in the future.

Quality of Crude Oil

An analysis of Sunburst crude taken from the Davey No. 1, the discovery well in the Kootenai sand, is given, as the result of a straight distillation test by a local refinery:

Gravity	36.3° Bé
Gasoline	26.0 per cent., 58.6° Bé, end point 428° F., initial boiling point 102° F.
Engine distillate	6.0 per cent.
Kerosene	15.0 per cent., 42° Bé, flash point 150° F., fire test 170° F.
Gas oil	10.0 per cent.
Fuel residue	43.0 per cent.

This gasoline percentage can be increased 10 per cent. by better refining methods.

The Ellis crude ranges from 29° to 38° Bé. gravity. Local refiners say that it has furnished as high as 37.5 per cent. gasoline and 23 per cent. of kerosene. This oil runs 0.5 to 1.5 per cent. sulfur, giving it a peculiar odor. The oil can be refined and gives results comparable to the Salt Creek crude. A complete analysis is not available at present.

GAS WELLS

Nearly all the wells drilled have gas enough to furnish one or more drilling wells. Four gas wells that have been completed show from 1,500,000 to 7,500,000 cu. ft. of gas per day, all from the Kootenai horizon. These are the Lewis well, the Barr well in sec. 2-35-2W, the Sindon well in sec. 1-35-2W, and the Berg well in sec. 6-33-1W. The Lewis and Sindon wells are good for 1,500,000 ft. of gas, each, the Barr for 2,000,000, and the Berg for 7,500,000 cu. ft. The gas from the Berg well is being piped to Shelby for commercial purposes. If enough good wells of the same type are completed, a gas line will be built to Great Falls, where the city and smelters can consume 15,000,000 cu. ft. per day, making a gas field a profitable venture.

CASINGHEAD GASOLINE

The gas in this field can be utilized for casinghead gasoline. The Ohio-Berg well is reported to give 0.70 gal. per 1000 cu. ft., with a volume of 7,500,000 cu. ft. per day. Other wells in the field show from 1 to 4 gal. per 1000 cu. ft., so that as further operations take place, the natural gasoline business will develop.

GEOLOGY

This area ranges from 3200 to 4400 ft. above mean sea level. A high escarpment occurs on the northwest side of the dome; the highest elevation is 4200 ft. The top of the Sunburst dome is 3600 ft. in elevation. The main part of the structure lies inside the escarpments and must be considered a broad rolling area with relief of from 100 to 200 ft. A few shallow coulees cut the dome but, except where the glacial action has occurred, the topography follows the low dips of the area.

About 20 mi. east of the northern limit of the field, the big laccolithic mass of West Butte rises 3500 ft. above the rolling plains. Farther east is the Middle or Gold Butte, and then East Butte. These three buttes are called the Sweetgrass Hills. On the east side of East Butte, one can secure a good columnar section of the beds from the base of the Eagle to 300 ft. below the top of the Madison limestone.

This area has been the scene of much glacial action. The topographic features, while generally controlled by structure, have been greatly altered by glacial action. There are numerous small glacial moraines throughout the field. A rough estimate of the thickness of the ancient glacial sheet as furnished by the heights of glacial action on the Sweetgrass Hills, is 1000 to 1200 ft., according to C. J. Hares, Sherry Thompson, and Dorsey Hager. Glacial till and boulders cover most of the area except where erosion has cut through and exposed the Colorado shale. It is difficult, for this reason, to find good exposures over a large part of the area, and accurate readings are uncertain, which accounts for the many differences of opinions on the part of the geologists.

General Stratigraphy

The main formations encountered in this field are noted in the geologic column here given. The first two formations are found on the sides of the field and are unimportant, so far as drilling is concerned. However, they occur on the outside of the structure and are useful in delineating general structural features of a regional nature.

GEOLOGIC COLUMN

Two Medicine	500+	Massive sandstones and shales.
Eagle sandstone	200	Massive sandstone that forms the escarpment west, north, and east of field; not found on top of dome.
Colorado shale	1750	Shales at top; sands at base furnish sufficient gas in places for drilling operations; basal sands possible producing horizons.
Kootenai beds	450	Green, red, and white shales; pepper and salt sands; carry oil and gas in commercial quantities.
Ellis	200	Thickness in various places over field varies due to unconformity; black shales; siliceous limestone at base; most consistent oil horizon of field.
Madison limestone	800	Massive limestone carrying showings of oil throughout.
Devonian	375	Limestone and black shale, carries showings of oil in Troy-Sweetgrass test; possible producing formation.
Silurian	50	Anyhydrite beds.

The log on page 1110 gives a good idea of the local stratigraphy, also of the drilling conditions.

The highest beds exposed in this area lie on the north, near Sweetgrass, on the escarpment 4 mi. northwest of Kevin, and in the syncline 12 mi. east of the field. The Two Medicine sandstones and shales (which are equivalent to the Claggett shales, the Judith River and the Bearpaw formations) are exposed there. Below these horizons is the Eagle sandstone, of three members and 200 ft. thick. The Colorado shale is exposed over most of the area, extending from the Sweetgrass escarpment south to Great Falls. In the vicinity of the top of Sunburst dome, the lowest beds exposed are equivalent to the Mowry shale beds of Wyoming. These beds lie 80 to 100 ft. below a sandstone carrying inocerami, an excellent horizon marker; 200 ft. above the inocerami beds are iron-stained concretionary beds that are found in many places over the field. These beds form an excellent marker and are used for contouring where other evidence is lacking. They have numerous yellow and red-stained concretionary members, which are prominent enough to give dip readings. These beds range from 60 to 70 ft. in thickness, and the interval from the top of those beds to the top of the Madison limestone is 1550 to 1600 ft., the Colorado shale is 1700 to 1800 ft. thick in the field.

Below the Colorado shale is the Kootenai formation, a series of non-marine brackish and fresh-water beds. The Kootenai formation carries in the upper part green, red, pink, blue, and white shales, interspersed with sandstones from 5 to 70 ft. thick. These sandstones produce oil and gas in parts of the field, notably the oil in Sunburst-Ohio group of wells in sec. 34, T.36,R2W. The lower part of the Kootenai contains some black shales, which in the Sweetgrass Hills show plant remains.

Irregularity of Kootenai Beds

The Kootenai beds are most lenticular in nature; the sands are not persistent. The Sunburst producing horizon dies out 1 mi. north of the Discovery well, is absent 3 mi. east, and is not found 3 mi. southwest nor 5 mi. south. New sands come in and die out rapidly. One sand found in the California-Abell well was 70 ft. thick, 3 mi. south it was 5 ft. thick, 4 mi. north, 5 ft. thick.

Production will be found in various places in the Kootenai beds and correlations will be made tying such sands together, but the author hesitates to make definite correlations at present. The Berg gas well is said to be the Sunburst sand, but is 5 ft. below it as nearly as can be ascertained. As the gas is wet, oil should be found in this sand. Future drilling will ascertain this.

Ellis Formation

Below the Kootenai is the Ellis formation, of Jurassic age, which consists of dark, nearly black shale, carrying much lime. At the base is a sandy limestone 5 to 20 ft. thick, that is a consistent oil horizon in the field and from which the main production is to be expected. An unconformity exists between the Kootenai beds and the Ellis formation. Also the Ellis formation lies unconformably upon the Madison limestone. The Madison limestone is shown, from the log of the Troy-Sweetgrass well, in sec. 21, T.34N,R1W, to be 800 ft. thick on the top of the dome. Below this formation, the Devonian shales and limestones were found, which were 375 ft. thick and petroliferous. An anhydrite-carrying bed, probably Silurian, marked the change between beds of the Devonian and the Silurian age.

Origin of Oil

There are many theories regarding the origin of oil in Montana and Wyoming. It is the opinion of the author that the oil in the Kootenai beds and in the Ellis in northern Montana is indigenous, or native, to those formations. There is plenty of organic material in the basal Kootenai shales, from which oil might have been derived, even though those beds are of brackish and fresh-water origin. Also, the Ellis beds consist of marine shales, black in color at the outcrop and carrying many fossils. If oil came from the Colorado shales, as has been suggested, it has so far failed to saturate the basal Colorado sands, which do, however, carry a little oil in parts of the field, and furnish gas wells sufficient for drilling purposes. The main oil production is, however, 300 to 600 ft. below the Colorado beds. The Madison limestone is considered, by some geologists, as the source of oil, but there is no need to look beyond the Kootenai and the Ellis beds for material from which oil might be derived. The nature of the building of marine limestone, such as the Madison, seems to preclude them from being the source of oil, although the possibility is granted.

Typical Log in Sunburst Field

Depth	Character of Formation
0–60	Blue shale
60–280	Light blue shale
280–285	Gray shale
285–370	Light blue shale
370–400	Black sandy shale (Showing gas at 400 ft.)
400–470	Light blue shale
470–480	Sandy blue shale
480–520	Light blue shale
520–700	Sandy gray shale
700–740	Gray sand
740–750	Gray shale
750–780	Yellow shale
780–810	Blue shale
810–825	Sandy gray shale
825–965	Blue shale
965–995	Sand carrying some water
995–1020	Dark gray shale
1020–1040	Blue shale
1040–1050	Gray sand
1050–1075	Blue shale
1075–1085	Light gray shale showing gas
1085–1105	Sand carrying gas
1105–1125	Blue shale
1125–1150	Sand carrying gas
1150–1200	Top Kootenai
1200–1215	Gray sandy shale
1215–1240	Red shale
1240–1390	Gray sandy shale
1390–1418	Gray shale
1418–1425	Reddish shale
1425–1450	Red shale
1450–1460	Gray shale
1460–1470	Sand
1470–1485	Gray shale
1485–1500	Red shale
1500–1535	Gray shale
1535–1545	Sand carrying gas and oil
1545–1550	Good flow of oil
1550–1565	Oil sand (Kootenai oil horizon)
1565–1580	Black shale
1580–1590	Broken sand
1590–1650	Black shale, top Ellis formation
1650–1680	Limey shale
1680–1760	Dark limey shale
1760–1765	Lime shell
1765–1775	Dark sandy limestone (Ellis oil horizon)
1775–1780	White limestone (Madison)

Geological Structure

The regional features of this area are very interesting. The Sunburst oil field is situated on top of a gigantic dome, which in turn is superimposed upon the axis of the great Sweetgrass arch, see Fig. 1. This

Sweetgrass arch is a structural feature comparable in size to the Cincinnati arch of Ohio and Kentucky, to the Bend arch of Texas, and to the buried granite core of central Kansas. The arch is a northward plunging fold that starts 50 mi. south of Great Falls and extends northward into Canada for 100 mi. The average northward plunge from Great Falls to the Canadian line is 16 ft. per mi. The western limit of this fold is 60 mi. west of the axis in the basin just east of the Rocky Mountains. The eastern limit of the arch, near the Sunburst oil field, is the syncline 12 mi. east of the axis. This syncline changes strike and swings east rapidly. The Sweetgrass Hills, three large masses of igneous rocks, Fig. 1, rising above the plain have played a minor part in the formation of the arch, as the hills are east of the syncline.

The oil production is found on a great dome superimposed on the main arch. The highest part of this dome is in or near sec. 28, 29, 32, 33, and 34, T.34N,R1W, and sec. 4 and 5, T.33N,R1W. The water level on the north lies 600 ft. down from the dome, as shown by the water levels in the Maley well in sec. 31-36-1W, and in the Boyce well, sec. 23-36-2W. If this same governing contour, the 1650, extends around this field, the possible productive area will include 212 sq. mi. General structural conditions are important but local conditions, such as lensing and porosity, seem to govern the accumulation of oil into commercial pools on the main structure. A discussion of the dry holes in the field gives some index as to what may be expected in the way of changes from place to place.

The dips on this fold are low; 1 to 2 degrees is a high dip. The average dip is 50 ft. per mi. although local changes may run as high as 180 ft. per mi. The Sunburst Dome is the first example of a fold of such low dips producing in the Rocky Mountain area. The low dips were given by many geologists as one of the main reasons for condemning this area when drilling was first begun.

Production Range on Contours

So far, no wells have produced oil below the 1650-ft. contour, which to date must be set as the limiting contour of the field. The highest contour is 2250 ft., so the productive range will, in all probability, be over a contour range of 600 ft., and covering five full townships, or 212 sq. mi. It is not thought possible for all this large area to produce, but a large part of it may be expected to give commercial production.

Structure Maps

Figs. 2, 3, and 4 show the difficulties of work in this area, how readily differences of opinion can result, and the progressive stages of development. Fig. 2 is the original map of Jamieson and Howard, the work being based almost entirely on surface data. The general theory of a big dome was established by Jamieson and Howard, and the Sunburst Oil and Gas Co. drilled on their report. In the spring of 1922, geologists for four large operating concerns repudiated the map and the report of Jamieson and Howard. Only when oil was found in the Sunburst discovery well, did the geologists generally recognize the domal folding to exist.

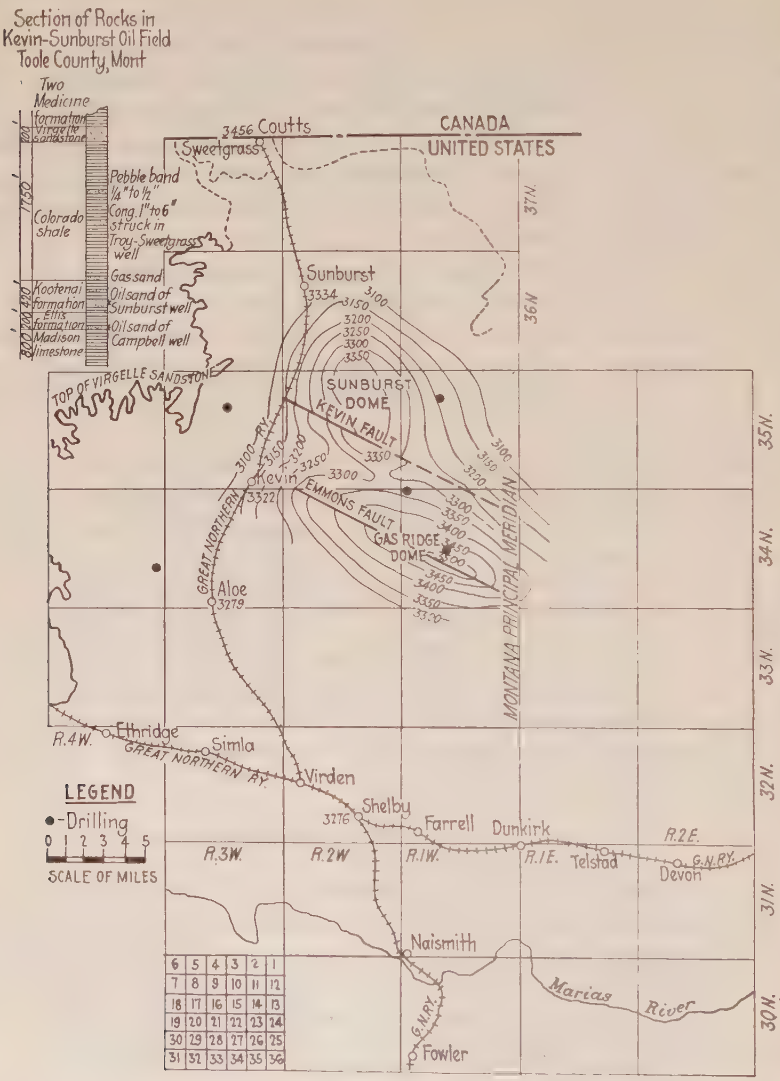

FIG. 2.—MAP OF SUNBURST DOME, AFTER JAMIESON AND HOWARD. CONTOURS BASED ON MOWRY BEDS, OCT. 31, 1921.

It is interesting to note that a low syncline exists between the California Abell well, in sec. 26, and the Zimmerman well, in sec. 23, as shown by the low elevations of the basal Ellis in the Newman well in the southwest quarter of sec. 23, a condition suggested by Jamieson and Howard's original map, which indicates a faulted syncline through this area. In view of the differences of opinion on this point, based on the original map of Jamieson and Howard, this quite marked vindication of their early work is worthy of comment. It is too early to determine the details of faulting.

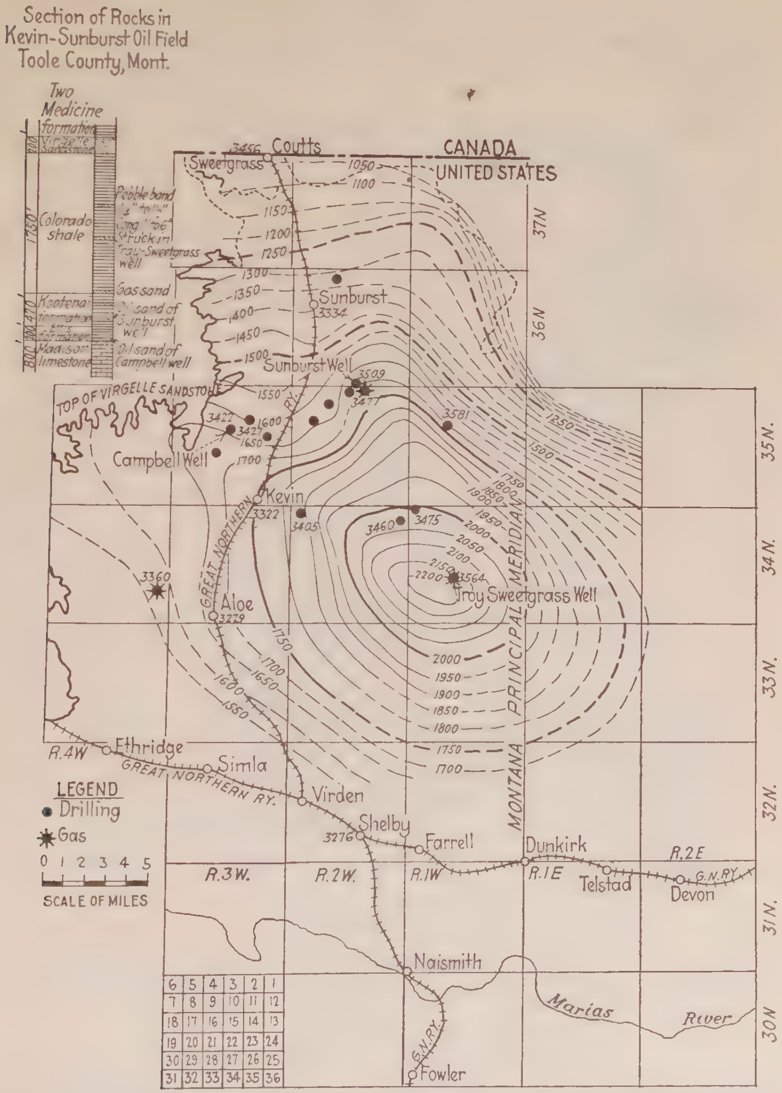

Fig. 3.—Map of Sunburst Dome, by Collier, U. S. G. S. Contours on top of Madison limestone, July, 1922.

Collier, in his preliminary report of July, 1922, fails to mention the work of Jamieson and Howard, and other geologists, which is, no doubt, an oversight. He bases his contour map, Fig. 3, on the top of the

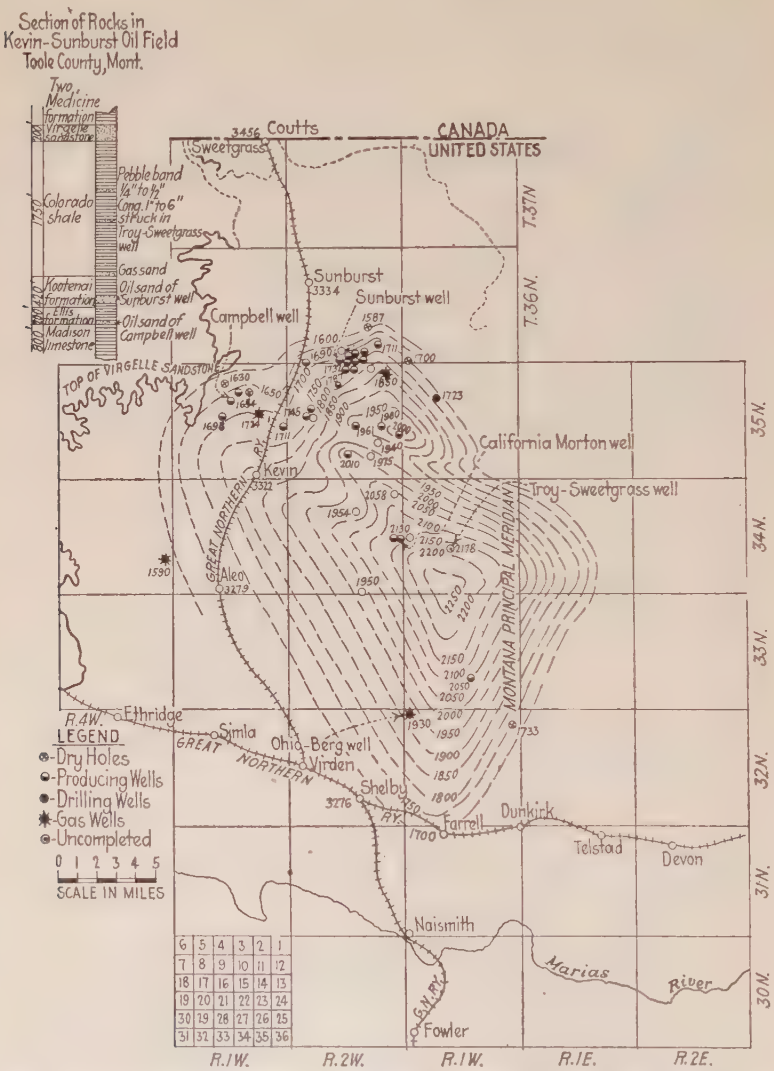

FIG. 4.—MAP OF SUNBURST DOME, BY DORSEY HAGER AND G. G. HOWARD. CONTOURS ON TOP OF MADISON LIMESTONE OBTAINED FROM WELLS AND SURFACE DATA, NOV. 25, 1922.

Madison limestone. On the whole this map summarizes conditions fairly well. It embodies information gained from a number of sources and after more careful, detailed work had been undertaken by a number of companies. However, there are a number of differences, especially in the extreme southern and eastern ends of the field, as shown definitely by wells drilled since Collier's work was completed. His work was, however, an excellent guide to conditions in the southern part of the field.

Fig. 4 gives the latest interpretations, as obtained from new well-log data and additional field work based on new correlations. However, this map will, no doubt, be changed in minor details as further drilling results are known.

Fig. 5 shows the development, to date, in the northern end of the field. It also shows nearly a township of land within possible producing limits. Taken with the geological maps, it would indicate over a full township that should have productive wells over all, or most, of the area.

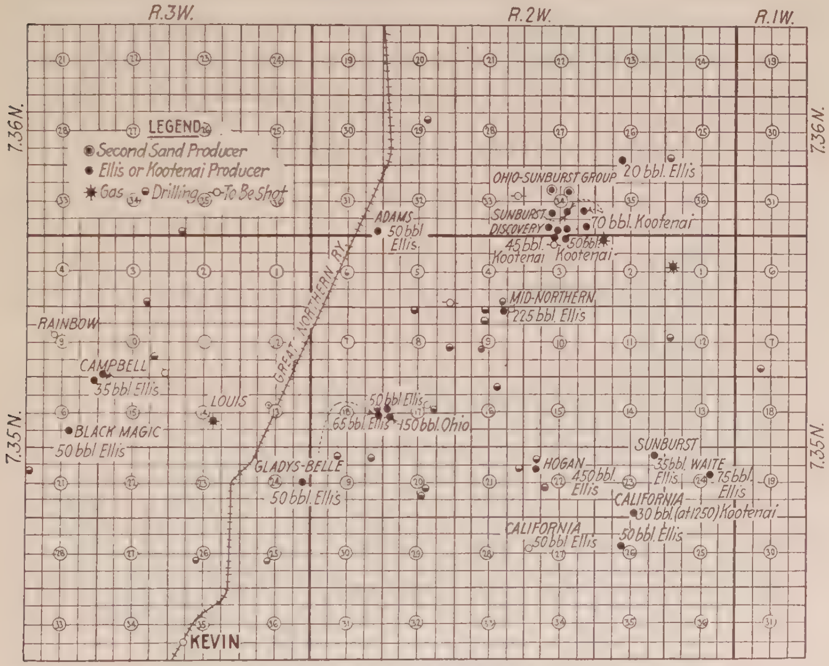

Fig. 5.—Development in northern end of Sunburst Field, Nov. 25, 1922.

Economic Conditions

The general economic conditions are summarized as follows: Transportation facilities, fuel, water, drilling and operating costs, market for product.

Railroad facilities are good. The main line of the Great Northern R. R. enters Shelby, and a branch line extends from Shelby north to Sweetgrass where a branch line of the Canadian Pacific runs to Lethbridge 60 mi.

farther north. The branch line runs along the west side of the Sunburst field. The towns of Sunburst and Kevin are small centers for local operations. Good roads extend from Shelby to the field, so that motor trucks can be used to advantage for ten months out of twelve. The winters, while cold, are generally open enough for motor transportation.

The short distance to railroad makes oil transportation simple. The Illinois Pipe Line Co. (a common carrier) has a line into the field, a 37,500-bbl. storage tank at Sunburst, and loading racks 1 mi. south of Sunburst. This company has planned for a comprehensive network of pipe lines, mains and laterals, which will be built as rapidly as required. At present there are 12 mi. of main 4-in. line, and 10 mi. of laterals. The price, this year, for transporting oil will be 20 c. per barrel, but this charge will be lowered when production increases.

Weather conditions are favorable for development. There are occasional snowfalls in winter, but the Chinook winds seldom let snow stay on the ground for more than two weeks at a time. At times the temperatures run to 40° below zero but not often. Zero is a fair average for three and one-half months, and operating conditions are no more unpleasant than in Pennsylvania, Ohio, or the Mid-Continent, so far as snow and cold are concerned, although the winters are longer. The dry air and clear weather allows drilling operations to continue all winter where water lines are buried.

A water scarcity is being faced now. At present, water for drilling operations is obtained from small reservoirs scattered over the field. These reservoirs are built in various coulees and catch the run-off from melted snow and from rains. An adequate supply of water for temporary needs can, however, be secured from reservoirs. Reservoirs capable of furnishing drilling water for 1000 wells are now being built in the field; these will furnish enough water next year. However, a water line should be laid to one of the streams near the West Butte of the Sweetgrass Hills to afford water for future development on a large scale in the north part of the field. For the southern end of the field, water could be secured from the Marias River.

Fuel for drilling operations is now furnished from gas wells and from oil wells. The Ohio Oil Co. uses gas for its pumping power, and for drilling where the gas is near by. The oil from the wells is an ideal fuel. The present price, November, 1922, at the well is $1 per barrel for fuel purposes. There is plenty of oil and gas to care for a heavy drilling campaign in the field.

Operating Costs and Operations

The cost of drilling and the general operation of wells is higher at present than it will be later, because of the cost of initial wells, and the unsettled conditions always found in new fields. Single wells 1600 to

1700 ft. deep are completed at a cost of $12,000 to $15,000. Dry holes, the same depth, where casing is salvaged and rigs are used again, cost $8000. Some holes have been drilled for as low as $5000. By drilling a number of wells, buying supplies in quantity, and by economical operating methods, the cost of wells can be reduced to $10,000 for a fully equipped pumping well. At present, drilling contracts can be made for $4 per foot, fuel and water being furnished by the operator. Labor contracts can be secured on the basis of $2 per foot.

Some wells in the field have been drilled with portable National No. 28 machines. The most popular and serviceable outfit, because of the ease in handling casing, is a 72-ft. derrick with 5-in. standard rig irons and calf wheel. Such derricks are pulled down when the wells are completed and put on pumping jacks. An ordinary 12- by 12-in. engine is used for drilling.

The average 1600-ft. well requires the following casing:

Size, Inches	Weight, Pounds	Amount, Feet
12½	50	200 (can be pulled)
10	40	1000 (can be pulled)
8¼	28–32	1400
6⅝	20–24	250 liner
or 5$\frac{3}{16}$	17	250

The 12½- and 10-in. casing can be salvaged and a liner of 6⅝ or 5$\frac{3}{16}$ in. is used in completed wells. In many cases the holes use 10-in. casing at the start; 2-in. tubing (4½ lb.) is employed for pumping in most wells.

Wells are completed in from 18 to 30 days, where no unusual drilling difficulties are encountered. The Kootenai formation is liable to cave in; and in some instances casing is carried close to the tools. Underreaming is necessary in such instances to save casing. The Ellis and Colorado formations stand up well, however.

Pumping Wells

Some individual wells in the field are now being pumped by walking beam; but wherever a group of wells is completed, the group is pumped from a central pump, the wells being on pumping jacks. One power pump is used for eight to twelve wells; snow and frost make it difficult to pump a larger number of wells from one plant.

Market for Oil

The market for northern Montana oil is vast. It can be shipped to the Pacific coast and will supply Washington, Oregon, Idaho, Montana, North Dakota, Minnesota, as well as northwest Canada.

The posted market price for oil November, 1922, at Sunburst is 70c per bbl.[3] The field cannot be developed, with profit, at this price.

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[3] Price Jan. 20 raised to $1 average.

The local price should be affected by the building of a refinery at Calgary, now planned by the Imperial, and by local refineries that will undoubtedly be built at Shelby and at Great Falls.

SUMMARY

The outstanding facts of this field are:

1. Discovered Mar. 14, 1922, by the Gordon Campbell well.

2. Main boom caused by discovery of oil on Sunburst dome, June 5, 1922; this well induced the large oil companies to enter the field.

3. Oil is found in the Kootenai beds of basal Cretaceous age and in the basal Ellis beds of Jurassic age.

4. Depths to the Kootenai beds range from 1250 to 1550 ft.; depths to the Ellis range from 1350 to 1800 ft.

5. Production ranges in the Kootenai from 20 to 100 bbl. per day; production in the Ellis ranges from 20 to 450 bbl. per day.

6. Oil is 29° to 34° Bé, in Ellis horizon; 35° to 38° in Kootenai horizon.

7. Possible oil-bearing area, 212 sq. mi.

8. Range on structure contours 2250 to 1650 ft.; 600 ft. height.

9. One township already largely proved oil bearing.

10. Twenty-nine producing wells have a total production of 2200 bbl. per day at average 75 bbl.; 30 wells are being drilled. A heavy drilling campaign is planned for 1923.

11. Pipe-line facilities already furnished by Illinois Pipe Line Co.

12. Market vast; does not compete with other fields.

13. Economic conditions favor operations, because of railroad facilities near field, fair roads, gas and oil fuel available, generally open winters, water available from reservoirs and by lines to neighboring sources, cost of wells fairly low.

14. Large companies operating in field insure intense development program.

CONCLUSION

Developments to date in the Sunburst dome justify the conclusion that it is not one big pool, but will consist of several areas closely connected and highly productive in part. Such a condition calls for very careful study and for development campaigns that are safeguarded by all available technical information. A field of great potentiality has been developed. Over a period of years, oil to the value of several hundred millions of dollars should be extracted from this area.

The occurrence of oil in this great dome has special significance for other low-dip folds in Montana. There are other folds known on the Sweetgrass arch, but none of them of the magnitude of the Sunburst dome. The Cat Creek field in Fergus County is 200 mi. southeast. Other domes between these two fields offer possibilities as general geological conditions are similar over the intervening area.

ACKNOWLEDGMENTS

The material for this paper has been gathered from a number of sources.

1. Records of the Sunburst Oil and Gas Co., the Hager-Stevenson Oil Co., and the Ohio Oil Co.
2. Personal conferences with operators and geologists in field, particularly John MacFayden, C. J. Hares, and William Emery of the Ohio Oil Co., also Charles Emmons of Shelby, one of the pioneer geologists in the field.
3. Geological report and map of Jamieson and Howard, consulting geologists, Great Falls, made in 1921. This was the earliest comprehensive report of the field and its possibilities; the Sunburst Discovery well was drilled on their recommendation.
4. Preliminary report of the U. S. Geological Survey by Collier, in August, 1922.
5. Geological Bulletin No. 641-C by Eugene Stebinger, 1916.
6. Bulletin by Kemp and Billingsley in *Economic Geology*, on Sweet-Grass Hills, in 1918.
7. Personal observations by Grenville G. Howard, Henry S. Lamb, and the author, all of the Sunburst Oil and Gas Co.

DISCUSSION

F. G. CLAPP, New York, N. Y.—Developments in northern Montana have been surprising even to geologists. In the fall of 1921, I was assured by oil geologists of the state that practically no chance existed of oil being obtained in northern Montana. They predicted that the Gordon Campbell well, which was then drilling, would be a failure and that no oil would be found on the Sweetgrass Arch. They based their opinion on the fact that oil sands were absent in certain test wells that had then been made.

We should bear in mind the distinction between the Sunburst-Kevin field and the Sweetgrass Arch field. The Sweetgrass Arch is a great fold that extends north from central Montana into Canada and is perhaps 70 miles broad. The Sunburst Field is a large dome on that great Arch, just as in the Ranger district of central Texas and in the fields of the Cincinnati anticline many minor folds are superimposed on the main fold.

F. J. Fohs, New York, N. Y.—Are there any gas or oil wells in the southeastern part?

F. G. Clapp.—I am not familiar with the distribution of the wells other than those mentioned in the paper as read.

Dorsey Hager.—There are no oil or gas wells in the southeastern part of the Sweetgrass Arch at present. During 1923, a number of test wells will be driven in that region.

Relation of Bonuses and Costs to Present-day Prices of Crude and its Products

By Thomas Cox, Fort Worth, Tex.

(New York Meeting, February, 1923)

The following compilations are made from a series of investigations and are used to present the subject in an unbiased manner, as the writer does not represent any company or financial interest.

The present situation is complex. The producers are hard pressed by reason of the low price of crude, while the independent refiner is little better off, because of the corresponding cut in price of refined products, except on occasional contracts. If the real costs are analyzed, it will be seen that the land speculators and the lessors—royalty owners—make the larger profits and not the struggling legitimate operator or independent refiner.

The present-day (October, 1922) posted prices per barrel for crudes are:

Kansas, Oklahoma, Mexia, Gulf Coast, and Louisiana	$1.25
Healdton	0.75
Northern Texas and Central Texas	1.50
Pennsylvania	3.00
Wyoming, Elk Basin	1.25
Wyoming, Salt Creek	0.70
California Base	0.60

These prices are at the well. In addition, crudes must be gathered and transported to refinery (by pipe line or tank cars), refined, and the refined products marketed.

These crudes are of various gravities, ranging within a narrow margin for each field or district, so that averages are not difficult to deduce, and it is these averages that really determine the average operating conditions.

In order to show the present realizations, the prices and costs now obtaining are listed against the average yields; the figures are also adjusted to show what prices are necessary for such refined products in order to meet actual costs and a living return. This is shown by the following illustrations.

Present price of refined products f.o.b. refinery (Oct. 17, 1922) as published in *Oil and Gas Journal* are:

	North and Central Texas	Dist. 3, Tulsa	Pennsylvania
Gasoline 56–58, per gallon	$0.115	$0.115	$0.175
Naphtha 48–52, per gallon	0.10	0.105	0.1425
Kerosene 45, per gallon	0.045	0.05	0.0825
SR lubricating stocks, per gallon	0.07	0.08	0.16
Gas oil, per gallon	0.025	0.03	0.0625
Fuel oil, per barrel	0.95	1.05	2.52

COSTS AND REALIZATION

Oklahoma-Kansas Crudes, Average 36° Gravity

Costs

Posted price	$1.25
Gathering and pipe line	0.27
Average premium	0.20
Refining	0.35
	$2.07

Yields

	Av. Per Cent.	Gals.	Av. Price	Value	Price Necessary to Balance Overhead	Value
Gasoline	25.00	10.50	$0.115	$1.21	$0.1325	$1.39
Kerosene	15.00	6.30	0.05	0.315	0.0575	0.362
Lub. oil	6.00	2.52	0.08	0.202	0.0920	0.232
Gas oil	22.00	9.24	0.03	0.277	0.0346	0.320
Fuel	26.00	10.92	1.05	0.273	1.20	0.316
Loss	6.00	2.52				
	100.00	42.00		$2.277		$2.62
Crude cost as above				$2.070	$0.55	
Operating margin profit				0.207		
Amount necessary for overhead and capital, as per detail, page 1128				0.55		
Deficit				$0.343 per bbl.		

This shows that the revised prices are necessary for the refiner to make realized costs and earn returns on his plant investment.

Similar calculations for Burkburnett crude, with the following costs:

Posted price	$1.50
Average premium	0.30
Gathering and pipe line	0.27
Refining (skimming)	0.25
	$2.32

and the following yield:

	Per Cent.
Gasoline	35
Kerosene	15
Gas oil	10
Fuel	38
Loss	2

and a selling price for the refined product of $2.46, show that when the costs are adjusted so that the refined products will meet the actual costs and a living return on the investment, there is a loss of 41 c. per barrel.

In the same manner, Ranger crude shows a loss of 33 c., when the adjusted costs are used instead of the following:

Posted price	$1.50
Gathering and pipe line	0.34
Average premiums	0.15
Refining	0.35
	$2.34

the yield being:

	Per Cent.
Gasoline	22
Naphtha	8
Kerosene	20
Lub. oil	18
Fuel	27
Loss	5

and the refined product selling for $2.56 a barrel.

The Pennsylvania eastern district crude shows a loss of 6 c. per bbl., when the refined products sell for $4.76, when the adjusted costs are used instead of the following:

Posted price	$3.00
Gathering and pipe line	0.27
Premium	0.50
Refining	0.50
	$4.27

and the yield is:

	Per Cent.
Gasoline	23
Kerosene	15
Lub. oil	25
Fuel oil	34
Loss	3

In Bureau of Mines Report of Investigation, Serial No. 2364, sample No. 377 showed a marginal operating profit of 32 c. when the costs were:

Posted price	$3.00
Gathering charge	0.27
Premium	0.65
Refining	0.50
	$4.42

and the yield was:

	PER CENT.
Gasoline and naphtha	32.5
Kerosene	17.8
Gas oil	9.4
Light lub. oil Medium lub. oil	16.3
Residue	22.0
Loss	2.0

As the sample was over 40° gravity, it would command a high premium.

But when these costs were properly adjusted, selling the refined products for $4.74 would give a loss of 23 cents.

In June, the posted prices for crudes were:

Kansas, Oklahoma, and Louisiana	$2.00
Healdton	1.00
North Texas, Central Texas, and Mexia	2.25
Gulf Coast	1.25
Pennsylvania	3.50
Wyoming	1.40
California Base	1.10

PRICES REFINED PRODUCTS

	TEXAS	OKLAHOMA	PENNSYLVANIA
Gasoline, per gallon	$0.175	$0.18	$0.22
Naphtha, per gallon	0.165	0.1725	0.18
Kerosene, per gallon	0.04	0.0475	0.06
Gas oil, per gallon	0.025	0.03	0.055
Lub. oil, per gallon	0.08	0.09	0.18
Fuel oil, per barrel	0.90	1.05	2.00

Calculations similar to those just made show that in most cases the oil is sold at a loss. Oklahoma-Kansas crude having the following yield of refined products:

	PER CENT.
Gasoline	25
Kerosene	15
Lub. oil	6
Gas oil	22
Fuel	26
Loss	6

and the following costs:

Posted price	$2.00
Gathering and pipe lines	0.27
Average premium	0.20
Refining	0.35
	$2.82

when the refined products sell for $2.95, instead of a profit of 13 c., when the costs are adjusted so that the refined products will meet the actual costs and a living return on the investment, there will be a loss of 42 c. per barrel.

In the same manner, North Central Texas crude will show a loss of 30 c. per bbl., instead of the profit of 25 c. that is shown when the costs are:

Posted price	$2.25
Premium	0.15
Pipe line	0.355
Refining	0.35
	$3.105

the yield in this case being:

	Per Cent.
Gasoline	22
Naphtha	8
Kerosene	20
Lub. oil	18
Fuel oil	27
Loss	5

The Burkburnett crude will show a loss of 36 c. rather than a profit of 19 c., based on the following yield:

	Per Cent.
Gasoline	35
Kerosene	15
Gas oil	10
Fuel oil	38
Loss	2

and the following costs:

Posted price	$2.25
Premium	0.30
Pipe line	0.27
Refining	0.25
	$3.07

Pennsylvania, eastern district, crude, based on the following costs:

Posted price	$3.00
Premium	0.27
Gathering and pipe line	0.50
Refining	0.50
	$4.27

with the following yield:

	Per Cent.
Gasoline	23
Kerosene	15
Lub. oil	25
Fuel oil	34
Loss	3

shows a profit of 80 c. when the refined products are sold for $5.07 a barrel; but when the costs are properly adjusted in the manner shown, the profit is found to be but 25 cents.

If the posted price increases, refined products must also advance. For example, if Mid-Continent crudes were $3 per bbl., costs would be:

Average Kansas and Oklahoma

Costs

Posted price	$3.00
Average premium	0.20
Gathering	0.28
Refining	0.35
	$3.83

Yield

	Gallons	Prices Necessary to Return Invoice Cost	Adjusted Value
Gasoline	10.50	$0.264	$2.82
Kerosene	6.30	0.063	0.444
Lub. stock	2.52	0.133	0.30
Gas oil	9.24	0.044	0.416
Fuel	10.92	1.625	0.416
Loss	2.52		
	42.00		$4.38

This includes the necessary 55 c. allowed for overhead and capital returns, as analyzed on page 1128.

Ranger Crude

Costs

Posted price	$3.00
Average premium	0.15
Pipe line	0.355
Refining	0.35
	$3.855

Yield

	Gallons	June Prices	Prices Necessary	
Gasoline	9.25	17.5	0.231	$2.12
Naphtha	3.36	16.5		0.725
Kerosene	8.40	0.04		0.44
Lub. stock	7.56	0.08		0.80
Fuel oil	11.33	0.90		0.32
Loss	2.10			
	42.00			4.405

This includes the 55 c. allowed for necessary return of investment and overhead, as given on page 1128.

Burkburnett District

Cost

Posted price	$3.00
Average premium	0.30
Pipe line	0.28
Refining	0.25
	$3.83

Yield

	Gallons	June Prices	Prices Necessary	
Gasoline	14.7	17.5	$0.233	$3.436
Kerosene	6.3	0.040	0.055	0.342
Gas oil	4.2	0.030	0.0365	0.153
Fuel	15.9	0.025	0.119	0.45
	41.1			$4.38

This includes the 55 c., as stated on page 1128, for overhead and capital account.

Pennsylvania, Eastern District

Costs

Posted price	$4.00
Average premium	0.50
Gathering and pipe line	0.29
Refining	0.50
	$5.29

Yield

	Gallons	June Prices	Prices Necessary	
Gasoline	9.7	$0.220	$0.252	$2.45
Kerosene	6.3	0.060	0.068	0.425
Lub. stock, 6.25	10.5	0.18	0.208	2.18
Fuel and gas oil	14.3	0.0475	0.055	0.785
Loss	1.2			
	42.0			$5.840

This includes the necessary allowance for overhead and capital, as listed on page 1128.

Summaries

The summaries show that, with slight exception, the market prices of refined products do not balance the crude cost and the costs necessary to operate and make returns on the investment.

The adjusted prices shown are made on a percentage increase to make up the deficiencies.

There is usually a differential in the refinery price of 5 c. to the tank-wagon price and 3 c. from tank-wagon price to selling-station price; therefore, gasoline, on Oct. 15 market, should be selling at 19.5 c., in No. 3 zone, in order for the refiner to realize a living return. Kerosene and other products must also advance in a similar ratio if he is to attain this end. At this date, the tank-wagon price is 19 c. and the selling-station price is 22 c., which is also 2½ c. against the refiner.

OKLAHOMA

Posted Price	Realization	Crude and Refining	Necessary Allowance	Total	Per Barrel		Marketing	Necessary Price	Date
					Profit	Loss			
$1.25	$2.277	$2.07	$0.55	$2.62		$0.343	$0.115	$0.1325	Oct. 17
2.00	2.95	2.82	0.55	3.37		0.420	0.180	0.203	June
3.00		3.83	0.55	4.38	00	.000		0.264	Est.
				BURKBURNETT					
1.50	2.46	2.32	0.55	2.87		0.41	0.115	0.135	Oct. 17
2.25	3.26	3.07	0.55	3.62		0.36	0.175	0.20	June
3.00		3.82	0.55	4.38	00	.000		0.233	Est.
				CENTRAL TEXAS					
1.50	2.56	2.34	0.55	2.89		0.33	0.115	0.13	Oct. 17
2.25	3.555	3.105	0.55	3.655		0.30	0.175	0.19	June
3.00		3.855	0.55	4.405	00	.000		0.231	Est.
				PENNSYLVANIA					
3.00	4.76	4.27	0.55	4.82		0.06	0.175	0.176	Oct. 17
3.00	5.07	4.27	0.55	4.82	25		0.22	0.21	June
4.00		5.29	0.55	5.84	00	.000		0.252	Est.

EXPLANATION OF REFINING COST ALLOWANCES

In the report of the Commission of Corporation in the Petroleum Industry, prices and profit, the following controlling figures were deduced:

On page 46, the refinery investment is placed at $1.05 per bbl. of crude capacity of plant, of which 70 c. is for physical equipment and 35 c. for necessary investment in current assets. The marketing and sales investments on pages 47 and 599 are given as $1.24 per bbl. capacity. Skimming-plant operations cost from 27 to 42 c. per bbl., the average is 35 c., which is here used as being sufficiently high to permit operating all as skimming plants.

The additional allowances are:

FOR INVESTMENT AND DEPRECIATION	CENTS PER BARREL
15 per cent. on 70 cents of plant, or	10.5
8 per cent. on 35 cents current assets	2.8
Business risks	8.0
Crude storage carried, average 90 days at 8 per cent	5.0
General organization overhead, 10 per cent. of $1.05	10.5
Dividend 12½ per cent. of $1.05	13.2
Marketing expense, 4 per cent. of $1.24	5.0
	55.0

These figures cover all allowable costs and returns to the refiner and tend to show that the independent refiner is not making his costs and investment returns at present prices of refined products. For this, the general demands of the producer to the independent refiner for excessive premiums is somewhat responsible as the producer sells to the large pipe-line companies at posted market price, whereas the independent refiner has to pay high premiums and cash for what oils he can obtain.

Fig. 1 shows the price of crude from $1 to $4, and the necessary price of gasoline at the refinery to correspond to the cost of crude, considering the variations in yield; *e.g.*, crude at $2, Oklahoma-Kansas, 19.5 c.; Central Texas, 19 c.; North Texas, 17 cents.

The data given in Table 1 have been taken from the report of the Bureau of Mines, Serial No. 2364.

Table 1.—*Distillation Tests of Various Oklahoma and Kansas Crudes*

Oklahoma Crudes, 36° Gravity

Sample No.	Field	Gravity	Gasoline, Per Cent.	Kerosene, Per Cent.	Lub. Stock, Per Cent.	Gas Oil, Per Cent.	Fuel Oil, Per Cent.	Loss, Per Cent.
617	Osage.............	35.5	28.9	18.0	23.9	5.2	22.0	2.0
624	Pershing..........	35.5	26.4	19.9	23.9	6.4	20.4	3.0
625	Bigheart..........	35.5	28.1	19.1	17.5	11.4	20.9	3.0
580	Canary...........	31.5	20.4	19.2	19.0	13.5	24.9	3.0
771	Bartlesville.......	30.7	19.0	18.5	26.8	7.3	25.4	3.0
772	Ochelata..........	32.0	22.0	17.7	13.5	6.5	37.3	3.0
610	Delaware.........	31.1	19.9	19.1	17.5	11.4	29.1	3.0
611	Delaware.........	33.0	25.3	17.6	18.5	13.2	22.4	3.0
612	Bluff.............	31.9	22.1	19.5	18.9	10.7	25.8	3.0
754	Claremont........	33.7	23.8	19.2	18.1	12.0	23.9	3.0
620	Yale..............	33.2	24.8	17.8	15.5	9.7	29.2	3.0
733	Slick.............	30.0	22.9	13.0	22.1	5.9	33.1	3.0
734	Kellyville.........	28.6	14.6	15.9	24.2	5.9	36.4	3.0
737	Mounds...........	34.1	28.2	15.8	17.2	9.5	26.3	3.0
736	Glenn............	32.4	24.8	17.4	16.7	12.5	25.6	3.0
731	Turley............	31.3	20.3	18.5	26.4	6.4	25.4	3.0
753	Owasso...........	34.5	26.9	18.8	17.0	12.6	21.7	3.0
752	Broken Arrow.....	34.1	26.2	17.8	16.7	10.9	25.4	3.0
751	Wagner...........	32.0	16.9	17.5	19.6	11.1	31.9	3.0
738	Youngstown.......	31.0	22.2	16.0	16.9	13.0	28.9	3.0
739	Okmulgee.........	33.9	17.3	17.3	20.3	14.7	27.4	3.0
750	Muskogee.........	34.3	19.5	17.8	19.5	13.9	26.3	3.0
649	Comanche.........	33.4	26.3	17.1	15.7	11.0	26.9	3.0
	Average.........	34.0	22.9	17.7	19.4	10.3	26.8	2.9

TABLE 1.—*Continued*

Oklahoma Crudes, +39° Gravity

Sample No.	Field	Gravity	Gasoline, Per Cent.	Kerosene, Per Cent.	Lub. Stock, Per Cent.	Gas Oil, Per Cent.	Fuel Oil, Per Cent.	Loss, Per Cent.
614	Newkirk	40.1	35.9	18.9	15.2	11.2	15.8	3
768	Garber	45.2	52.4	15.7	14.6	3.9	10.4	3
591	Billings	40.1	40.4	17.6	14.2	10.7	14.1	3
619	Cushing	39.1	37.5	18.0	14.3	11.5	15.7	3
732	N. Bristow	39.9	38.8	18.1	13.8	10.7	18.6	3
740	Phillipsvilie	43.1	39.6	15.4	14.1	9.0	18.9	3
741	Bald Hill	41.6	37.6	16.4	15.2	10.5	17.3	3
644	Arbuckle	46.3	46.2	17.8	11.6	9.4	12.0	3
	Average	41.9	41.0	17.2	14.1	9.6	15.3	3
	Oklahoma Crudes, 36° to 39° Gravity							
615	Blackwell	38.7	36.3	18.0	14.1	10.7	17.9	3
622	Hominy	36.7	30.9	19.1	16.6	11.6	18.8	3
743	Deaner	39.5	22.1	16.4	16.4	11.3	30.8	3
744	Lyons	37.5	28.3	17.4	17.2	11.5	22.6	3
735	Beggs	38.7	29.9	14.9	19.6	6.4	26.2	3
745	Henrietta	36.3	25.3	18.3	17.0	12.1	24.3	3
742	Boynton	36.5	24.1	19.6	18.3	13.0	22.0	3
	Average	37.7	28.1	17.7	17.1	10.9	23.2	3
	Kansas, −36° Gravity							
581	Peru Sedan	30.0	19.0	16.6	17.4	10.7	33.3	3
582	Peru Sedan	28.7	12.6	18.4	28.9	5.3	31.8	3
583	Peru Sedan	33.2	20.6	17.1	19.2	11.3	28.8	3
584	Peacock	34.1	25.9	20.2	18.0	11.2	24.7	3
585	Elrod	34.1	25.9	18.6	17.6	11.2	26.7	3
586	Augusta	31.9	24.2	20.5	16.9	11.1	14.3	3
587	Eldorado	34.1	27.3	20.5	16.8	12.1	23.3	3
590	Elbing	33.6	29.8	20.7	15.8	13.3	21.7	3
597	Neodesha	35.5	29.9	15.0	15.5	10.5	29.1	3
596	New Albany	31.7	24.7	13.9	15.4	10.7	32.3	3
598	Independence	33.7	25.3	17.0	16.3	13.4	25.0	3
599	Wayside	28.2	17.3	14.6	15.1	11.1	38.9	3
601	Tyro	29.3	14.2	15.4	18.7	11.5	40.2	3
602	Rantoul	29.1	22.4	14.0	14.5	9.2	36.9	3
603	Osawatomie	27.0	17.1	16.3	17.3	9.8	36.5	3
604	Yates Center	27.5	7.8	18.9	21.9	14.6	33.8	3
605	Iola	19.4	0.8	8.4	29.1	4.7	54.0	3
606	Moran & Elsm	30.0	20.2	15.3	23.4	6.9	31.2	3
607	Urbana	29.8	18.8	16.3	25.1	4.7	32.1	3
608	Chanute	29.5	15.9	17.6	20.7	11.5	31.3	3
609	Erie	30.9	21.2	16.2	18.4	11.9	29.3	3
	Average	30.5	20.0	16.7	19.2	9.9	31.2	3

Kansas, +36° Gravity

Sample No.	Field	Gravity	Gasoline, Per Cent.	Kerosene, Per Cent.	Lub. Stock, Per Cent.	Gas Oil, Per Cent.	Fuel Oil, Per Cent.	Loss, Per Cent.
588	Potwin	43.5	45.0	17.1	12.1	9.9	12.9	3
592	Cattlemew	37.1	32.8	15.1	15.3	9.8	24.0	3
593	Sallyard	38.9	33.2	14.7	16.6	9.8	22.7	3
594	Teeter	36.5	30.1	16.2	18.5	11.3	20.9	3
	Average	39.0	35.3	16.9	14.5	10.2	20.1	3

Production of Crude

The crude producer is little, if any, better off; the following data show that the present bonus bidding and high royalty rate of the Department of the Interior are responsible for some of the higher costs of crude. The high bonus bids are unnecessary, uneconomical, and produce no good results.

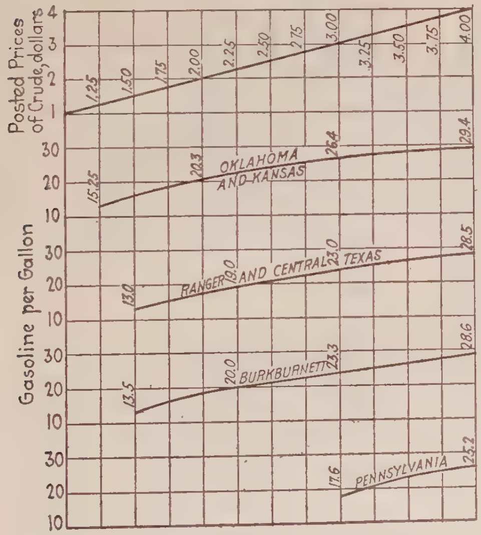


Fig. 1.—Prices of gasoline necessary to balance cost of crude.

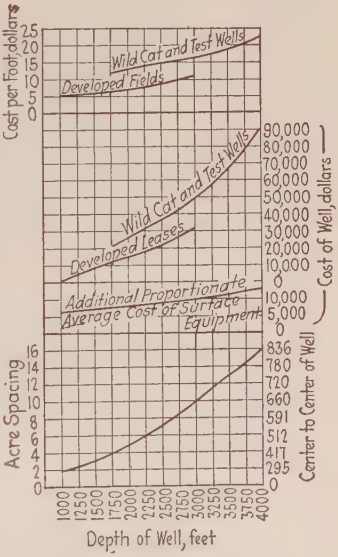


Fig. 2.—Cost of wells and spacing.

As high as $1,600,000 has been paid as a bonus for 160 acres of Burbank, Osage sales, and several bonuses have been over $600,000. These ranges will be used as examples from the prepared charts of average working conditions. When more detailed or exact figures can be obtained, they should be used.

General Items Embracing Operating Costs and Investments

1. Leasehold and land bonuses should be added and considered as part of the investment.

2. Dry holes drilled: average records show that they are 20 per cent. of all drillings; such shall be considered and added to the investment account.

3. Abandonment of well costs should be estimated and added in costs of operating.

4. Proportions of surface equipment costs must be added to cost of well investment, see Fig. 2.

5. An allowance for federal taxes, bonuses, overhead, losses, and general depreciations is suggested at 12 per cent. of price of oil.

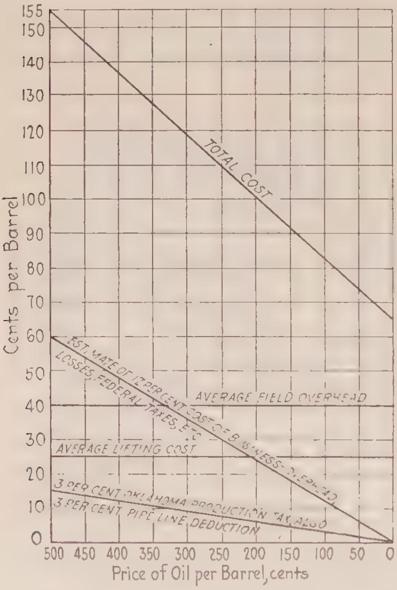


FIG. 3.—OPERATING COST ITEMS, OKLAHOMA CONDITIONS.

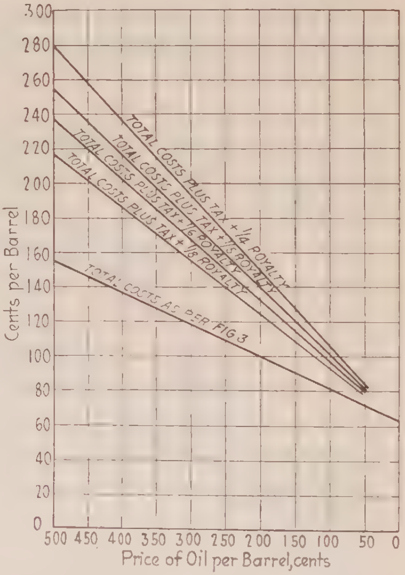


FIG. 4.—OPERATING COST ITEMS, INCLUDING ROYALTIES, OKLAHOMA.

6. In operating charges, the 3 per cent. Oklahoma state tax and 3 per cent. pipe-line gage deductions must be considered.

7. Interest must be earned on the total investment over the life of the property returnable proportional to the decline of production; see Fig. 5 for rate desired and add to total investment.

8. In the above no provision is made for dividends or returns of capital; in figuring the ultimate yield on Figs. 8 and 9, this must be considered.

9. The costs of wells on Fig. 2 has been compiled from a number of actual results and are averages. Individual districts may be above or below these figures and local known figures should be used. The cost of

a well should include: Derrick and all rig irons and crown blocks, engine, wire lines, boiler or rent of same, tools or rent of same, casing, cement (if used), bull ropes, water, fuel, all teaming, all sundries (such as oil, waste packing, lights, etc.) extras (such as circulators, casing shoes, plugs, landing clamps, torpedoes, explosives, etc.), field superintendent and overhead, per cent. general office overhead.

If not a flowing well, the cost should also include: Tubing and rods, head construction, head lines, trap and flow or gage tank, line or tail pump.

If flowing, it should also include the lead lines and the flow or gage tanks.

10. Dry holes must not be overlooked in the estimating of new territory.

11. Abandonment and plugging of wells are estimated at approximately $1 per ft. of depth, and should include all costs.

12. Operating costs are given in Fig. 3. The lifting and other costs shown are averages of a large number of operations; usually in the life of the property costs will run higher. Local conditions, of course, will govern. The general tendency is to underestimate the real costs, if long periods, many basing on the flush period's results.

13. The averages per acre yields are taken from the U. S. Bureau of Mines Bulletin 117 and from other sources of actual record, and can be used as a guide for average acre yield. Individual or later estimates should, of course, be used wherever possible.

Explanation of Charts and Illustrations of Their Use

The chart shown in Fig. 2 indicates the average cost of a well from 1000 to 4000 ft. The curved lines in the top section indicate the cost per foot at various depths. The middle section shows the well cost at various depths; the additional costs to be added for the proportion of the general surface equipment of the property are also given. The lowest section shows the proportionate spacing, consistent with the depth and cost of well. For example, the chart shows that a 2500-ft. well costs $8.88 per ft.; or $22,000, to which $9000 should be added as its proportion of the cost of the surface equipment. The economical spacing of the wells is one to 7 acres, or 550 ft. apart.

The chart in Fig. 3 illustrates costs with crude at $2.50, in Oklahoma;

Production tax, 3 per cent	$0.075
Pipe-line deduction, 3 per cent	0.075
Lifting cost	0.250
Average, 12 per cent	0.300
Overhead	0.400
Total	$1.100

The chart shown in Fig. 4 adds the royalty costs to operating costs and is shown in one-eighth, one-sixth, one-fifth and one-fourth royalty interests. With crude at $2.50, the chart in Fig. 3 shows that the production costs are $1.10; adding one-eighth royalty increases the cost to $1.41.

The chart shown in Fig. 5 is similar to that in Fig. 3, except that no 3 per cent. state tax is included.

The chart shown in Fig. 6 is similar to that shown in Fig. 4, and adds royalty to the estimated costs on the chart in Fig. 5.

The three diagrams shown in Fig. 7 are self explanatory: If it is desired to add interest as a working charge, the amount of the interest charge can be approximated by using the chart in the upper left-hand corner. If interest is to be figured at 8 per cent., read along the 8 per cent. line until it intersects the oblique line, then read along vertical line to scale, which in this case would be 25 per cent., meaning that 25 per cent. of the total investment cost must be added to cover interest on the investment during the life of the property.

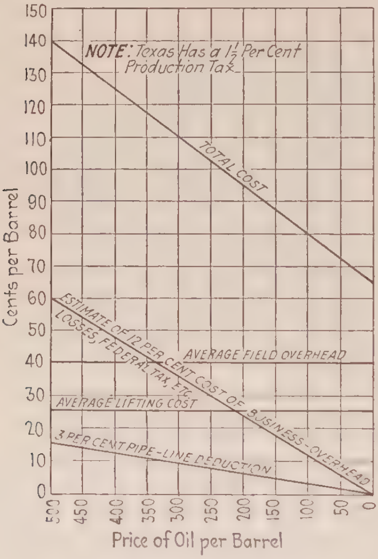


FIG. 5.—OPERATING COST ITEMS, GENERAL CONDITIONS.

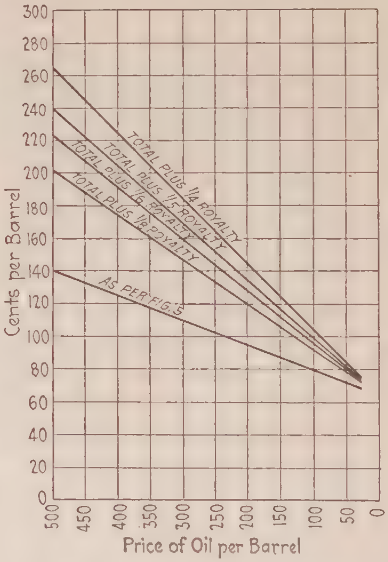


FIG. 6.—OPERATING COST ITEMS, INCLUDING ROYALTIES, GENERAL CONDITIONS.

In the chart in the upper right-hand corner, read along a given price for oil until intersection with oblique line, then read vertically to scale to find, in cents, what the 3 per cent. tax and the 3 per cent. pipe-line gage will amount to; that is, $2 oil equals 6 cents.

The third chart shows what the royalty per barrel, in cents, will be at varying royalties from one-tenth to one-fourth, and at various prices of crude from $1 to $7.50. For example, with crude at $4 and royalty one-eighth, follow the $4 crude line to its intersection with the one-eighth royalty oblique line—then read vertically along the scale, which in this case will be 50 cents. Fig. 8 shows the division, in barrels, of gross production. For instance, if the total estimate or actual gross production is 1,000,000 bbl., and the royalty interest one-fourth, the lessor's interest would be 250,000 bbl., reading to the left-hand margin, and the lessee's interest 750,000 bbl., reading to the right-hand margin.

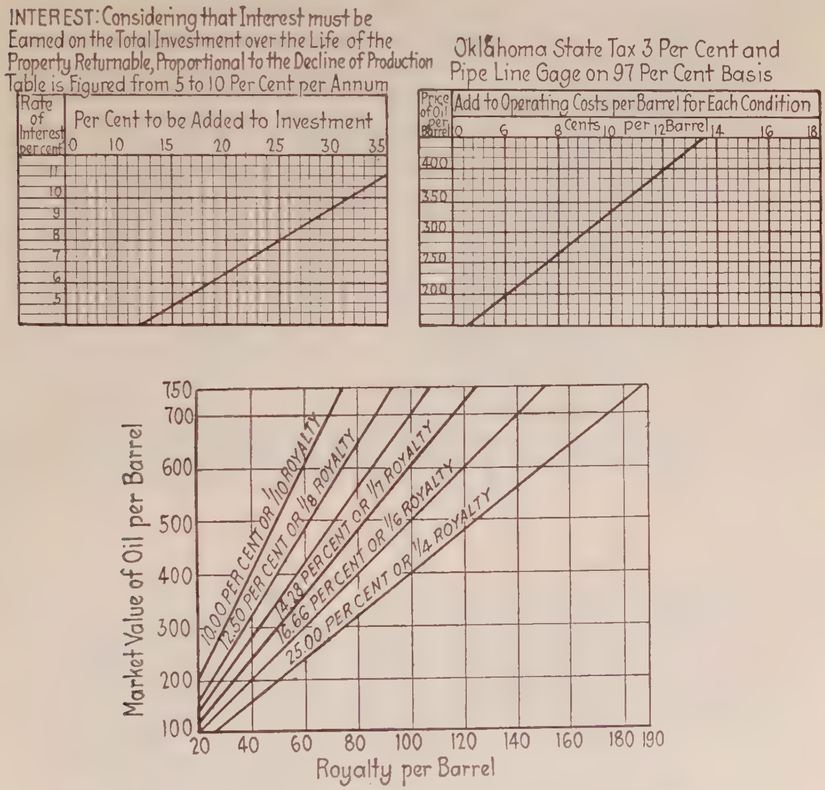
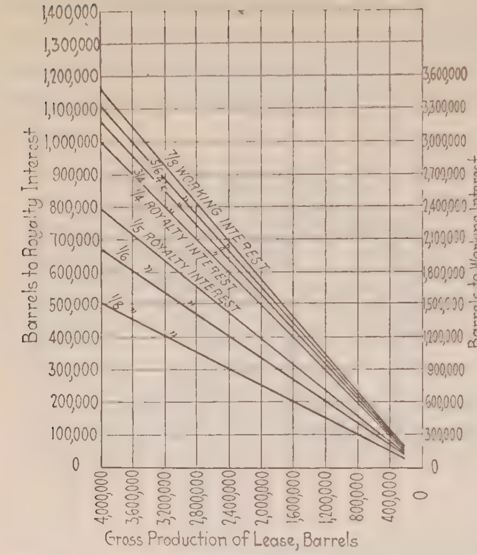


FIG. 7.—ROYALTY, INTEREST, AND PRODUCTION TAX.

The chart in Fig. 9 shows the necessary profits on operations to apply against repayment of investment. With one-fifth royalty and oil at $2.50 per bbl., read vertically from the $2.50 price of oil to the oblique line one-fifth; at this intersection read horizontally back to 89 c., which is the amount that will be required to apply against repayment of investment.

The diagrams in Figs. 10 and 11 make possible a speedy "try out," or the following of the actual results of conditions. In Fig. 10, the bottom line shows varying amounts from $100,000 to $1,800,000; the oblique



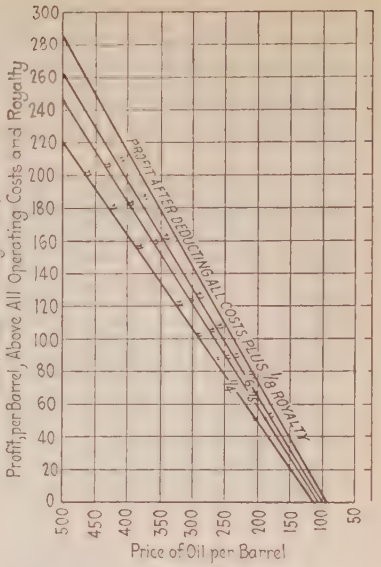


FIG. 8.—DIVISION OF ROYALTY AND WORKING INTERESTS OF GROSS PRODUCTION.

FIG. 9.—PROFITS, PER BARREL, APPLICABLE TO RETURN OF INVESTMENT.

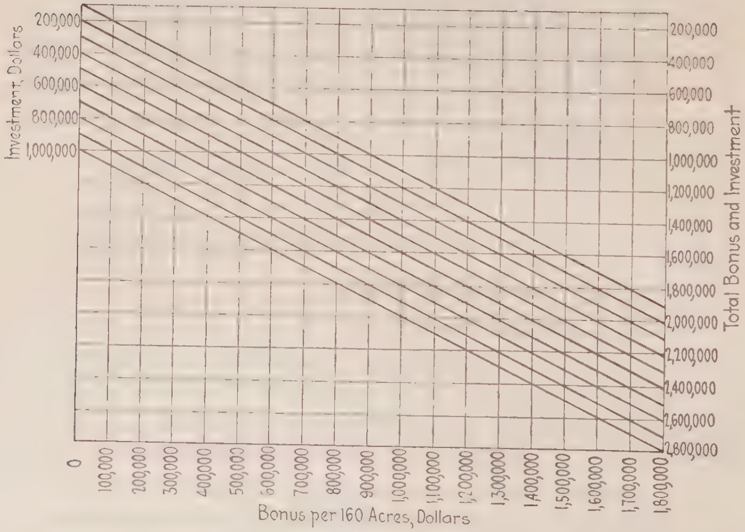


FIG. 10.—GROSS YIELDS NECESSARY FOR NET PROFITS TO REPAY BONUS AND INVESTMENTS.

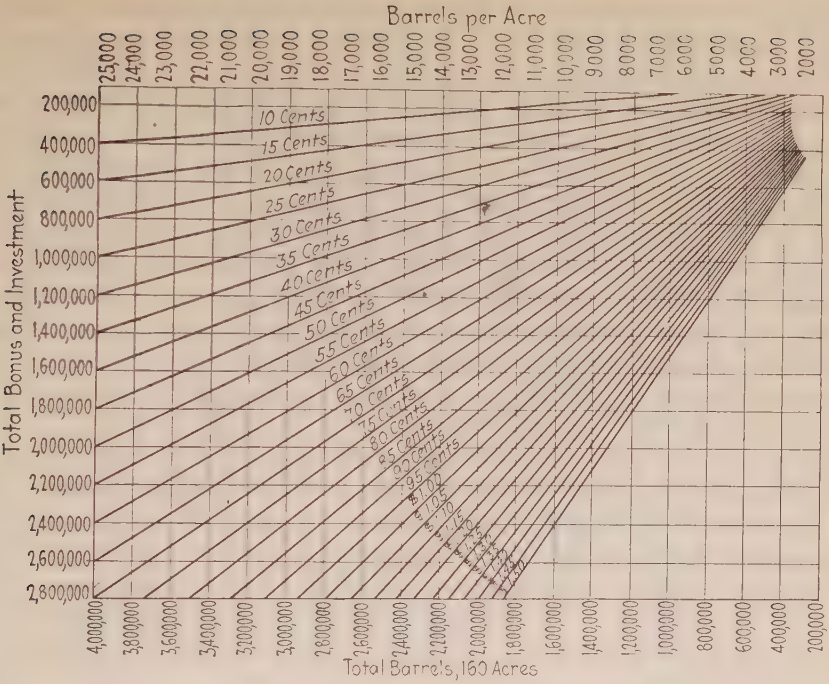


FIG. 11.—YIELD AND NET PROFITS PER BARREL.

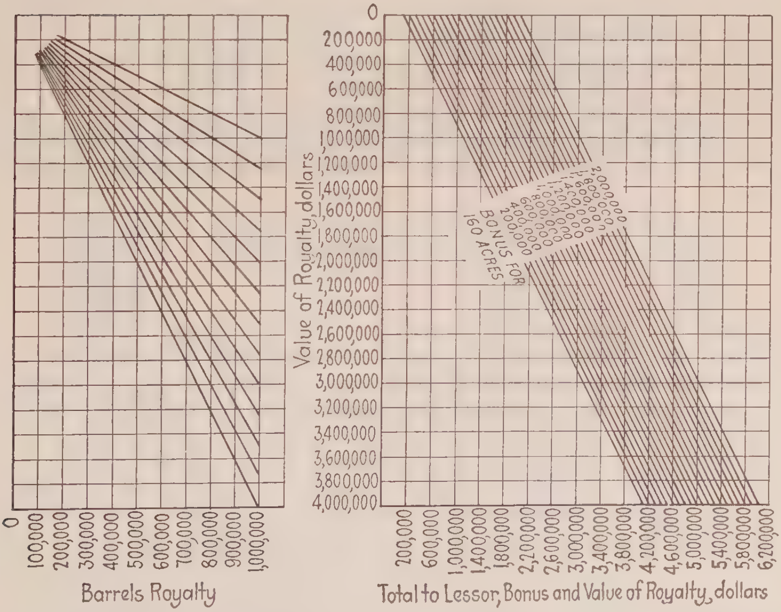


FIG. 12.—SHOWING ROYALTY BARRELS, VALUE AND BONUS PAID, REPRESENTING LESSOR'S TOTAL INTEREST.

line represents investment amounts $100,000 to $1,000,000; and the right-hand column from $100,000 to $2,800,000. In Fig. 11, the left-hand column corresponds to the total column at right of Fig. 10. The radiating lines show from 10 c. to $1.50 per bbl.; these amounts are the net per bbl. applicable to repayment of investment. The barrels at the top and bottom show the amount necessary per acre, or per 160 acres, to balance the "net" profit conditions. For example, if the bonus is $500,000 and the investment necessary to develop, etc., is $700,000, the net profit is 60 c. per bbl. Opposite the intersection of $500,000 with the oblique line of $700,000, in the right-hand column of Fig. 10, is $1,200,000, then, in Fig. 11, vertically below the intersection of the corresponding horizontal line and the 60 c. line, read 2,000,000 bbl. from the 160 acres, or approximately 12,500 bbl. per acre, which is the amount of production required to balance these conditions.

The value of the lessor's interest is shown in Fig. 12. For example, from Fig. 8 it was found that the royalty interest is 300,000 bbl. and that a bonus of $400,000 was paid and oil at average of $3 per barrel. In Fig. 12, under royalty barrels, at junction of 300,000 and $3, read $900,000, then proceed along same horizontal line to the intersection of the $400,000 bonus line, then directly downward and read $1,300,000.

Before beginning to use the charts, some preliminary calculations should be made:

1. The number of wells necessary to develop the property and an estimate of the cost.
2. All the items of expenditure, as outlined on page 1132.
3. Operating charge, including royalties, as found by the charts in Figs. 3 and 4.

For example, it is assumed that new wells and surface equipment on a 160-acre tract at Burbank and all estimated investments will cost:

Fifteen wells at $40,000 each	$600,000
All other surface equipment, buildings, power, etc	90,000
Total	$690,000

With royalty one-fifth on over 100 bbl. and 30° gravity wells; oil at $1.25 per bbl. and commanding a premium of 25 c., average cost of production from Figs. 3 and 4 is found to be $1.21. This leaves 29 c. per bbl. to repay cost of investment before any real earnings obtain.

The first illustration will take the bonus of $1,600,000. From Fig. 2, it was determined that the well costs will be $690,000. The total cost, including royalty, according to Fig. 4 is $1.21, and 29 c. per bbl. is applicable to the investment account. In Fig. 10, following the $1,600,000 bonus line vertically to its intersection with the $690,000 investment line, then horizontally, the total investment and bonus is found to be

$2,290,000. In Fig. 11, find the intersection of the $2,290,000 line and the oblique line of 29 c., thence, vertically below 7,900,000 bbl. is found to be the total production of 160 acres, or 49,500 bbl. per acre, required to repay the investment costs, including bonus, and there will be no profit with oil at $1.50 per bbl. Burbank field will undoubtedly make a new high record for Osage lands in per acre recovery, and it is highly probable that the first production may yield as high as 35,000 bbl. per acre, but the average of the field will not reach these figures.

Following the assumption that this yield of 49,500 bbl. was obtained, the division of interests as between lessor and lessee would be as follows: The lessor has received a bonus of $1,600,000 and a royalty of one-fifth, or 1,580,000 bbl., at $1.50 per bbl., or $2,370,000, making a total of $3,970,000, as shown in Fig. 12, as against no returns for the lessee, except the bare return of investment without interest or dividends.

This bonus of $1,600,000, on such basis of production, represents 20 c. per bbl. of the gross yield. On the basis of no bonus and same conditions of operating, the division of interest would be 59.6 c. per bbl. to lessor and 40.4 c. per bbl. to the lessee, which is a fairer proportion, and shows that such high bonuses cannot be paid and allow any profit at $1.25 crude, carrying 25 c. premium, or $1.50.

It is doubtful if such recovery can be realized, and it will not be as an average of the field. With such bonus bidding, the wards of the Interior Department are getting 100 per cent. of the property.

Using a more normal estimate of an average of 20,000 bbl. per acre recovery, or 3,200,000 bbl., for the 160-acre tract, with oil at $3, operating costs $1.195, royalty 60 c., bonus 50 c. reach a total of $2.295, leaving 70.5 c. for repayment of investment and profit. Fig. 11, with a bonus and investment of $2,900,000, and under 3,200,000 bbl., checks at 71 c., and would be divided:

Operating costs	$1.195	$3,824,000
Royalty	0.60	1,920,000
Bonus	0.50	1,600,000
Investment	0.215	690,000
Profit	0.490	1,566,000
	$3.000	$9,600,000

Of this amount the lessor would receive:

Bonus	$1,600,000
Royalty	1,920,000
	$3,520,000, or 69.2 per cent.

and the lessee,

Profit	$1,566,000, or 30.8 per cent.

Out of this 30.8 per cent. is to be paid interest and dividends on the invested capital, covering a period of possibly 12 years. These returns are not commensurate with the risks and volume of business involved. Typical production decline curves anticipate these conditions.

Illustrating the second bonus of $600,000 per 160 acres, and using the same development expense of $690,000, royalty one-fifth, premium 25 c. oil at $1.25 total operating costs $1.21, leaving 29 c. per bbl. to repay cost of investment before any real earnings appear.

As before, from Figs. 10 and 11, the total bonus and investment is found to be $1,290,000, and vertically below the intersection of this line and 29 c. profit line 4,450,000 bbl., or 27,800 bbl. per acre, is found to be the production required to pay the investment cost, and there will be no profit with oil selling at $1.50 per bbl. The yield must be more productive. In this instance, it is doubtful if the yield will reach the required amount.

Division of these results between the lessor and lessee shows that the lessor's interest would be:

Bonus	$ 600,000
Royalty (890,000 @ $1.50)	1,335,000
Total	$1,935,000

and the lessee would only be repaid his bare operating and investment costs.

Using the same normal estimate of 20,000 bbl. per acre, as in the former illustration, or 3,200,000 bbl. for the 160-acre tract, and with crude at $3 per bbl., the operating costs $1.195, royalty 60 c., leaving $1.205 for repayment of investment and profits. From Fig. 11, at the intersection of 3,200,000 bbl., and $1,290,000, total investment, 40.3 c. is found to be the amount necessary to repay bonus and development costs, leaving 80.2 c. profit. The cost would be divided:

Operating	$1.195	$3,824,000
Royalty	0.600	1,920,000
Bonus	0.188	600,000
Development	0.215	690,000
Profit	0.802	2,566,000
	$3.000	$9,600,000

Of this amount the lessor receives:

Bonus	$ 600,000
Royalty	1,920,000
	$2,520,000, or 49.6 per cent.

and the lessee:

Profit	$2,566,000, or 50.4 per cent.

FOR THE FINAL COMPUTATIONS OF THE LESSOR'S AND LESSEE'S INTERESTS AND ARRIVING AT THE PERCENTAGE OF EACH, THE FOLLOWING BLANK SPACES ARE TO BE FILLED IN, AND THE PERCENTAGE FIGURED IN THE USUAL MANNER

1. Lessor's Interest.................................... $————————
   From Fig. 12.
2. Lessee's Interest.................................... $————————
   From Figs. 8 to 10 or 11.
3. Total 1 plus 2 equals 100 per cent...................... $————————
4. Lessor, 1 divided by 3.................................. ———————— per cent.
5. Lessee, 2 divided by 3.................................. ———————— per cent.
   4 plus 5 equals.. 100 per cent.

FIG. 13.

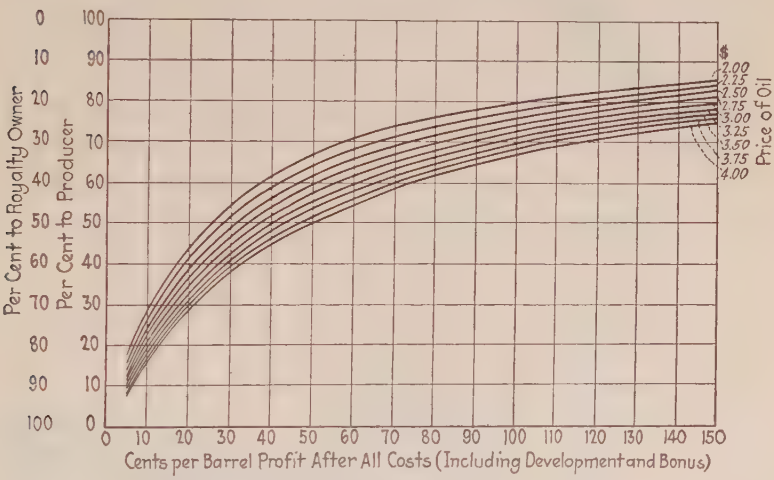


FIG. 14.—DISTRIBUTION OF PROFITS BETWEEN LESSOR AND LESSEE, ONE-EIGHTH ROYALTY.

Figs. 14, 15, 16, and 17 show the division of profit between lessor and lessee with varying prices of oil; for instance, with one-eighth royalty, crude at \$3 a bbl., and a profit of 60 c. In Fig. 14, find the intersection of the \$3 curve and 60 c. profit, then opposite this in the left-hand column it is found that 61.5 per cent. of the profit goes to the producer and 38.5 per cent. to the royalty owner.

Fig. 18 illustrates bonuses that may be bid for acreage. For example, a preliminary estimate of operating conditions tended to show that 25 c. per bbl. could be bid for such bonus, and that the geological data justified an expectation of 6000 bbl. per acre. Reading vertically from 6000 bbl. to the 25 c. line, thence horizontally to the left, shows \$1500 per acre, or \$240,000 for the 160-acre tract, as the bonus that may be bid, which is almost an even division of profits.

From the averages of various Oklahoma fields, as per list herewith, the average of 20,000 bbl. used is a high per acre recovery; at 10,000 bbl. per acre, such bonuses are impossible.

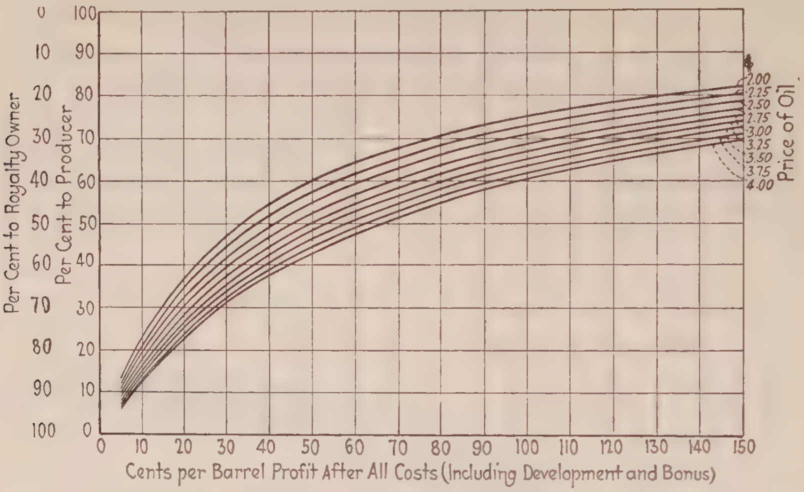


FIG. 15.—DISTRIBUTION OF PROFITS BETWEEN LESSOR AND LESSEE, ONE-SIXTH ROYALTY.

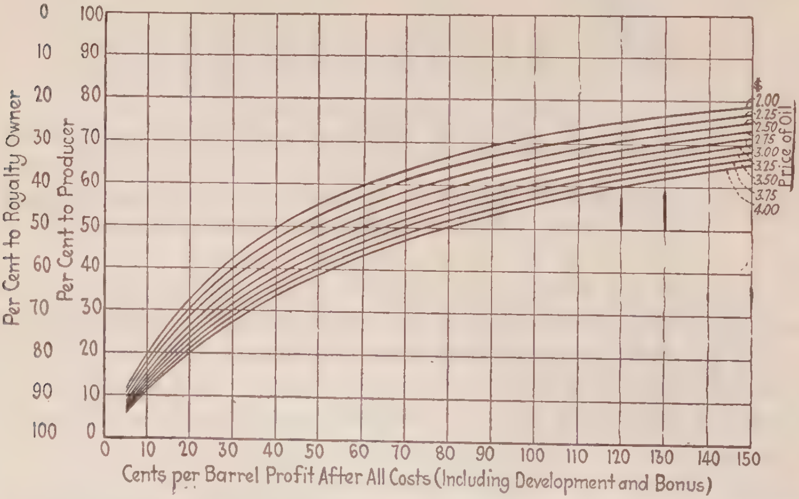


FIG. 16.—DISTRIBUTION OF PROFITS BETWEEN LESSOR AND LESSEE, ONE-FIFTH ROYALTY.

In the foregoing figures it will be noted that no allowances are made for failures, dry holes, or other extraordinary mishaps, of which many occur during the life of a property, and in order to obtain the results as

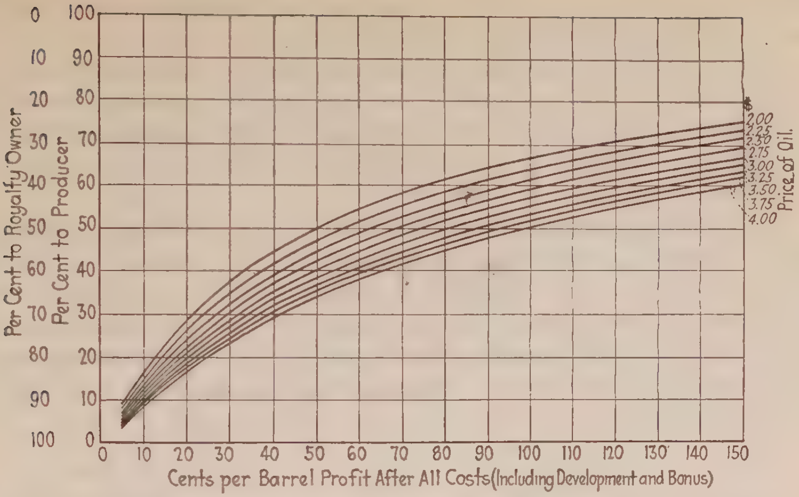


FIG. 17.—DIVISION OF PROFITS BETWEEN LESSOR AND LESSEE, ONE-FOURTH ROYALTY.

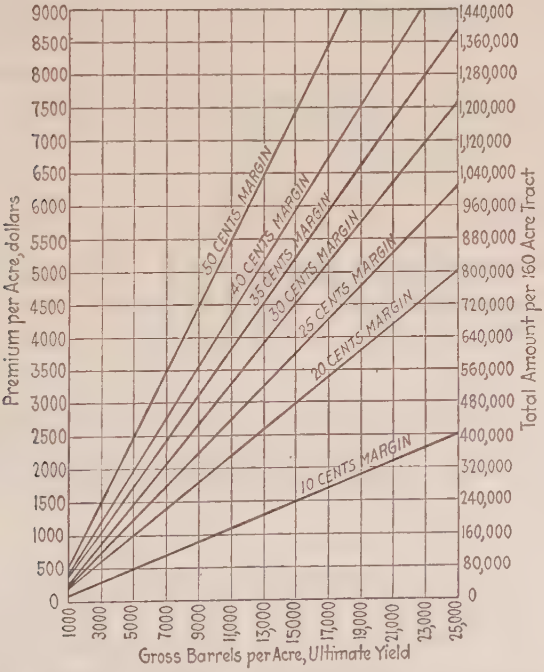


FIG. 18.—PRICES THAT CAN BE BID AT LAND SALES, BASED ON NET PROFITS THAT CAN BE APPORTIONED FOR SUCH BONUSES.

figured, full yields must be realized without extraordinary expenses. Such conditions, however, are seldom met.

The charts shown in Figs. 14–18 are made for average conditions and over the entire life of a property, anticipating the increasing costs with the declining production. The variations of prices and production have a fair range to permit of any practical conditions being run out to obtain a fairly correct estimate.

In the Mexia district, from August, 1921, to end of August, 1922, the records show that of 794 wells drilled, 540 are producers, 5 are gas wells, and 249 are dry holes; the total footage is 2,404,778, of which dry holes are 740,000 ft. The bonuses paid have been approximately $6,000,000; the gross production, 32,305,700 bbl. The posted market price value is $40,030,050. Figuring the average cost per foot for development at $10:

Dry holes will cost	$ 7,400,000
Proportion of bonuses	2,100,000
Total	$ 9,500,000
Value of oil produced	$40,030,050
Less one-eighth royalty	5,003,756
	$35,026,294
Operating costs and all overhead (estimated at 50 c. per bbl.)	$16,152,850
Development and bonuses	20,547,780
Deficit to industry	$ 1,674,336
Dry holes	9,500,000
Total deficit field	$11,174,336
Required to be produced to balance:	
At $1.25 oil	15,000,000 bbl.
At $1.50 oil	11,000,000 bbl.

The deficit of $1,674,336 will only require approximately 2,240,000 bbl. of oil at $1.25 to make up such balance. The dry holes, on a major part, may not be borne by the industry but by the public investments in such wildcat development.

In the Burbank field, 12,480 acres have been sold at Osage sales, with aggregate bonuses of $23,948,000; 499 producing wells, 14 gas and 1 dry hole had been drilled Oct. 1, 1922. These wells cost on an average $40,000, or $20,000,000, and the extra surface equipment, pipe-line tanks, buildings, etc. about $3,000,000 more, making a total of $46,948,000.

Using Fig. 4, with oil at $1.50 per bbl., the total cost, including royalty, is $1.25, leaving 29 c. toward repayment of this development and bonus cost; so that 162,000,000 bbl. would be required for such repayment.

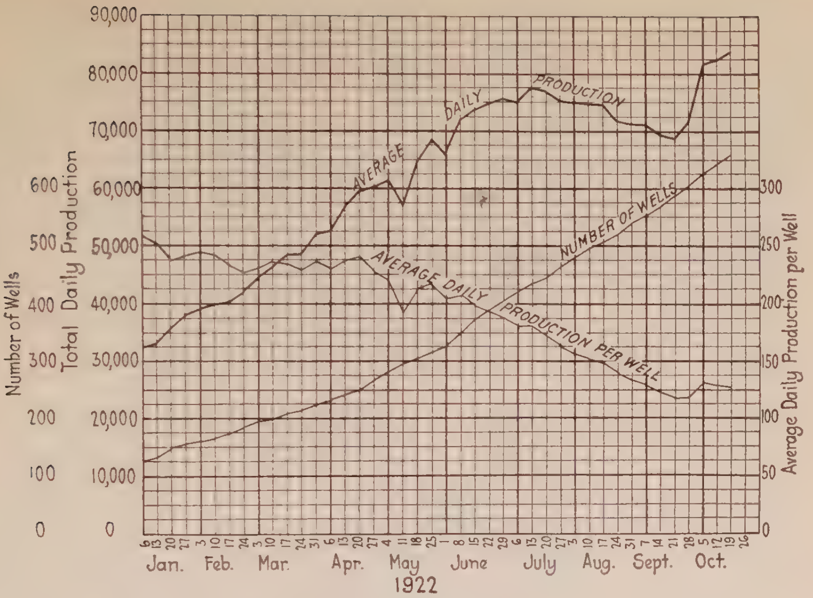


FIG. 19.—PRODUCTION AND WELL CURVES, BURBANK POOL.

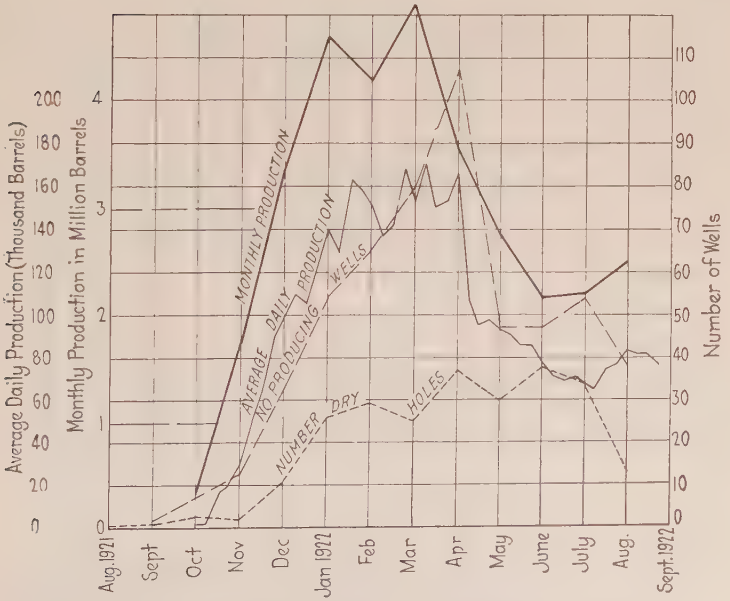


FIG. 20.—PRODUCTION AND WELL CURVES, MEXIA DISTRICT.

Mexia Production Data Including Mexia Proper, Currie Pool, Freestone and Limestone Counties

Month	Oil	Gas	Dry	Total	Total Feet	Cost at $10 per Foot	Average Daily Production, Barrels	Monthly Production, Barrels	Value at Posted Price	
1921										
August			1	1	3,510	$ 35,100				
September	2			2	6,060	60,600				
October			3	3	9,698	96,980	10,100	314,000	0.75	$ 235,500
November	13	1	2	16	49,146	491,460	58,300	1,748,000	0.75	1,311,000
December	33		11	44	133,100	1,331,000	108,800	3,377,300	1.00	3,377,300
1922										
January	55		26	81	245,099	2,450,990	148,400	4,599,100	1.00	4,599,100
February	65		29	94	284,777	2,847,770	149,800	4,192,900	1.25	5,241,125
March	79		25	104	314,917	3,149,170	157,300	4,873,600	1.25	6,092,000
April	107	1	37	145	435,444	4,354,440	119,800	3,547,600	1.50	5,321,400
May	47	3	30	80	243,033	2,430,330	89,000	2,756,000	1.50	4,134,000
June	47		38	85	255,305	2,553,050	72,700	2,178,700	1.50	3,268,050
July*	54		34	88	273,894	2,738,940	71,300	2,209,800	1.50	3,314,700
August	38		13	51	150,785	1,507,850	81,000	2,508,700	1.25	3,135,875
Totals	540	5	249	794	2,404,778	$24,047,780		32,305,700		$40,030,050

* July and August production is influenced by Currie District production being included. Currie District, Navarro County, caused a rise in production at middle of June, 1922.

Basing this on 1250 possible locations, the proportionate area would be 5000 acres developed, or an average of 33,000 bbl. per acre for the whole average.

At $2 oil, the surplus would be 59 c., requiring approximately 80,000,000 bbl., or 16,000 bbl. per acre gross. At $3 oil, the surplus would be $1.205, requiring 39,000,000 bbl. or 7600 bbl. per acre. These figures just repay the development and bonus costs and no profits will accrue. The price of crude is, of course, the important factor in these considerations. These results do not confirm the general public ideas of there being "millions in oil."

*Total Ultimate Production per Acre**

District	Number of Properties	Number of Barrels, Average per Acre	District	Number of Properties	Number of Barrels, Average per Acre
Bartlesville	93	1,800	Augusta, Kans	30	5,591
Muenster	9	4,400	El Dorado	60	5,448
Adair	135	1,780	Osage	10	3,500
Bird and Glenn	91	2,850	Healdton	24	12,600
Okmulgee	46	3,400	Cleveland	15	3,320
Cushing	19	8,800	Blackwell	3	2,900
Muskogee	3	2,000	Garber	16	5,800
Glenn Pool	60	4,050	Yale	5	4,000
Cushing	33	4,440	Sapulpa	16	700
Augusta, Kans	27	4,800			

* From Bureau of Mines *Bulletin* No. 117 and other sources.

This article has been written after the author has studied the economic condition of the industry for a long period, and the charts have been prepared from a vast amount of data. The figures used as cost of refining are the averages taken from a published report of the Federal Trade Commission, which includes a nominal return of cost on investment and equipment.

The operating costs used are low over the whole life of a property. Some individual cases may be less, but the major companies usually have a large administration charge. Investment in producing properties becomes a highly amortized asset, and should be figured as returnable according to the decline of production.

CONCLUSIONS

These illustrations show that under the existing conditions, the following facts appear.

1. The bonuses bid at Osage sales are too high for the present price of oil, as such bonuses and royalty give to the wards of the Interior Department nearly the whole of the returns, and generally the bonuses

demanded by the lessors and their land brokers are too high for profitable investment and operating.

2. If such bonuses and high royalties are necessary to get production, the price of crude must advance fully $1 per bbl. if the producer is to obtain a fair return on his investment.

3. Under normal operations, the present price ($1.25) of crudes just permits operators to earn their operating and investment costs, but no average surplus for dividends and interest.

4. The present prices of gasoline and refined products are not high enough to permit the independent refiner to make any returns on his costs and investment.

Acknowledgment

Acknowledgments are made to Mr. Jesse Stickel for aid in checking the paper, and to Mr. Dan E. Allen for drafting the charts.

DISCUSSION

J. E. Pogue, New York, N. Y.—I would like to make a plea for more data on costs. I have several times had occasion to try to formulate costs, but the data are not available in petroleum literature. We need more exact data of this kind so as to measure the profit element.

Stanley C. Herold, Pittsburgh, Pa.—As the prices of oil have changed since this paper was written, I made some calculations to see how the figures given in Table I would be affected for the various grades of oil. Taking the average oil at the bottom of page 1129 I find that, according to the method the author used, the loss to the refiner is 15.7 c. per bbl. In the case of the average oil, at the top of page 1130, my calculation shows a loss of 17.2 c. per bbl. The third group, beginning with Blackwell, showed a loss of 30.2 c.; the fourth group a loss of 37.1 c.; and the last group a loss of 36 c., when calculated on the prices prevailing on Feb. 6, just preceding the last raise in price. If the refiners are not receiving more for their product since this last raise in crude prices, 10 c. must be added to the figures that I have just given.

Thomas Cox (author's reply to discussion).—Actual cost figures are difficult to obtain. Operators will seldom give out their costs. The writer had access to some refining costs, of plants from 5000 to 10,000 bbl. daily capacity, which were in excess of those used in the calculations. The cost illustrations made rather tend to err on the low side, which would therefore make the losses greater. The subject presents a condition that needs many adjustments in prices of refined products to the independent refiner, as he must get sufficient to yield some profit, to continue in the business. The average crude producer cannot pay high land bonuses and make sufficient profit at present prices of crude to repay such investment, and a reduction in the price of crude is not to be figured on, unless these bonuses are reduced or abolished.

The problem is not so much one of overproduction reducing the price, as the storage is only about a three months supply, but, as to where and how the supply is to be maintained, and at a price that will permit a living profit to both producer and refiner. Independent storage would of course care for much of the fluctuations and in regulating prices.

The continuous operating of refiner and producer can only be made at the expense of not earning their proper depreciation, interest and dividend requirements, and calls for some form of coöperation among operators to reach the desired remedy.

Oil Development on the Isthmus of Tehuantepec

By Stirling Huntley, Pittsburgh, Pa.

(New York Meeting, February, 1923)

With the threatened falling off in production of the lighter oil pools of the Tampico embayment in Mexico, a general search of that country for oil-producing regions has resulted in renewed activity in the region of the Isthmus of Tehuantepec, one of the first districts exploited in Mexico. Since the drilling of the first well in the San Cristobal section by the Pearson interests in 1902, drilling has been continuous, except during 1917–1921, when practically all development was stopped by the revolutionary activity. Early in 1921, a drilling campaign was inaugurated for the purpose of testing out the known salt-dome pools on the flanks of the salt masses, rather than on their tops where previous tests had been located.

On Dec. 7, 1921, the Aguila Co., with which are consolidated the Pearson interests and which has been the only company active in developing this region, unexpectedly drilled into a sand horizon at 3015 ft. with its Concepcion well No. 18. The hole was unable to handle the production, as a 1400-ft. string of 12½-in. casing was hanging in the hole, a salt-water horizon drilled through at about 800 ft., was not cemented off, and 3000 ft. of 4-in. drill stem with rotary bit was in the hole. This well's potential capacity was never determined, but it made 7000 bbl. of 25° Bé. oil through the bit and 4-in. drill stem, in one of its sporadic flows before sanding up, and thus proved itself capable of the largest potential production of any well on the Isthmus. The well showed salt water before caving and sanding up with the drill stem still in the hole, and will probably never be used for production. Wells Nos. 22 and 23, drilled nearby, have proved disappointments after making good showings of oil.

In the spring of 1922, Ixhuatlan wells Nos. 28 and 29, deep tests supposedly on the flank of the known salt cores in that field, went into solid salt; No. 28 at 2250 ft. and No. 29 at 2400 ft. No. 28 was continued in the salt for 1000 ft. before operations were stopped. Later completions in this pool have proved disappointing. Drilling has not been resumed since the shutdown in 1917 except in the Soledad, Concepcion, and Ixhuatlan areas, with the exception of the locations being drilled in Filasola, approximately 500 m. from the bank, on the opposite side of the river from Concepcion. Except for Concepcion, results have been negative for all wells drilled since the shutdown.

As most of the drilling has been shallow, it is only natural that Star machines, cable tools, combination systems, circulating and rotary systems have been tried. The common occurrence in most wells of heaving sands, caving holes, sticky gumbo, in fact all the difficulties of Gulf Coast drilling in an aggravated form, have here resulted in the exclusive use of the rotary system.

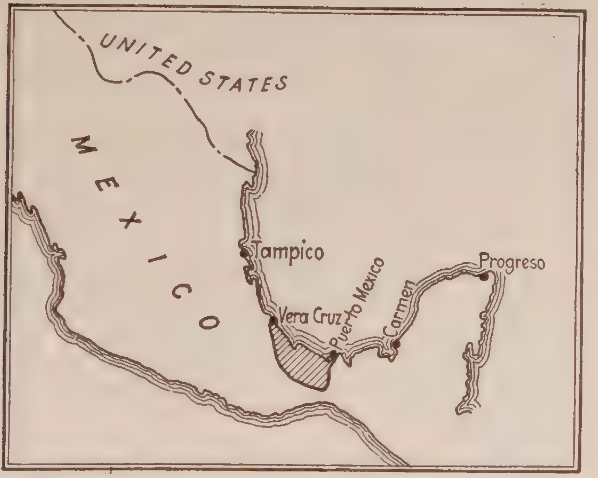


Fig. 1.—Section of Mexico described.

In the early days of development (1908) so optimistic were the opinions of the region that a refinery, with a daily capacity of 18,000 bbl. was erected at Minatitlan on the Coatzacoalcas River, almost at the head of navigation for small ocean-going tankers. During the shutdown in 1920, a fire destroyed part of the plant but it was rebuilt to handle a maximum of 12,000 bbl. daily. Since 1921, it has been operated at a daily average of possibly 6000 bbl., for the most part with crude from elsewhere in Mexico or from the United States. The maximum monthly production from the best fields of the Isthmus has never exceeded 80,000 bbl. No crude has been drawn from Isthmus wells since 1917, except that used for local fuel.

As most of the territory in which operations have been carried on is low, badly drained, near river level, and therefore subject to floods in this unusually wet climate, most of the oil pools are connected with river transportation by small narrow-gage railroads, for it was thought impossible to maintain dirt roads of any length. The narrow-gage railroad to the Tecuanapa pool has been abandoned; the railroads to the Concepcion, Ixhuatlan, and Soledad camps have been rehabilitated and extended and are now in use. Small-gage pipe lines have paralleled these narrow-gage tracks, but, for the most part, they would have to be entirely rebuilt if it should be necessary to transport production through them. In expectation of considerable new production, the Aguila Co. has constructed new tankage facilities both at the Concepcion wells and at the Minatitlan refinery during the past year.

In the past, production has been obtained from the Ixhuatlan, San Cristobal, Capoacan, Soledad, Concepcion and Tecuanapa fields; drilling has been done in the districts of Palmatota, Tuzandepetl, Tonalapa, Potrerillos, Buenavista, Sayula, Juile, Amesquite, Teterete, Santa Ana and Filasola, but without success.

The deepest wells in the region have been drilled in the Ixhuatlan and Palmatota districts; in every case, however, drilling has stopped at less than 3500 ft. Filasola No. 5, recently completed, reached a depth close to 4000 ft. With the possible exception of Jaltipan well, no wells have been drilled to the presumably porous Cretaceous lime, the pay horizon in other parts of Mexico. A total of 220 wells has been drilled, nearly three-fourths of which have been dry; the largest well used for production was ranked at 475 bbl. a day before it went to salt water. All producing wells have been shallower than 1200 ft., with probably 600 ft. as the average depth for all wells drilled.

When it is realized that most of the drilling has been done in semiproved territory, the past drilling, as shown by the accompanying statistics, makes a very poor showing. Wildcatting occurred only in the earlier years of development, so that the high dry-hole average cannot be attributed to that cause. Against the expenditures for about 220 drilled wells, narrow-gage trackage, tankers, tankage, pipe lines, launch and barge service, refinery facilities and fire losses, standard-gage trackage connecting refinery with government railway facilities, camp, office and employees' house construction, loading racks for cars and tankers, and the maintenance of many of these items for long periods of time, is placed the comparatively small income from the six above-mentioned producing districts; complete statistics of the production of these districts are not available.

The Aguila Co. has undoubtedly suffered a loss from its Isthmus operations since the beginning, and only its great success in the Tampico fields have enabled it to continue operations in this section. This company, though it holds immense tracts of land under concession from the Diaz and Madero govenments, and also leases taken up in the name of Pearsons & Son, the Vera Cruz Land and Cattle Co., and the Compañia

Mexicana de Bienes Inmuebles, covering most of the areas favorable to oil development, has always maintained a hostile attitude toward other operators coming in, and drilling and production information has been consistently withheld.

PECULIARITIES OF BEDS AND PAY ENCOUNTERED

Probably 95 per cent. of the drilling to date has been on the tops of salt domes, and the pay has usually been in a dolomite overlying these domes. In some cases this dolomite has been unusually porous and, in the Concepcion pool, wells making 1000, 2000, and 3000 bbl. of salt water have been encountered. The dolomitic cap in the Tecuanapa pool is considerably thicker than at Concepcion; in fact, it is the thickest that has been encountered in any pool.

A sample of salt core from Ixhuatlan No. 28 was clear, pure, and opaque; another sample contained a stringer of hematite cutting upward at an acute angle into the salt mass. Salt cores from the Soledad field showed the top at least of the salt mass in that locality to have an appearance and color peculiarly like quince marmalade, in other words, a pinkish orange.

The red beds described in the Tonalapa log were salt for the most part, though red, tuffaceous beds of an igneous ash composition may be expected to some degree in these beds above the salt.

Salt water is usually encountered in the same pay as the oil, and is produced with it; not as an emulsion, however, as in many instances in the Gulf Coast and Tampico fields, but as water that can be almost entirely separated out by gravity.

In the Gulf Coast salt domes and in much of the Tampico production, the temperature of the salt water is so much higher than that of accompanying oil that the encroachment of salt water is signalized by a sudden rise in the temperature of the oil being drawn. In the Isthmus salt-dome region, both water and oil are at comparatively low temperatures; the absence of emulsified oil would seem to be further proof that there was no great heat action accompanying the formation of the salt domes in this region, nor any great chemical reactions at present. It is possible that if drilling is carried to the Cretaceous lime, this condition will be changed. The Aguila Co., in late years, has been guided in its drilling by contours drawn on the tops of the salt masses, which usually have a form elongated in the direction of the axis or axes of folding within a localized area; see Fig. 2.

TYPES OF OIL

Most of the seepages of the region, and they are numerous, are of an asphalt base (black, tarry, viscous) very like the seepages of the Tampico fields. These are often accompanied by a strong odor of sulfur; in fact,

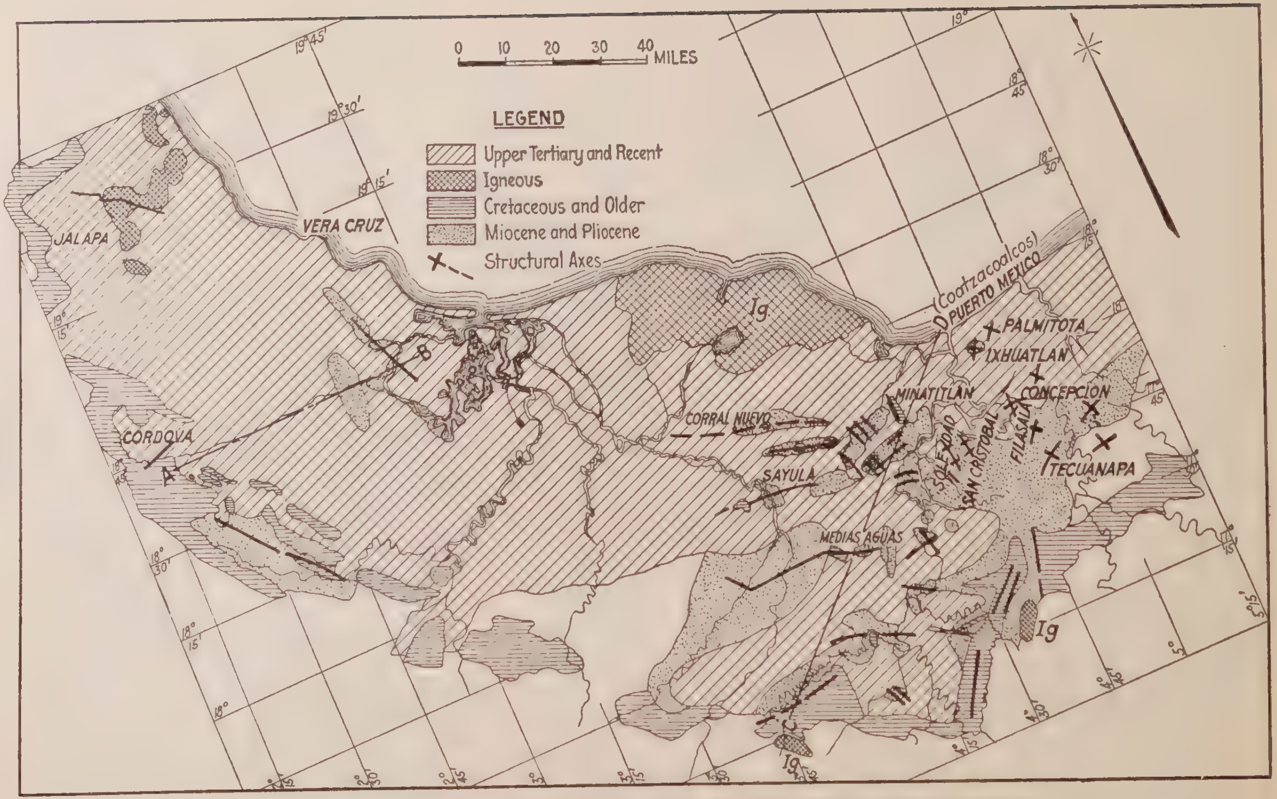


FIG. 2.—SOUTHERN VERA CRUZ AND PART OF ISTHMUS OF TEHUANTEPEC.

sulfur springs are common. The Ixhuatlan pool had several oil pays, usually of different gravities. The oil from this pool contained 10 per cent. or more water, which was partly settled out by gravity. This water contained an appreciable percentage of magnesium sulfate. Producing wells, however, yield paraffine, mixed base and asphaltic oils of varied degrees. The Minatitlan refinery was built to handle paraffine base oils and had to be remodeled to refine the Tampico asphaltic oils.

In the Encantada section, a light-oil seepage at the bottom of a small arroyo furnishes a fluid that is light brown in color, but straw yellow in transmitted light, and has a marked odor of kerosene. No residue accumulates on the side of the seep and the liquid, when rubbed on the hands, evaporates in a short time, leaving a strong odor of kerosene.

NOTES ON GEOLOGY OF AREA

Late Cretaceous and early Tertiary time, until the close of the Oligocene, seems to have been a period of uplift for the Tehuantepec area, as well as the Vera Cruz embayment to the north. Late Cretaceous, Oligocene, and Eocene deposits exist in the Tampico embayment along the coast, to the north, and extensive Eocene and Oligocene (Balthasar district) deposits are known in Tabasco and Chiapas; but formations of none of these ages have been recognized in surface exposures nor in logs of wells drilled on the Isthmus nor in the Vera Cruz embayment adjoining the Isthmus on the north, lying between the Tehuantepec area and the southern end of the Tampico embayment.

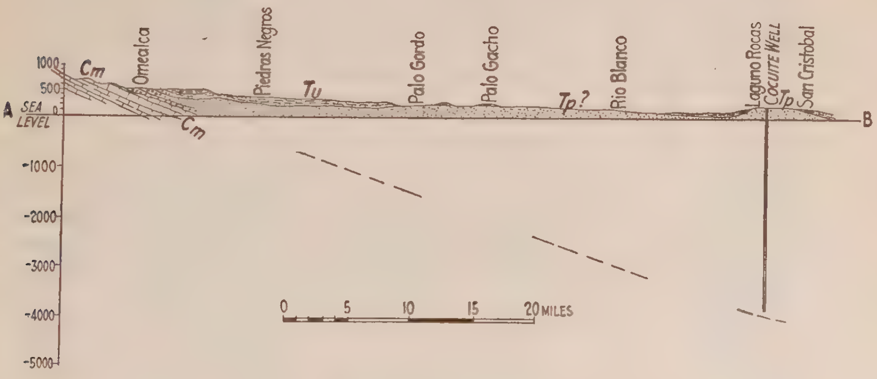


FIG. 3.—SECTION ALONG LINE *AB*, FIG. 2.

The absence of beds of these three periods in these two areas seems to prove that the Isthmus and the Vera Cruz embayment were uplifted as a unit at the close of the middle Cretaceous period, which would coincide with the period of greatest stress in the Tampico embayment, when the

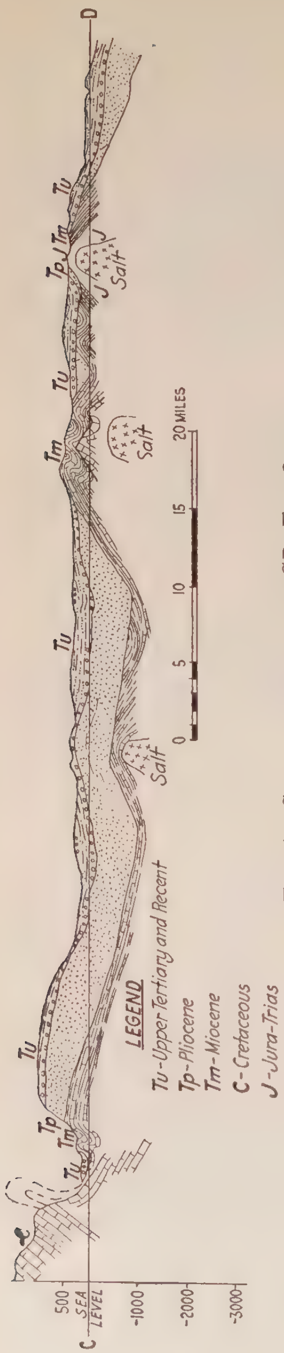


FIG. 4.—SECTION ALONG LINE *CD*, FIG. 2.

mountains were formed which have determined for that region the shore line of each succeeding geologic period.

This major movement in middle Cretaceous period, if not initiating at least reflected an earlier tendency to the continental hinge movement, which is reflected in the marked change in direction of the coast line of this part of Mexico, and the effect of which in the Tehuantepec area is shown by the shattered conditions and intersecting faults of the Cretaceous lime salient, forced out toward the Gulf in the Tehuantepec region, Fig. 2.

These two areas, the Isthmus and the Vera Cruz embayment, a unit in their uplift, were evidently a unit in their descent, and were equally subjected to a deposition of 3000 ft. or more of Miocene shales, if we may judge from the beds at the surface over part of the Isthmus and also from the metamorphosed shales under the lava flows present on the north flank of the Vera Cruz embayment, and from the 3000 ft. of thinly bedded, weathered brown shales lying back of the mountain front in the Santa Maria Tuxtepec region. After the deposition of the Miocene, there seems to have been a period of strong folding, if we may judge from the dips in the Miocene shale overlying the main folds in the Isthmus. It was during this time, or possibly in the previous period of uplift, that the first eruption of the volcano of San Martin occurred. This igneous activity extended over the time succeeding the deposition of the Miocene shales, as evidenced by local areas of metamorphosed blue shale included within the San Martin-San Andres area. It is known that the last period of volcanic activity was in 1796. Evidence of contemporaneous volcanic activity is found in the presence of tuff, ash and debris of various sorts included in the Pliocene and later beds. In view of the fact that the igneous activity of the Tabasco-Chiapas region to the north and east, and more especially that of the Tampico embayment to the north, resulted in the intrusion into the Tertiary sediments of systems of dikes marking the subsurface folding and lines of weakness, it seems remarkable that in this region no such dikes have occurred.

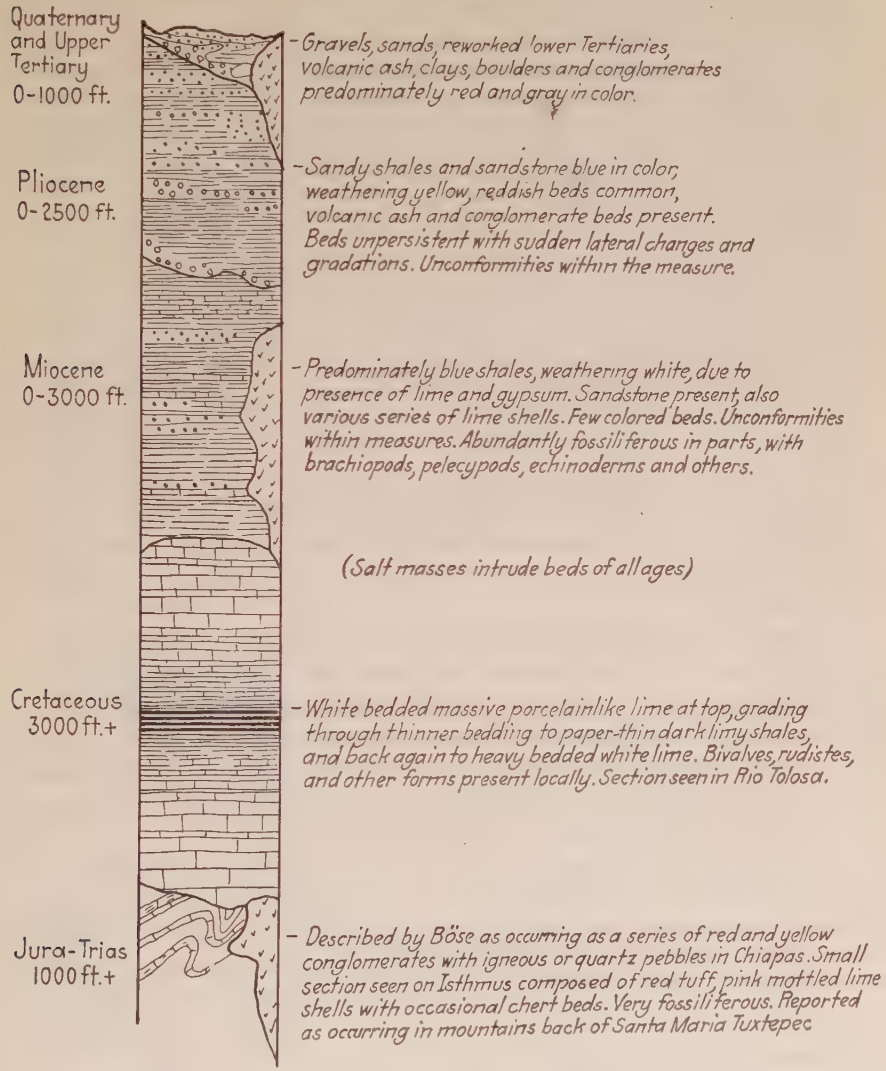

Quaternary and Upper Tertiary 0-1000 ft.
-Gravels, sands, reworked lower Tertiaries, volcanic ash, clays, boulders and conglomerates predominately red and gray in color.
Pliocene 0-2500 ft.
-Sandy shales and sandstone blue in color, weathering yellow, reddish beds common, volcanic ash and conglomerate beds present. Beds unpersistent with sudden lateral changes and gradations. Unconformities within the measure.
Miocene 0-3000 ft.
-Predominately blue shales, weathering white, due to presence of lime and gypsum. Sandstone present, also various series of lime shells. Few colored beds. Unconformities within measures. Abundantly fossiliferous in parts, with brachiopods, pelecypods, echinoderms and others.
(Salt masses intrude beds of all ages)
Cretaceous 3000 ft.+
-White bedded massive porcelainlike lime at top, grading through thinner bedding to paper-thin dark limy shales, and back again to heavy bedded white lime. Bivalves, rudistes, and other forms present locally. Section seen in Rio Tolosa.
Jura-Trias 1000 ft.+
-Described by Böse as occurring as a series of red and yellow conglomerates with igneous or quartz pebbles in Chiapas. Small section seen on Isthmus composed of red tuff, pink mottled lime shells with occasional chert beds. Very fossiliferous. Reported as occurring in mountains back of Santa Maria Tuxtepec

FIG. 5.—COLUMNAR SECTION, MEXICAN ISTHMUS REGION.

There seems to be an erosional unconformity between the Cretaceous and the Miocene beds on the Isthmus, as would be expected, and also an unconformity between the Miocene and the Pliocene. The beds succeeding the Pliocene present a bewildering mixture of littoral, eolian, and lacustrine deposits, which spread over the Isthmus in large part, and which form the cover for the Vera Cruz embayment area.

Deformation in the region of the Isthmus of Tehuantepec is marked by a number of peculiar and unusual features. The dominating factor in determining the direction of the major folding forces that have acted on this area is undoubtedly the tendency toward a continental hinge movement. This movement is reflected not only in a change in direction of the coast line and in the irregularities in the line of Cretaceous-Tertiary contact, which later results in the salient shown in Fig. 2, but also in the interruption of normal folding, shown to be proceeding from the direction of the embayment. This interruption consists of a general shattering of the more resistant beds with a series of intersecting faults or sharp folds, the intersection of which in most cases offered greater freedom for the circulation of deep-seated waters, and from this, in a manner not entirely understood by geologists, proceeded the formation of the numerous salt domes from which all of the Isthmus oil production to date has come.

The salt domes of the Isthmus region are of two general types: quaquaversals, or crossed-axes domes, and domes arising from salt mass intrusions along major anticlinal axes. The latter occur almost exclusively west of the Coatzacoalcos River, there being no long continued San Cristobal-Sayula fold nor a parallel fold to the south through the DeGheest property, both proceeding from the direction of Tabasco, as stated by Redfield.[1]

An interesting study could be made of the arrangement of these short folds and faults, with their relation to lines of deformation present in the surface outcrop of the Cretaceous, and their tendency to line up in systems in some cases seemingly radial from an approximately common center. This suggested alignment differs somewhat from that proposed by Hartley,[2] and on the other hand, De Golyer[3] seems to indicate that no aligning of axes can reasonably be made.

A puzzling feature of reconnaissance work in this region is the presence, in various localities, of massive Cretaceous limestone at or near the surface. These outcrops are usually opened by the natives and used as lime pits. The writer encountered one of these lime pits near the town of Texistepec from which samples of tarry oil were taken. The lime is usually so waterworn and leached that bedding planes are indistinguishable.

Perhaps the most surprising feature encountered in surface reconnaissance is the discovery of steeply folded Triassic beds emerging from the

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[1] Arthur H. Redfield: Isthmian Oil Fields of Mexico. *Eng. & Min. Jnl.* (1921) **111**, 510–514.

[2] Petroleum Geology of the Isthmus of Tehuantepec. *Econ. Geol.* (1917) **12**, 581–588.

[3] Theory of Volcanic Origin of Salt Domes. *Trans.* (1919) **61**, 456.

Tertiaries in the vicinity of Chinameca near the railroad between Jaltipan and Carmen. These beds have nothing in common with any other beds in the Isthmus, and thus are easily recognized. The salt core of this uplift was entered almost at once by the Tonalapa well. No area of Cretaceous lime is found at the surface along this fold, but the patch of Triassic beds brought up through about 6000 ft. of Cretaceous lime and Tertiary sediments seems to have sampled all the formations on the way up, as evidenced by the finding at the Triassic-Upper Tertiary contact of fossils and remnants from beds that normally are far below the surface.

In conclusion, it would seem evident, from facts here recorded, that development in the Isthmus region will continue to result in an economic loss if drilling is confined to the tops of salt domes and to the Tertiary sands on their flanks. It is possible that quantity production can be obtained if the probably porous Cretaceous lime can be encountered within a drillable depth on a favorable uplift. So far this has not occurred except in the doubtful instance of the Jaltipan well, where it is probable that the position of the lime encountered was the result of local uplift by a salt mass.

TABLE 1.—*Average Monthly Production during Time Fields Were Operated*

	BARRELS
Ixhuatlan	8,400
San Cristobal	700
Capoacan	950
Soledad	540
Tecuanapa	3,850
Other fields	760
	15,200

TABLE 2.—*Comparison of Oil of Minatitlan Zone with That of Potrero del Llano*

PRODUCT	POTRERO DEL LLANO PER CENT.	TECUANAPA PER CENT.
Gasoline	5– 6	8
Kerosene	12–18	39
Illuminating oil	.....	9
Gas oil	3	10
Lubricating oil and paraffine	2– 3	34
Combustible oil	50–70	

TABLE 3.—*Production of Tehuantepec District*

	1914 BARRELS	1915 BARRELS	1916 BARRELS	1917 BARRELS
San Cristobal	13,393	14,640	8,183	2,412
Tecuanapa	96,688	98,803	24,143	
Ixhuatlan	100,459	94,990	110,980	18,529
Soledad	9,101	9,154	6,480	
Capoacan	9,120	10,567	10,868	2,252
Total	228,761	228,154	160,654	23,193

TABLE 4.—*Production Data*

Name of Property	No. of Well	Date Started	Date Completed	Depth, in Feet	Initial Daily Production, Barrels	Remarks
Tuzandepetl	1			1665	Dry	Rock salt 1650–1665 ft. abandoned
	2			1210	Dry	Salt water, abandoned
Buena Vista	1			1125	Dry	Abandoned
Jaltipan	1			961	Dry	Lime 912 ft., sulfur water, abandoned.
	2			361	Dry	Abandoned
Capoacan						This pool, with San Cristobal and Soledad, is estimated by Hitchman to have produced 1,300,000 bbl.
Capoacan	1	9–4–05	10–21–05	600	12	Salt at 571 ft.
	2			522	28	
	3			437	21	
	4					
	5			360	8	
	6				Dry	Show, abandoned
	7			170	10	
	8	2–9–07	3–24–07	720	Dry	670 ft. dolomite, abandoned
	9				Dry	Salt water, abandoned
	10			416	2	
	11				Dry	Salt water, abandoned
	12			375	2	
	13				Dry	Salt water, abandoned
	14				Dry	Show, abandoned
	15			395	2	
	16				Dry	Show, abandoned
	17				Dry	Salt water, abandoned
	18			424	Dry	Show, abandoned
	19				Dry	Show, abandoned
	20			277	Dry	Show, abandoned
	21				Dry	Show, abandoned
	22				Dry	Show, abandoned
	23			120	4	
	24			197	4	
	25					
	26				Dry	
	27			207	31	Show, abandoned
	28					Show, abandoned
	29					Show, abandoned
	30					
	31				Dry	Show, abandoned
San Cristobal	1			415	Dry	Showing, abandoned
	2					
	3			452	Dry	Show, abandoned
	4				Dry	Abandoned
	5				Dry	Abandoned
	6					
	7				Dry	Abandoned
	8			425	40	26.8° Bé.
	9			444	2	26.8° Bé.
	10			459	2	26.8° Bé.
	11				Dry	Salt water, abandoned
	12			355	10	26.8° Bé.
	13				Dry	Salt water, abandoned
	14	4–12–06	6–30–06	645	Dry	Salt water, dolomite 290–575 ft., salt 575–645 ft., abandoned
	15				Dry	Abandoned
	16			425	5	26.8° Bé.
	17				Dry	Abandoned
	18					
	19					
	20			952	Dry	Dolomite 660 ft., rock salt 951 ft., salt water, abandoned
	21					
	22			168	10	26.8° Bé.
	23			253	12	26.8° Bé.
	24			276	4	26.8° Bé.
	25				Dry	Abandoned
	26				Dry	Salt water, abandoned
	27				Dry	Show, abandoned
	28				Dry	Abandoned
	29				Dry	Abandoned
	30				Dry	Salt water, abandoned
	31			234	7	26.8° Bé.
	32			216	73	26.8° Bé.
	33			184	Dry	Show, abandoned
	34				Dry	Abandoned
	35				Dry	Salt water, abandoned

TABLE 4.—*Production Data—Continued*

Name of Property	No. of Well	Date Started	Date Completed	Depth, in Feet	Initial Daily Production, Barrels	Remarks
	36				Dry	Salt water, abandoned
	38					
	39			120	2	26.8° Bé.
	40				Dry	Show, abandoned
	41				Dry	Show, abandoned
	42				Dry	Salt water, abandoned
	43			307	20	26.8° Bé.
	44			321	4	26.8° Bé.
	45				Dry	Abandoned
	46			216	4	26.8° Bé.
	47				Dry	Abandoned
	48				Dry	Abandoned
	49			249	Dry	Show, abandoned
	50			180	5	26.8° Bé.
	51				Dry	Abandoned
	52				Dry	Show, abandoned
	53			415	40	
	54				Dry	Show, abandoned
	55				Dry	Show
	56				Dry	Show, abandoned
	57				Dry	Show, abandoned
	58			441	Dry	Show, abandoned
	59				Dry	Show, abandoned
	60				Dry	Show, abandoned
Ixhuatlan........						This pool is estimated to have produced between 3,000,000 and 4,000,000 bbl.
	1	2-5-11		845	12	El. 26.6 ft., came in 500 bbl., 23.3 Bé. Int. pumper
	2	5-8-11		1045	75	El. 25.8 ft., 2,000 bbl., salt water between 300–400 ft., 40 per cent. salt water Int. pumper, pumping 30–60 bbl. in spring '22
	3	7-3-11		1136	42	66 per cent. salt water Int. pumper, 23.3° Bé. el. 27.8 ft.
	4	9-?-11		1261	3	El. 20.9 ft., 20 bbl. per wk., pumper, 8 per cent. salt water 23.3°
	5	11- ?-11		1200	13	50 per cent. salt water Int. pumper 23.3° Bé.
	6	2- ?-12		1627	25	17°–29° Bé., 32° Bé., dolomite 1600 ft., el. 30 ft., capped
	7	8- 3-12		2497	Dry	Dolomite 2465 ft., abandoned
	8	7- 3-12		880	30	Int. pumper, 20 per cent. salt water, 23.3° Bé.
	9	10- ?-12		980	30	23.3° Bé.
	10	3- ?-13		785	30	25 per cent. salt water, Int. pumper, 23.3° Bé.
	11	8- ?-13		511	20	Flowed 300 bbl., 4 per cent. salt water Int. pumper, 23.3° Bé.
	12	11- ?-13	11- ?-14	1220	50	No dolomite, 50 per cent. salt water, Int. pumper, 23.3° Bé.
	13			690	75	Flowed 300 bbl., Int. pumper.
	14	4- ?-14		535	Dry	Never reached oil sands, expected to finish, abandoned
	15	8- ?-14		800	Dry	Suspended, abandoned
	16			730	Dry	Abandoned
	17			513	93	23.3° Bé.
	18			940	Dry	Abandoned
	19			1225	Dry	Abandoned
	20					Location abandoned
	21			204	Dry	Abandoned
	22			1010	Dry	Abandoned
	23					Location abandoned
	24			1995	Dry	Oil sand 800 ft., rock salt 900–1995 ft., abandoned
	25	6- ?-22				Drilling
	26	2- ?-17		907	Dry	Abandoned
	27					Location abandoned
	28	21	22	3350	Dry	Oil show 1850 ft., salt 2250–3350 ft., to plug back
	29	5- ?-22	8- ?-22	2450	Dry	Salt 2400–2450 ft., abandoned
	31	?- ?-22	?- ?-22	1930	Dry	Salt 1890–1930 ft., abandoned
	32	?- ? 22		1300	......	Drilling, November, 1922
Concepcion.......	1	2- 3-09	4- 8-09	250	Closed	Rock salt 246 ft., 14 bbl. oil and 2 bbl. salt water at 128 ft., dolomite 128–246 ft., abandoned

TABLE 4.—*Production Data—Continued*

Name of Property	No. of Well	Date Started	Date Completed	Depth, in Feet	Initial Daily Production, Barrels	Remarks
	2					Abandoned
	3					Abandoned
	4					Abandoned
	5					Abandoned
	6					Abandoned
	7					Abandoned
	8					Abandoned
	9					Abandoned
	10					Abandoned
	11					Abandoned
	12					Abandoned
	13					Abandoned
	14					Abandoned
	15	6- 5-10	6-13-10	223	Dry	Dolomite 200 ft., salt water 2000 bbl., abandoned
	16					Abandoned
	17					Abandoned
	18		12- 7-21	3015	7000	Salt water 800 ft., show 1800 ft., drill stem in hole, well lost 3000 ft., abandoned
	19					Fishing, May, 1922, at 3000 ft.
	20					Drilling, June, 1922
	21	2-17-22				Drilling, June, 1922
	22					Nov., 1922, sidetracking casing
	23					Nov., 1922, sidetracking casing
Soledad..........	1				Dry	Showing, abandoned
	2				Dry	Showing, abandoned
	3				Dry	Abandoned
	4	2-26-10	11-27-10	452	Dry	Oil sand 197 ft., salt 415–452ft. Abandoned
	5				Dry	Abandoned
	6			298	62	
	7				Dry	Abandoned
	8			2545	3	
	9				Dry	Abandoned
	10				Dry	Show, abandoned
	11				Dry	Show, abandoned
	12			1151	3	
	13				Dry	Abandoned
	14	8-19-10	10- 9-10	1366	Dry	Salt water, abandoned
	15				Dry	
	16			344	3	
	17				Dry	Abandoned
	18				Dry	Abandoned
	19			216	3	
	20				Dry	Abandoned
	21				Dry	Abandoned
	22			699	52	
	23			505	7	
	24	?- ?-22	4-15-22	300	Dry	Rock salt, abandoned
	25	?- ?-22	5-15-22	600	Dry	Rock salt, abandoned
	26	?- ?-22	?- ?-22		Dry	Rock salt, abandoned
	27	?- ?-22	?- ?-22	2622	Dry	Salt, 2037–2622 ft., dry, abandoned
Tecuanapa.......						Hitchman's estimate 360,000 bbl. produced to 4–30–16. Government figures give 350,000 bbl. produced to 12–17. Three wells productive of all drilled.
	1	6-13-10	8-10-10	439	475	34° Bé. oil, el. 108.3 ft., dolomite, shallow sulfur water
	2	9- 4-10	12-30-10	910	Dry	Salt, fresh sulfur water, el. 120.6 ft., no dolomite.
	3	1- 5-11	2-11-11			Abandoned
	4	3- ?-11	4- ?-11	2250	Dry	Shows oil and gas, no dolomite, abandoned
	5	8-13-11	9-20-11	500	200	Dolomite 468 ft., 34° Bé., el. 102.6 ft.
	6	?- ?-11	?- ?-11	538	Dry	Dolomite, el. 97.8 ft., salt, 1000 bbl. sulfur water, oil and gas show 175 ft., abandoned
	7	1-11-12	3- 6-12	560	Dry	Salt, strong gas, el. 107.8 ft., dolomite, abandoned
	8	4- 4-12	4-30-12	492	200	34° Bé. from dolomite, el. 147.7ft.
	9	5- 5-12	6-16-12	656	Dry	Salt, 500 bbl., salt water, el. 169.8 ft., dolomite, abandoned
	10	6- 1-12	7- ?-12	522	Dry	Abandoned
	11	6- ?-12		536	Dry	Abandoned
	12			600	Dry	Abdanoned

TABLE 4.—*Production Data—Continued*

Name of Property	No. of Well	Date Started	Date Completed	Depth, in Feet	Initial Daily Production Barrels	Remarks
Palmatota	1		3- ?-14	2840	Dry	Blue shale hole, show in sand 1750 ft., abandoned
Filasola	1			1122	Dry	Dolomite 250 ft., abandoned.
	2			1155	Dry	
	3			364	Dry	Abandoned
	4					
	5			4000	Dry	Salt, show at 2520 ft., to perforate casing
	6			2400	Dry	Dolomite, gas at 2350 ft.
Teterete	1			282		Rock salt 272 ft., asphalt and sulfur, abandoned
Santa Ana	1			325		Abandoned
Potrerillos	1	7- 7-02	8-15-02	1200	Dry	Show, abandoned
	2			860	Dry	Abandoned
	3			675	Dry	Abandoned
	4			1450	Dry	Abandoned
	5	7- 1-05	10-21-06	1312	Dry	Several shows, abandoned
	6			772		
	7			712		
	8			630	Dry	Show at 440 ft., abandoned
Sayula	1	5- 8-05	8-26-05	1374	Dry	Salt water, abandoned
	2			1574	Dry	Shallow salt water, abandoned
	3			2250	Dry	Salt water, oil shows, abandoned
	4		8- 7-07	750	Dry	Oil shows and salt water, abandoned
Juile	1			1815	Dry	Rock salt 1350–1815 ft., Pliocene fossils 658 ft., salt water, abandoned
Tonalapa				2453	Dry	Mostly rock salt, though not reported so, abandoned
Amesquite			3-22-05	1050	Dry	Salt water with oil shows, abandoned

TABLE 5.—*Well Logs*

JALTIPAN NO. 1

FEET	
0– 456	Blue marl
456– 461	Water sand, salt water
461– 497	Blue marl, showing gas
497– 638	Blue marl
638– 912	Blue marl, lime, and sulfur
912– 920	Lime and sand
920– 935	White sand and lime; hole filled with water; pulled casing and under-dammed; shut water off.
935– 960	Lime sand and sulfur
960	Well filled up with sulfur water, casing pulled and well abandoned; left 324 ft. of 10-in. casing in hole.

IXHUATLAN NO.2

El. 25.8 ft.

FEET	
0– 100	Blue marl, sand
100– 200	Blue marl
200– 300	Blue marl, lime shell, blue marl
300– 400	Blue marl, sand 2000 bbl. water with little salt, blue marl, sandy shale, blue marl
400– 500	Blue marl
500– 600	Blue marl, oil sand 4 ft., sandy blue marl, oil sand 2 ft., sandy blue marl

FEET	
600– 700	Sandy blue marl, oil sand 4 ft., blue marl
700– 800	Blue marl, lime shell, blue marl, oil and gas, blue marl
800– 900	Lime shell, oil sand 2 ft., sandy blue marl, sand, sandy blue marl, heaving sand
900–1000	Blue marl, sand, sandy blue marl
1000–1045	Sandy blue marl, oil sand 27 ft., oil and blue marl. 23.3° Bé. 4 per cent. salt water.

Casing

0 to 416 ft. of 13-in.
416 to 910 ft. of 10-in.
910 to 1045 ft. of 8-in.

CONCEPCION No. 15

Commenced June 5, 1910
Completed June 13, 1910

FEET	
0– 40	Dry alluvial
40– 120	Yellow clay and blue marl
120– 125	Brown oil sand, 2 bbl. oil per day
125– 145	Blue marl
145– 200	Blue marl, small oil show
200– 223	Dolomite, 2000 bbl. salt water per day, oil color.

Star Rig

Casing

18 ft. of 13-in.
200 ft. of 10-in.

JUILE No. 1

FEET	
0– 60	Yellow clay
60– 350	Blue clay
350– 370	Gray sand, salt water
370– 385	Sandy clay
385– 445	Gray quicksand
445– 510	Blue clay
510– 520	Blue clay, sandy streaks
520– 545	Blue clay
545– 555	Gray sand
555– 635	Blue clay
635– 658	Gray sand, salt water
658– 670	Hard sand, Pliocene fossils
670– 710	Blue clay, hard, sandy
710– 725	Sandy blue marl
725– 860	Gray sand loose
860– 880	Blue clay
880– 940	Sand, slightly clayey
940– 961	More clayey
961–1030	Gray salt-water sand
1030–1090	Blue clay
1090–1100	Blue clay and gypsum

Feet	
1100–1350	Blue clay
1350–1420	Very salty, dark red clay
1420–1436	Variegated red and gray clay, crystals of salt
1436–1815	Rock salt.

Tonalapa No. 1

Feet	
0– 20	Yellow clay
20– 52	White marl
52– 85	Limestone and flint, cold fresh water
85– 110	Limespar and pyrites of iron
110– 128	White marl
128– 158	Limestone
158– 182	Black shale, cold fresh water
182– 194	Limestone, very hard
194– 215	Hard black shale
215– 219	White marl
219– 260	White and brown lime, salt water
260– 333	Lime and slate
333– 380	Red rock
380–2050	Salt
2050–2055	Limestone
2055–2120	Brown shale
2120–2165	Red rock
2165–2180	Blue sandy shale
2180–2290	Salt mixed with red clay
2290–2300	Black clay
2300–2360	Black clay
2360–2418	Salt
2418–2450	Salt and black slate
2450–2453	Black slate

Sayula No. 3

Feet	
0– 45	Yellow clay
45– 200	Blue shale, gas at 135–145 ft.
200– 270	Gray and blue shale and sand, cold brackish water
270– 325	Blue shale and sand
325– 435	Blue shale
435– 455	Blue shale and sand, salt water
455– 470	Sand and shale
470– 490	Hard sand shell, water shut off.
490– 550	Blue shale
550– 555	Shale and sand, oil
555– 590	Blue shale
590– 595	Hard shell, oil
595– 615	Loose dark gray sand, salt water
615– 662	Blue shale
662– 670	Loose dark gray sand, oil
670– 815	Blue shale, salt water
815– 820	Loose gray sand
820– 870	Blue shale
870– 875	Gray salt-water sand

Feet	
875– 908	Blue shale
908– 912	Soft gray salt sand
912–1015	Blue shale
1015–1020	Shale and sand
1020–1180	Blue shale, salt water
1180–1200	Blue shale and gray sand
1200–1392	Blue shale
1392–1400	Blue shale and gray salt-water sand
1400–1570	Blue shale
1570–1640	Hard sandy shells and blue clay
1640–1825	Blue shale
1825–1830	Loose gray sand
1830–1933	Blue shale
1933–1940	Gray sand
1940–1965	Blue shale
1965–2100	Gray sand with thin layers of blue shale
2100–2153	Blue shale
2153–2160	Hard sandy shell
2160–2175	Blue shale
2175–2250	Conglomerate sand and shale

Well abandoned and casing pulled June 22, 1907.

NOTE.—This well contained salt water to within 150–175 ft. of the top of the hole all the way down; each stratum of sandy formation appeared to be flooded with salt water.

Cannel Coal and Carbonaceous Shale Deposits of Pennsylvania*

By Charles R. Fettke,† Ph. D., Pittsburgh, Pa.

(New York Meeting, February, 1923)

Before the Drake well on Watson Flats below Titusville, Pa., inaugurated the modern petroleum industry on Aug. 28, 1859, a considerable industry, based on the manufacture of mineral oils through the destructive distillation at relatively low temperatures (750° to 800° F.) of cannel coals and shales in closed, air-tight retorts, had already sprung up in Europe and America because of the increasing cost of animal and vegetable oils and fats and the growing demand for illuminating and lubricating oils. In 1860, eight out of 55 such coal-oil companies in the United States had plants located in Pennsylvania.[1] Of these, the Lucesco Works, costing $120,000, situated near Kiskiminetas Junction, was the largest operating in the country. It was equipped with 10 revolving 2.5-ton retorts and 16 stills for rectifying the oil, each holding 2000 gal. The total output of crude oil was almost 6000 gal. per day.[2]

The discovery of an abundant supply of natural petroleum by the Drake and other wells stopped the distillation of coal oil in America. Most of the smaller plants were closed and others were converted into petroleum refineries. At present, in spite of the temporary over-production of petroleum, the coal and oil-shale carbonization industries are again attracting the attention of far-sighted oil companies who realize that the gradual exhaustion of the American petroleum reserve will probably compel a gradual return to the destructive low-temperature distillation of coals and oil shales within the next 10 or 20 years.

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† Associate Professor of Geology and Mineralogy, Carnegie Institute of Technology.

[1] Charles Baskerville: Economic Possibilities of American Oil Shales. *Eng. & Min. Jnl.* (1909) **88**, 151.

[2] Thomas Antisell: "The Manufacture of Photogenic or Hydrocarbon Oils from Coal and Other Bituminous Substances," 134. 1859.

A study of the cannel coal and bituminous shale deposits of Pennsylvania has been made by the writer, in connection with an investigation of the oil resources of such coals and shales for the State Bureau of Topographic and Geological Survey. In size and extent, the deposits proved to be insignificant when compared with the oil-shale beds of the western United States, but some of their characteristics are thought to be of sufficient interest to deserve record.

Cannel coal has a fine, even, compact texture, black color, dull luster, and conchoidal cross fracture. In mining, it usually breaks into rudely rectangular blocks of varying dimensions. Cannel coals have a relatively high percentage of volatile matter compared to their fixed carbon, higher, as a rule, than that of the ordinary coals of corresponding rank. For that reason, when they are subjected to low-temperature distillation the amount of oil obtained from them is considerably greater than that obtained from ordinary coals. This oil has a lower specific gravity than the low-temperature tar produced from the latter, ranging in most cases from about 0.9 to 0.95. Its content of phenolic compounds and unsaturated hydrocarbons is also considerably less.

Cannel coals, after burial underneath a load of sediments and subjection to earth movements, have undergone devolatilization just as ordinary coals. Ashley has, therefore, classified them into brown sub-cannel of lignitic rank, black sub-cannel of sub-bituminous rank, cannel of bituminous rank, and canneloid of semi-bituminous, semi-anthracite and anthracite ranks.[3] In the case of the canneloid varieties, the percentage of volatile matter, as compared with the fixed carbon, is too low to yield any important amount of oil on carbonization.

As a rule, cannel coals have a higher ash content than ordinary coals. This is undoubtedly due to the fact that fine silt found its way more readily into the bodies of open water in which they accumulated than into the dense growth of vegetation of the ordinary peat marsh. The deposits mined on a small scale in Pennsylvania at the present time contain from 11 to 27 per cent. of ash. All gradations are found from nearly pure cannel coal, through cannel shale, to carbonaceous shale. Carbonaceous shales of the cannel type, however, give a considerably higher yield of oil than those more closely related to the ordinary type of coal, with the same percentage of organic matter. This is due to the original difference in composition of the vegetal matter from which the two have been derived.

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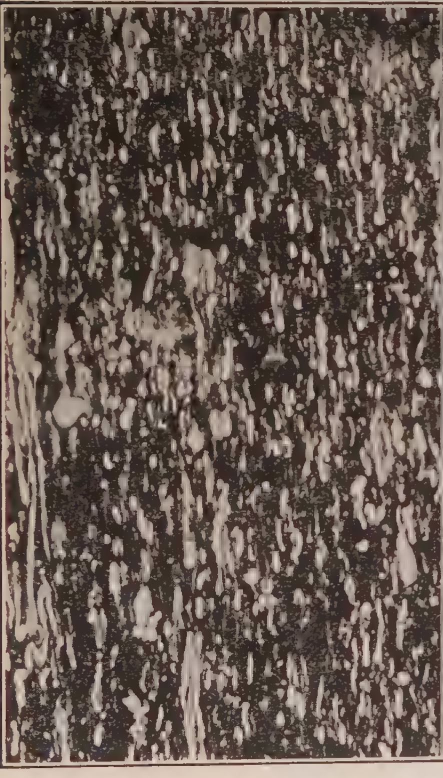
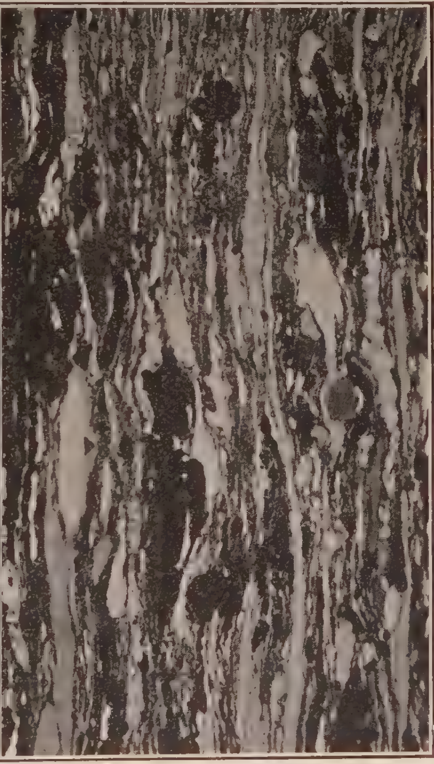
[3] George H. Ashley: Cannel Coal in the United States. U. S. Geol. Survey *Bull.* 659 (1918) 10.

Cannel coal deposits rarely have a horizontal extent comparable to that of many ordinary coal beds. The carbonaceous and cannel shales sometimes found associated with them, however, often extend over larger areas than the cannel coals themselves. Bituminous coal frequently occurs associated with the cannel in the same bed.

While cannel coals and shales may resemble one another rather closely in appearance they are not all composed of the same type of material.





Courtesy R. Thiessen.

Fig. 1.—Spore cannel coal from the Upper Kittanning horizon, Paragon Coal and Coke Co., Cannelton, Pa. × 200.

Fig. 2.—Humic cannel coal, G. W. Currie mine, 10 miles north of Nealeys, Butler Co., Pa. ×200.

Three main types amongst the Pennsylvania occurrences have been recognized by Dr. Reinhardt Thiessen of the United States Bureau of Mines, who has made a detailed and comprehensive microscopic study of the samples collected by the writer. These were described by him in a paper, "On the Origin of Petroleum," given before the American Chemical Society during the September, 1922, meeting at Pittsburgh. He has called these the true spore, the humic, and the Kiskiminetas types.

The cannel coal from the famous Cannelton deposit in northwestern Beaver County, at the Upper Kittanning horizon, is a typical representative of the true spore group. The most conspicuous and abundant constituents of this coal are a variety of spore cases. Roughly, 90 per cent. of the total organic matter consists of spore cases and macerated spore matter. The large majority of these are microspores, mostly of one type, only a few megaspores being present. Some particles of woody degradation matter and occasionally resinous appearing particles occur amongst the spores. Fig. 1 shows the appearance of this coal in thin section under the microscope.

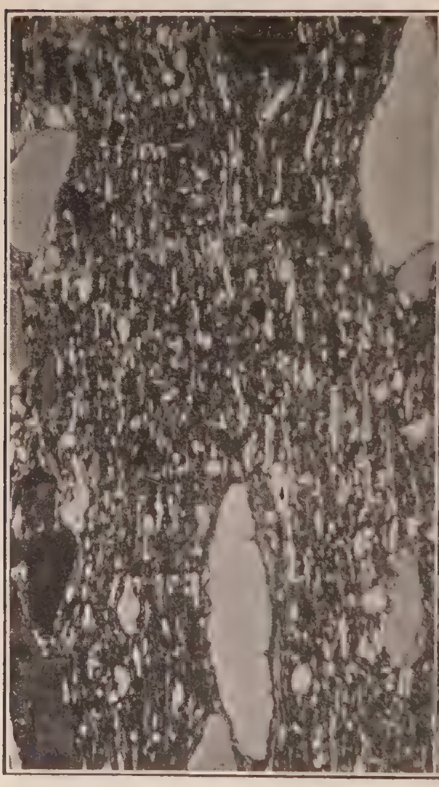
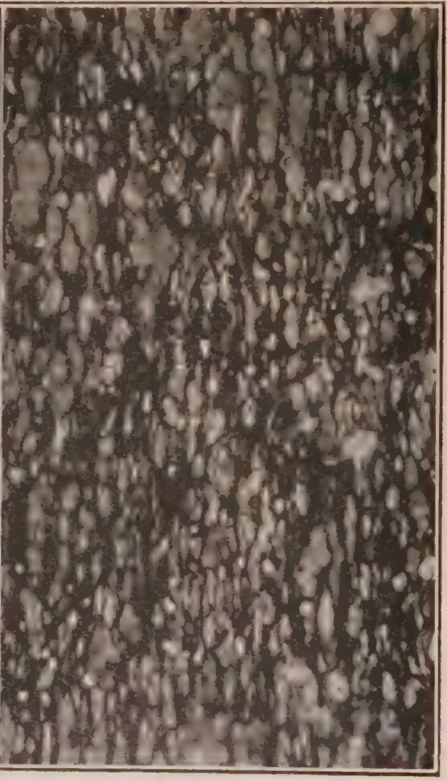


FIG. 3.—HUMIC CANNEL SHALE, UPPER FREEPORT HORIZON, DIAMOND COAL AND COKE CO., OAKMONT NO. 1 MINE, BARKING STA., ALLEGHENY CO., PA. ×200.



Courtesy R. Thiessen.

FIG. 4.—KISKIMINETAS TYPE OF CANNEL SHALE, LOWER FREEPORT HORIZON. FROM OUTCROP ¾ MILE SOUTHEAST OF KISKIMINETAS JNC., PA.

The chief constituent of the humic coals and shales is derived from the cellulosic parts of plants. With this woody degradation matter is associated a considerable amount of resinous matter which is next in importance; less important amounts of spore and cuticular matters are also present. Fig. 2 represents a cannel coal from a country bank on the Currie farm in central Butler County, which is typical of this type. The cannel shale occurring above the Upper Freeport coal in the Thick

Freeport district of northeastern Allegheny County, shown in Fig. 3, also belongs in this class.

Only two occurrences of cannel shale of the Kiskiminetas type were discovered in Pennsylvania during the present investigation. One of these, shown in Fig. 4, lies above the Lower Freeport coal in the vicinity of Kiskiminetas Junction; the other occurs at what is probably the Sharon horizon in the Pottsville, near Leesburg Station, Mercer County. The origin of the predominant organisms of these shales is not yet known. Thiessen describes them as being very irregular in structure, with scarcely any uniform characteristic except their yellow resinous appearance and transparency; in this respect they resemble the ordinary spore cases, but are perhaps a little lighter in color. Their general form is roughly lenticular to oval; the surface is exceedingly irregular, varying from merely rough to branching. The mass varies from solid to porous, sometimes showing traces of canals.

The essential difference between cannel and ordinary coals lies in the nature of the plant debris from which the two have been derived. From the character of the cannel coals, their mode of occurrence, and the shape and relation of their deposits to ordinary coals, it appears that most cannel coal deposits were laid down in open-water basins or channels in peat-forming swamps. Such open bodies of water in a swamp would gradually become filled by material of the type that the microscope has revealed to be present in cannel coal. Winds would blow spores, pollen grains, seeds, etc., from the surrounding marsh; many types of floating plants would thrive in them.

The cannel coal and shale deposits of Pennsylvania are confined almost entirely to strata of Pennsylvanian age. The horizons at which they occur, and the number of individual deposits at each horizon brought to the attention of the writer are given in the following table:

Permian Period.	
Greene formation,	
Upper Washington	1
Washington formation	0
Pennsylvanian Period.	
Monongahela formation,	
Redstone	1
Conemaugh formation,	
Bakerstown	1
Brush Creek	2
Allegheny formation,	
Upper Freeport	4
Lower Freeport	2
Upper Kittanning	15
Middle Kittanning	2
Lower Kittanning	2
Pottsville formation,	
Sharon (?)	1

TABLE 1.—*Analyses of Three Typical Samples of Cannel Coal and Shale*

	Cannel Coal, Upper Kittanning, Per Cent. (*A*)	Cannel Shale, Lower Freeport, Per Cent. (*B*)	Cannel Shale, Upper Freeport, Per Cent. (*C*)
Proximate Analysis:			
Moisture	0.73	0.73	0.18
Volatile matter	39.22	26.62	32.12
Fixed carbon	32.57	26.45	36.17
Ash	27.48	46.20	31.53
Ultimate Analysis:			
Hydrogen	4.97	3.70	4.58
Carbon	60.00	43.27	57.35
Nitrogen	1.01	0.76	1.19
Oxygen	4.29	4.49	3.12
Sulfur	2.25	1.58	2.23
Ash	27.48	46.20	31.53
*Carbonization Test:**			
Volume of oil	50.1 gal.	37.5 gal.	38.2 gal.
Light oil recovered from gas	2.8 gal.	2.4 gal.	2.6 gal.
Ammoniacal liquor	8.2 gal.	11.7 gal.	10.2 gal.
Ammonia	3.24 lb.	2.07 lb.	5.01 lb.
Calculated as ammonium sulfate	12.57 lb.	8.03 lb.	19.43 lb.
Weight of residue	1332 lb.	1460 lb.	1446 lb.
Volume of gas, sat. with water, at 60° F. and 30-in. mercury	6266 cu. ft.	4602 cu. ft.	6501 cu. ft.

(*A*) Mine of Paragon Coal and Coke Co., on Cannel Run, north of Cannelton, Beaver Co.

(*B*) From outcrop ¾ mile southeast of Kiskiminetas Jnc., Westmoreland Co.

(*C*) Valley Camp mine No. 3, on Pucketa creek, 1 mile east of Parnassus, Westmoreland Co.

* Made in 3500-cc. vertical retort, heated gradually to 760° C. during 8 hr. Yields computed on basis of 2000 lb. raw material.

The largest deposit of cannel coal or shale thus far discovered in Pennsylvania occurs at the Upper Freeport horizon in Allegheny County. Over a considerable portion of northeastern Allegheny County the Upper Freeport coal is remarkably well developed, sometimes reaching a maximum thickness of 10 ft.; this area is commonly spoken of as the "Thick Freeport" district. Over about 48 square miles the bituminous coal is overlain by a bed of cannel shale ranging in thickness from 10 in. to 3.5 ft. and averaging about 2 ft. over the entire area. Ten representative samples from widely separated localities gave oil yields ranging from 13.4 to 39.1 gal. per ton. The average yield, weighted according to the thicknesses represented by these 10 samples, was 26.6 gal. Approximately 100 million tons of cannel shale of this average yield are present in the deposit. Unfortunately, the bituminous coal is rapidly being mined out and the cannel shale is being left in such a condition that it will not

TABLE 2.—*Distillation of Oil from Upper Kittanning Cannel Coal (A)*

Specific gravity of dehydrated oil, 0.9452; sulfur, 0.32 per cent.; nitrogen, 0.73 per cent. Air distillation: Bar., 29.114 in.; first drop, 44° C.

Temperature, Deg. C.	Per Cent. by Volume	Sum, Per Cent.	Per Cent. Cut	Speific Gravity of Cut
Air Distillation				
To 50	trace	trace	15.44	0.7753
50 to 75	1.33	1.33		
75 to 100	1.07	2.40		
100 to 125	3.80	6.20		
125 to 150	4.47	10.67		
150 to 175	4.77	15.44		
175 to 200	5.93	21.37	13.43	0.8730
200 to 225	7.50	28.87		
225 to 250	6.40	35.27	13.80	0.9137
250 to 275	7.40	42.67		
Vacuum Distillation at 45 mm.				
To 200	4.33	4.33	40.00	0.9731
200 to 225	8.00	12.33		
225 to 250	8.67	21.00		
250 to 275	8.67	29.67		
275 to 300	10.33	40.00		
Residuum, by diff.	17.33		17.33	1.0806
	100.00		100.00	

Carbon residue of residuum (Conradson method), 11.0 per cent.

be accessible by the time that petroleum prices shall have reached a point where it will become profitable to recover oil from material of this character.

In the vicinity of the junction of the Allegheny and Kiskiminetas Rivers, near Freeport, a bed of cannel shale covering approximately 2.15 square miles occurs associated with the Lower Freeport coal. Its relationship to the Upper and Lower Freeport coals is brought out by the following section exposed along a small run flowing northward into the Kiskiminetas River, ¾ mile southeast of Kiskiminetas Junction:

	FEET
Sandstone roof	
Upper Freeport coal	3.5
Clay	3
Limestone	2
Platy sandstone	25
Carbonaceous shale	2
Cannel shale	2.75
Dark gray shale	7
Lower Freeport coal	2.67
Gray shale.	

TABLE 3.—*Distillation of Oil from Lower Freeport Cannel Shale (B)*

Specific gravity of dehydrated oil, 0.8790; sulfur, 0.33 per cent.; nitrogen, 0.48 per cent. Air distillation: Bar., 29.316 in.; first drop, 51° C.

Temperature, Deg. C.	Per Cent. by Volume	Sum, Per Cent.	Per Cent. Cut	Specific Gravity of Cut
Air Distillation				
To 50	none	none	13.8	0.7524
50 to 75	0.9	0.9		
75 to 100	1.1	2.0		
100 to 125	3.4	5.4		
125 to 150	4.0	9.4		
150 to 175	4.4	13.8		
175 to 200	5.2	19.0	11.3	0.8105
200 to 225	6.1	25.1		
225 to 250	6.8	31.9	14.8	0.8414
250 to 275	8.0	39.9		
Vacuum Distillation at 46 mm.				
To 200	4.7	4.7	37.1	0.8960
200 to 225	5.7	10.4		
225 to 250	8.3	18.7		
250 to 275	8.7	27.4		
275 to 300	9.7	37.1		
Residuum, by diff.	23.0		23.0	0.9550
	100.0		100.0	

Carbon residue of residuum (Conradson method), 4.3 per cent.

The average yield from five samples representing different portions of the deposit was 36.5 gal. of oil per ton. In thickness, the shale averages 2.5 to 3 ft., so that 7 to 8 million tons of cannel shale were originally present in this locality. The deposit is of considerable historical interest on account of the flourishing "coal" oil industry which it supported during the period just prior to the development of the petroleum industry in the United States. Three of the eight coal-oil operators in Pennsylvania in 1860 had plants located in the vicinity of Kiskiminetas Junction, among which was the Lucesco Works previously mentioned. Mining operations in the past apparently were confined to that portion of the deposit lying east of the Allegheny River, so that nearly one-half of the area still represents virgin ground.

The famous Cannelton bed of cannel coal is situated in the northwestern corner of Beaver County. This deposit appears to occupy an old channel at the Upper Kittanning horizon which had the shape of a huge ox-bow, as shown in Fig. 5. The channel has a total length of nearly

TABLE 4.—*Distillation of Oil from Upper Freeport Cannel Shale (C)*

Specific gravity of dehydrated oil, 0.8975; sulfur, 0.55 per cent.; nitrogen, 0.62 per cent. Air distillation: Bar., 29.120 in.; first drop, 45° C.

Temperature, Deg. C.	Per Cent. by Volume	Sum, Per Cent.	Per Cent. Cut	Specific Gravity of Cut
Air Distillation				
To 50	0.27	0.27	18.20	0.7725
50 to 75	1.13	1.40		
75 to 100	2.20	3.60		
100 to 125	2.80	6.40		
125 to 150	4.93	11.33		
150 to 175	6.87	18.20		
175 to 200	4.80	23.00	11.87	0.8472
200 to 225	7.07	30.07		
225 to 250	8.07	38.14	20.14	0.8706
250 to 275	12.07	50.21		
Vacuum Distillation at 48 mm.				
To 200	1.33	1.33	33.32	0.9108
200 to 225	4.33	5.66		
225 to 250	10.33	15.99		
250 to 275	8.33	24.32		
275 to 300	9.00	33.32		
Residuum, by diff.	16.47		16.47	1.0054
	100.00		100.00	

Carbon residue of residuum (Conradson method), 7.6 per cent.

3½ miles and a width of only 600 ft. According to White,[4] who visited the locality in 1876, and I. F. Mansfield, who formerly mined the deposit on an extensive scale, the cannel coal reached a thickness of 12 ft. along the middle of the channel, although in most places the thickness did not exceed 7 ft. A good thickness was maintained for a width of about 300 ft. on both sides of which the coal thinned rapidly. Overlying the cannel coal along the middle of the channel is as much as 6 ft. of cannel shale. This also thins comparatively rapidly on both sides, but probably extends over a greater width than the cannel coal. Underlying the cannel coal is a thin bench of bituminous coal. A little bituminous coal also occurs interbedded with the cannel in places. At the extreme northwest and southwest ends of the ox-bow, the cannel coal and shale disappear entirely and only bituminous coal remains. The cannel coal from the Cannelton deposit yields an average of 50 gal. of oil per ton, while the shale yields 33 gal.

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[4] I. C. White: Report of Progress in the Beaver River District of the Bituminous Coal Fields of Western Pennsylvania. *Report Q*, Second Geological Survey of Pennsylvania (1878) 223–234.

The existence of the Cannelton deposit, according to I. F. Mansfield, was known to the Indians at least as early as 1750. Mining on a small scale was commenced early in the last century, but it was not until about 1855, when the cannel coal and the richer portions of the cannel shale associated with it began to be sought for the manufacture of oil, that extensive operations were undertaken. Four companies produced oil from it in 1860. From that time mining was carried on practically continuously until about 1900, when the thicker portions of the deposit

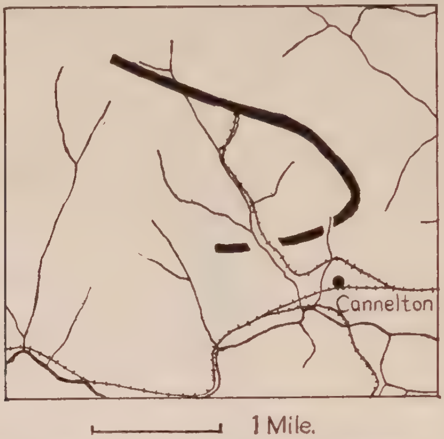

FIG 5.—EXTENT OF UPPER KITTANNING CANNEL COAL AT CANNELTON. BEAVER CO., PA.

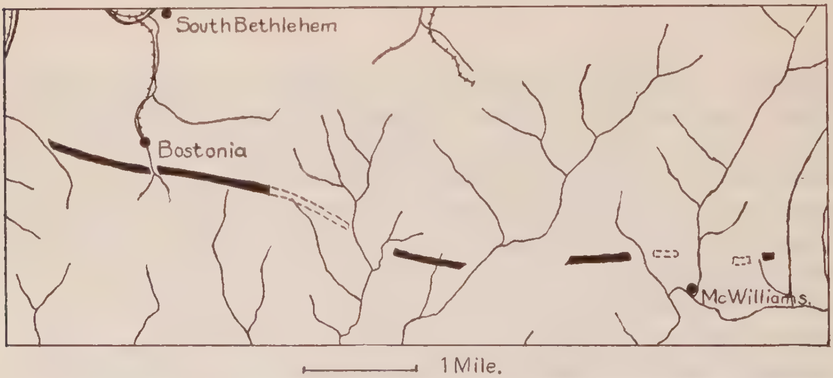

FIG 6.—UPPER KITTANNING CANNEL COAL, NORTHERN ARMSTRONG CO., PA.

had practically been worked out. Operations since that time have been only of a desultory character.

The writer estimates that a maximum of about 2 million tons of cannel coal were originally present in the Cannelton deposit and perhaps an equal tonnage of cannel shale. On account of the wasteful methods of mining employed in the early days, probably not more than one-half of the coal was extracted; most of the cannel shale also remains. On account of the poor condition of the underground workings of the old mines, it is extremely doubtful whether it will be possible to extract profitably any great percentage of the cannel coal and shale still remaining.

Another channel-like deposit of cannel coal, also at the Upper Kittanning horizon, occurs in northern Armstrong County. It extends from a point about two miles southwest of South Bethlehem in an easterly direction for a distance of about 6.7 miles to a point about ¾ mile northeast of McWilliams, as shown in Fig. 6. Nearly one-half of the original deposit has been removed by erosion. The black areas in the figure show those portions of the channel that have either been mined out or are being mined at present; the dotted areas show virgin ground. The maximum width of the channel over which coal of minable thickness has been found is approximately 800 ft. The cannel coal has its greatest development along the center line of the channel and thins comparatively rapidly on both sides. The bituminous coal, which immediately underlies the cannel, maintains a more nearly uniform thickness over the entire area. Considerable variations in thickness of the cannel coal also occur along the center line of the channel. Fig. 7 shows a cross-section of the channel in the No. 9 mine of the Pine Run Coal Co., at Bostonia. The entire tonnage of minable cannel and bituminous coal originally present in these portions of the channel which escaped erosion was probably not much in excess of 4 million tons.

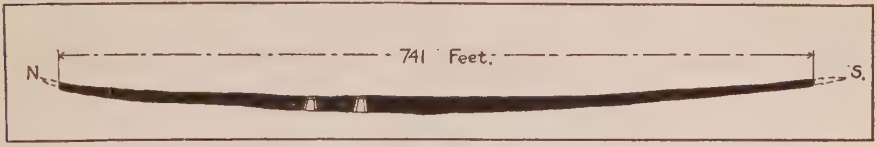

Fig. 7.—Cross-section at No. 9 mine, Pine Run Coal Co., Bostonia, Armstrong Co., Pa.

The above described deposits are the most important thus far discovered in Pennsylvania. All the others examined were of relatively insignificant size. While there is always a possibility of discovering other similar deposits in the area of high-volatile coals west of the 65 isocarb line, particularly in the Allegheny formation to which most of the cannel coal and shale occurrences of the state are confined, and especially in places where this formation is covered by overlying strata and does

TABLE 5.—*Composition of Air Distillation Fractions (All Percentages by Volume)*

Fraction, Deg. C.	Per Cent. of Crude Oil	Sp. Gr. Fraction (*a*)	Per Cent. Tar Acids in Frac. (*b*)	Per Cent. Tar Bases in Frac. (*c*)	Per Cent. Neutral Oil in Frac.	Per Cent. Neutral Oil Removed by 95 Per Cent. H_2SO_4	Per Cent. Neutral Oil Removed by 98 Per Cent. H_2SO_4	Per Cent. Saturated Hydrocarbons in Frac.
Upper Kitanning Cannel Coal (*A*)								
Light oil from gas........		0.6982	0.0	0.3	99.7	24.2	39.6	60.2
To 175.......	15.44	0.7753	4.8	0.5	94.7	29.0	44.6	52.5
175 to 225....	13.43	0.8730	21.8	2.8	75.4	30.0	52.0	36.1
225 to 275....	13.80	0.9137	15.0	5.5	79.5	37.0	61.0	31.0
Lower Freeport Cannel Shale (*B*)								
Light oil from gas.........		0.6887	0.6	0.3	99.1	24.0	40.8	58.7
To 175.......	13.8	0.7524	1.5	0.5	98.0	24.0	43.5	55.4
175 to 225....	11.3	0.8105	5.5	1.0	93.5	24.0	46.0	50.5
225 to 275....	14.8	0.8414	3.8	1.8	94.4	27.0	50.0	47.2
Upper Freeport Cannel Shale (*C*)								
Light oil from gas.........		0.7055	0.2	0.3	99.5	23.0	37.0	62.7
To 175.......	18.20	0.7725	7.3	1.0	91.7	25.6	40.0	55.0
175 to 225....	11.87	0.8472	16.2	1.6	82.2	23.0	40.0	49.3
225 to 275....	20.14	0.8706	8.9	2.8	88.3	24.0	44.0	49.4

(*a*) At 15.56° C. (*b*) Removed by 10 per cent. NaOH solution. (*c*) Removed by 20-per cent. H_2SO_4.

TABLE 6.—*Composition of Residue from Carbonization Tests*

	Cannel Coal, Upper Kittanning, Per Cent. (*A*)	Cannel Shale, Lower Freeport, Per Cent. (*B*)	Cannel Shale, Upper Freeport, Per Cent. (*C*)
Moisture....................	0.5	0.6	1.0
Volatile matter..............	2.6	0.6	1.7
Fixed carbon................	52.7	32.4	53.8
Ash.........................	44.2	66.4	43.5
Nitrogen....................	0.52	0.28	0.79
Sulfur......................	1.82	1.66	1.82
Character of residue..........	Non-fused; particles intact, but tend to adhere.	Non-fused.	Non-fused; particles intact, but tend to adhere.

TABLE 7.—*Comparative Oil Yields of the Three Samples*

Sample	Volatile Matter Plus Fixed Carbon in Original Sample, Per Cent.	Average Yield per Ton by Carbonization to 760° C., in 3500-cc. Retort			Yield per Ton on Ash- and Moisture-free Basis		
		Oil, Gal.	Gas, Cu. Ft.	Fixed Carbon in Residue, Lb.	Oil, Gal.	Gas, Cu. Ft.	Fixed Carbon, Lb.
Cannel coal (*A*) Upper Kittanning	71.79	50.1	6266	702	69.8	8728	978
Cannel shale (*B*) Lower Freeport	53.07	37.5	4602	473	70.7	8671	891
Cannel shale (*C*) Upper Freeport	68.29	38.2	6290	752	55.9	9211	1101

not crop at the surface, it is not likely that sufficient quantities of such coals or shales are present to support any large cannel coal or shale-oil industry in Pennsylvania.

Before attempting to erect plants to recover oil from cannel coals and shales from any particular deposit, too much stress cannot be laid on the necessity of first making careful and thorough investigations as to the extent of the deposit, and the quantity and character of the oil which the material will yield. The deposits are so erratic in these respects that careful prospecting, preferably by rather close core drilling, is necessary to obtain the essential data.

The results of carbonization tests on three typical samples of cannel coal and shales from Pennsylvania are summarized in the accompanying tables, the proximate and ultimate analyses having been made for the Pennsylvania Bureau of Topographic and Geological Survey by the United States Bureau of Mines, at its Pittsburgh Station.

DISCUSSION

RALPH ARNOLD, Los Angeles, Calif.—Are they using these oils commercially now?

C. R. FETTKE.—Not at the present time.

RALPH ARNOLD.—It is a reserve then.

C. R. FETTKE.—Yes.

S. LINKER, Brooklyn, N. Y.—What relation has the carbonaceous shale to the cannel coal and does the cannel coal ever grade directly into the carbonaceous shale, or must there be present an intervening deposit of cannel shale? Is it possible to find a cannel shale deposit that is not associated with cannel coal? So far as known, the oil shales in South Africa are not associated with cannel coal, and they have a canneloid appearance both macroscopically and microscopically, so that they may be considered a cannel shale. What is the difference between cannel shale, bituminous shale, and carbonaceous shale? Should these terms be used synonymously and indiscriminately, or should they be used with a definite meaning in mind? A short definition of each of these terms, when used, would clear the oil-shale technology of much confusion.

Charles R. Fettke.—I have been applying the term carbonaceous shale to shales that contain organic debris which was deposited contemporaneously with them at the time that they were laid down. I do not include under that term shales that later became impregnated with oil from some outside source.

An examination of the literature shows that the terms carbonaceous and bituminous shale are given different meanings by different writers; it is therefore difficult to decide which one should be applied to a particular case. The term oil shale should be restricted to those carbonaceous shales which, on carbonization, will yield sufficient quantities of oil to be worthy of consideration as possible sources of oil on a commercial scale.

H. A. Buehler, Rolla, Mo.—Missouri has peculiar beds of cannel coal. They are sink-hole deposits, in some cases 100 ft. thick, although restricted in lateral extent; they may not be over 200 ft. long by 100 ft. wide. In the total thickness there are layers that would be called carbonaceous shale. The deposits will distil up to 60 gal. per ton. It makes a rather good domestic coke and if the deposits were larger we would be able to use the coal.

Charles R. Fettke.—While there are no cannel-coal deposits in Pennsylvania that equal the Missouri occurrences in thickness, some of the channel-shaped deposits, which may be 8 miles long and only 800 ft. wide, frequently are 15 ft. thick along the center line of the channel.

Francis Nicholson, Charlotte, W. Va.—What method of distillation is used; also what type of retort?

Charles R. Fettke.—Two types of retorts were used during the present investigation. Preliminary tests were made on the samples in the oil-shale assay retort, which has been developed at the Salt Lake City station of the Bureau of Mines. A larger cast-iron vertical retort, 18 in. long, with a 4-in. internal diameter, equipped with a thermocouple pyrometer and set in a gas-fired brick furnace was used to obtain sufficient quantities of oil for fractional distillation and other tests. The temperature of the retort was raised gradually over an interval of 8 hr. Oil usually stopped coming off at the end of 4 to 5 hr. when a temperature of 900° F. had been reached. After that only gases were evolved.

FRANCIS NICHOLSON.—What was the gravity of the crude oil obtained?

CHARLES R. FETTKE.—I have not the figures, in terms of Baumé, but the specific gravity usually ran between 0.9 and 0.95. The Cannelton cannel coal, which is a spore coal, yielded oil with a specific gravity of 0.9452; the Lower Freeport cannel shale, an oil that has a specific gravity of 0.8790; and the Upper Freeport cannel shale, an oil that has a specific gravity of 0.8975. Specific gravity determinations were made upon the dehydrated oils at 15.56° C.

RALPH ARNOLD, Los Angeles, Calif.—The last named oil would be about 26° to 30° Bé.; the 0.875 gravity oil would bé 30° Be.; and the 0.95 oil would be 17° Bé.

Note on a Scheme for the Study of the Physics of Cast Iron

By Richard Moldenke, Watchung, N. J.

Note.—At the request of the American Society for Testing Materials, this paper was presented at the New York Meeting, February, 1923, in connection with the "A. S. T. M. Tentative Specifications for Foundry Pig Iron" and "The Physics of Cast Iron and Its Bearing on All Cast-iron Products and Specifications for Cast Iron," by William R. Webster. The discussion of this paper was not published by the Institute, having been referred to the American Society for Testing Materials, as these papers were again presented for discussion at the meeting of that Society at Atlantic City, June, 1923. The result of these discussions is shown in the following abstract from the Committee A3 report on cast-iron specifications:

> During the past year, the adaptability of the tentative specifications for pig iron for the steel and the malleable foundry, in addition to the iron foundry proper, have been studied and suitable changes and rearrangements are being made which appear to cover the situation properly. In spite of the experience of foundrymen to the contrary, there is a persistent call from consumers of castings for definite compositions for given lines of castings. To respond to this call for information desired by the consumer—information every modern foundryman should have and guide his procedure with—the committee has requested the Sub-Committee on General Castings to take up this matter with a view of constructing tables of analyses ranges for important lines of foundry product. These analyses ranges, however, are to be given as "recommended practice" and not intended for actual specification. In this way, the consumer of the foundry product will get the information he wants; the furnaceman will be guided in the production of his pig irons to prepare for analyses ranges which are more closely standardized and have less of "freak" requirements made by inexperienced metallurgists; and the foundryman will not be bound by chemical requirements he knows will not work out well with his particular regional supplies of melting stock.

Doctor Moldenke's paper is published in full as it shows clearly the good results of the Institute's work in 1895 and a comparison of the same with the present knowledge of the metallurgy of cast iron.—Bradley Stoughton, *Chairman, Iron and Steel Committee.*

In March, 1895, William R. Webster contributed a carefully prepared scheme for the study of the physics of cast iron to the American Institute of Mining Engineers at its Florida meeting. This summary of the interrelation of the chemical and physical characteristics of cast iron with the operations of daily foundry practice caused a wide discussion of the then little understood though universally used material. Mr. Webster has requested the author to bring the subject up to date and, while glad to comply, he must pay that with all the knowledge that has been gained in the quarter century that has passed since the subject was brought to the attention of the Institute—and we now are pretty familiar with cast iron in all its aspects—there is practically nothing to change and little to add to the facts as given by Mr. Webster originally,[1] and in the discussions.

The suggested lines for discussion and investigation as they would appear at the present time are as follows:

I. Relation between chemical composition and physical properties, such as fusibility, fluidity, set, contraction, chill, hardness, strength, specific gravity, etc.

II. Influence of variations in foundry practice upon chemical composition and physical properties, as in the following summary:

Mixture Characteristics.—Pig and scrap proportions, steel additions, charcoal or coke irons, "direct" blast-furnace metal.

Metals' Characteristics.—Clean or with sand, rust, "burnt" scrap, shot, borings, salamanders and heavy section pieces.

Melting Characteristics.—Rational or poor charging and melting practice in cupola or furnace, normal hot iron or oxidized "lifeless" quick-setting metal. Duplexing in electric furnace for better superheat, deoxidation and desulfurization.

Temperature Characteristics.—Pouring temperature and rate of cooling due to mold surface characteristics, whether green or dry sand, metal molds, use of chills, section of castings, refractory quality of mold, gating of molds.

Molecular Treatment.—Annealing, seasoning, rumbling, repetition, heating and cooling.

Effect on

Grain structure.
Internal shrinkage, porosity.
Blowholes.
Hardness.
Strength.
Internal stress.
Carbon condition.
Changes in other contained elements.

The following bad characteristics in castings are sufficiently important to warrant special mention in separate paragraphs:

III. *Shrinkage and Segregation.*—As specially affected by composition, oxidized metal through bad melting practice, cold metal, irrational gating methods, insufficient feeding opportunity, low ferrostatic pressure.

IV. *Blowholes.*—As specially affected by oxidized ("burnt") metal in charges or through bad melting practice, cold iron, low ferrostatic pressure, wet mold surfaces, wet or insufficiently vented cores.

While the scheme for study outlined in 1895 differs but little from the one above given, the conditions obtaining in foundry practice today are, nevertheless, vastly improved. Buying pig iron by its analysis is the rule everywhere and fracture is seldom an issue. The foundry foreman of today has no difficulty in calculating his mixtures chemically, and the composition of the various types of castings are almost standardized. Melting practice has now been placed upon a scientific basis in the foundry, so that there is little excuse for heavy losses from inferior metal. There still remains, however, a critical study of the principles and practice connected with the rational gating of molds.

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[1] *Trans.* (1895) **25**, 84, 964; (1896) **26**, 997.

A discussion of every point included in the above scheme of study would assume the bulkiness of a book, hence no attempt will be made in that direction. The summary of a life's work in the production of good castings—necessarily interwoven with the chemical and physical characteristics of cast iron—is as follows:

1. Use the best materials suited to the production of the castings wanted.
2. Employ correct melting, molding and finishing process.
3. Gate the molds correctly, so that the good materials, properly manipulated, may be given the best chance possible to produce good work.

It was through the active work of the American Institute of Mining Engineers in the study of pig iron that the change was brought about from the purchase of pig iron by its fracture to its classification and purchase by its chemical composition. That work was taken up by the Committee on Cast Iron of the American Society for Testing Materials, which brought into general use its specifications governing the grading and purchase of pig iron.

The present condition of the pig iron situation is about the same as the status of steel when the change was made from iron to the use of steel for structural purposes. At that time, every engineer desired to have his name on his specifications. Since then, the whole matter has been cleared up and we now have standard specifications which are in general use. These results were brought about by giving the producers and the consumers equal representation and vote on the committees preparing these specifications. This same plan can be followed to great advantage in pig iron. Under the present conditions, the requirements of the founders sent to the blast-furnace men are not uniform even when they are making the same grade and weight of castings and many requirements are "hair splitting," which entails unnecessary trouble to all parties. The natural solution of the present condition is to collect the requirements under which pig iron has been sold for each of the grades of castings, manufactured malleable iron, acid open-hearth steel, basic open-hearth steel, etc., the work to be done by a committee having an equal representation of blast-furnace men, whom we term the producers, and the consumers who manufacture the finished product. The requirements to insure reliability of their finished product have already been put into use and need not be further considered.

If this course is followed, it would enable any blast furnace in the country to produce a much larger output on a given grade of pig iron than ever before, as the number of grades would be reduced and the hair-splitting requirements removed. In dull times, this would enable the furnaces to stock up considerable of each grade of iron with much more certainty than under the present conditions.

Federal Taxation of Mines

By L. C. Graton, Washington, D. C.

(Chicago Meeting, September, 1919)

The federal taxes on incomes and excess profits are of course heavy. In 1917, the value of the mineral production of the United States was a little in excess of $5,000,000,000. The total of federal taxes for that year, for the mineral producing companies, as computed by themselves, amounts, according to tentative compilations, to about $207,000,000, and the total for the comprehensive group of mineral and metal industries to about $766,000,000, which figures represent respectively 9.5 per cent. and 35.1 per cent. of the total tax paid by all corporations. Because of greater complexities in applying the revenue laws to mineral properties than to most other industries, opportunity is afforded for wide differences of view. The taxes indicated above, therefore, are not final and may conceivably be very materially increased. For 1918, the taxes are expected to be still greater, but no compilations are yet available. Taxes at a high rate are to continue indefinitely. The subject is clearly of gravest import to the mining industry in particular.

The Commissioner of Internal Revenue, who is charged by law with the administration and collection of the taxes, is determined that these functions shall be performed in a just and businesslike way. Fairness and finality are to be the chief objectives. In view of the peculiar conditions that attach to the tax program as applied to mines and other wasting industries, like oil, gas, and timber, the Commissioner has resorted to professional advice and assistance by manning a Sub-division of Natural Resources with competent engineers, brought in from the several industries, into whose hands he has placed for solution the technical questions involved.

These engineers have a two-fold duty: (1) to devote to the solution of these problems their own knowledge and experience; (2) through their acquaintance with their respective industries, and with the men engaged in those industries, to focus upon these problems, by conferences and otherwise, the best opinion of the country. The principal object of this paper is to acquaint the mining profession with the attitude of the

Bureau of Internal Revenue and to enlist to the fullest possible extent its interest and professional coöperation.

Attack upon the problems of mine taxation must necessarily start from a clear understanding of mining economics, of which the prominent and distinctive features are hazard and exhaustible character of assets. Closely related to this last feature is the necessity of maintaining efficient mining organizations indefinitely instead of limiting their existence to the life of the particular mine now being worked.

The principal questions to be settled revolve about the matter of mine valuation, but they are complicated by the fact that most of the values must be established as of a date several years ago. Among a variety of valuation methods that might be employed, the one direct, professional, and established method is that which capitalizes income by determining the present value of total expected earnings.

For the general application of this present-value method, there must be established a set of factors intended to adjust it to those examples which afford less data than the amount it requires ideally. Investigations must be conducted with more or less detail and thoroughness into a variety of subjects which enter into the application of the method, such as relation of interest rates to hazard, ratio of proved to prospective ore in various types of deposits, effect of change in grade of ore and in rate of output, estimate of future production cost and selling price of product, what constitutes discovery of a mine, and methods of depletion and depreciation.

In the determination of all these matters closest touch with the industry will be maintained, with the sole idea of establishing principles that will be so sensible and sound that they may apply not only to the final settlement of taxes for the years now under review but may govern likewise through subsequent years so long as present or similar revenue laws prevail. In this way, it is hoped that the suspense and confusion that now beset the mining industry because of the uncertainty pervading the taxation atmosphere may be materially alleviated if not wholly removed. Sympathetic coöperation on the part of the industry is absolutely essential to the attainment of this object.

Introduction

Proper Relation Between Taxpayer and Government

At the outset of the discussion of so difficult and critical a subject as the fair taxation of our mines by the federal government, it may be well to put into words certain fundamental and axiomatic truths as a basis for further discussion and argument. Several of these facts are true independently, others are consequences of some that precede.

1. Government, according to modern conceptions, is no longer to be regarded as deriving its supreme and sovereign authority from some external and fictitious source, such as is implied in the phrase, "The divine right of kings." Modern government is merely an organization of society for the conduct of the functions and the life of the people in a more effective, just, and satisfactory manner than would be possible by the independent effort of individuals.

2. What we call "the Government," meaning the specific machinery and personnel of government, is thus in fact a servant which society engages for the performance of certain of its work.

3. Taxes are funds that society puts at the disposal of its servant for the accomplishment of the needed work.

4. The fundamental object of taxation is, therefore, not to impose a burden or a penalty on the taxpayer; rather quite the opposite. The sole object of taxation is to secure the funds by means of which the government may be enabled to extend to the taxpayer those benefits that he could not so well, if at all, secure for himself.

5. Consequently, any taxation program should be administered with the utmost sympathy and consideration and with the least possible disturbance of the taxpayer's normal activities. Justification for deviation from such a sympathetic and coöperative attitude can arise only in case the taxpayer shows unwillingness to carry through his part of the arrangement. An attitude of arrogance or of suspicion adopted at the outset by the tax-collecting agency toward the taxpayer would be as much out of place as in the case of an ordinary employee toward his employer; and such a policy would inevitably kill that spirit of coöperation which certainly ought to prevail if for no other reason than to secure the best results most easily and simply and continuously.

6. It goes without saying, on the other hand, of course, that a tax levied by a law enacted by the agency which society has created and maintained is not something to be dreaded or evaded. The payment of a tax should carry with it a very genuine conviction on the part of the taxpayer that it is for value received and should be regarded in the same way as the rent bill, the meat bill, or the wages of the cook, none of which the sensible and fair man will either begrudge or endeavor to escape, but on the contrary will ascertain to be correctly and fairly computed and will then pay promptly and fully.

7. Finally, rather for the sake of completeness than because reference to the matter is necessary, let it be added that the tax-collecting agency, charged as it is with the heavy responsibility to the society that is its creator and master, must act with decision and firmness at all times, and in any case of evident and deliberate transgression of the spirit of the tax obligation must unhesitatingly resort to use of the authority with which it is empowered.

Object and Scope of Paper

It is my purpose to address you, not only as an employee of the Bureau of Internal Revenue, which is charged with assessment and collection of the Federal taxes, but also as a member of this Institute, as one whose normal activities are confined to the mining industry and whose fundamental interest is in its welfare, but who, mainly by chance and for a very short time, has had an opportunity from the inside to come in contact with the tax-collecting organization and thus to see and learn a number of things in which many of you are likely to be interested but with which, perhaps, not all of you have had occasion to become fully acquainted. I wish, therefore, to discuss with you in entire frankness, as one engineer to a group of engineers, the problems involved in the application of the present tax laws to the mining industry and the best methods by which those problems may be solved. I cannot emphasize too strongly, and I trust you will bear definitely in mind throughout, that my remarks reflect personal, not official opinions, that they may possibly be suggestive but certainly are not authoritative.

I am not sent here as a propagandist for the Government. The idea that this subject should be presented before you arose from a recognition of the magnitude of the task and from a realization of the assistance that might be gained from such an organization as this Institute. A few weeks ago, I proposed the idea to the Head of the Income Tax Unit, who readily gave assent and approval to the suggestion.

Furthermore, this outline of the situation with regard to mine taxation is presented by me rather than by some one of greater experience and authority both in general and in the specific problems of mine taxation only for these two reasons: because no time ought to be lost in initiating a spirit of coöperation between the Government and the mining taxpayer, and because in the building up of that part of the Bureau's organization which will deal with the specific problems relating to mines, it has happened that, until a few weeks ago, I was the only representative of the mining industry proper, though oil and gas and timber are already pretty well provided for. In the consideration of the topics presented in this paper, I have derived much benefit from discussions with two of my colleagues in the Bureau, J. C. Dick and J. H. Hance, valuation engineers in the Mining and the Oil Sections, respectively. This paper has been somewhat revised since its presentation at the Chicago meeting in order to place in connected relation to its main structure certain ideas suggested by the discussions of others; reference to the source of these ideas is made by footnotes.

The purpose of this paper is essentially twofold: First, to bring what I trust may be regarded as good news; second, to invoke from you such coöperation, advice and assistance as can come from no other group of men in so full a measure or so authoritative a fashion.

Attitude of the Bureau of Internal Revenue

Until very recently, the Bureau of Internal Revenue has been a relatively subordinate branch of the Government, and has had little attention from the average citizen. Since the passage of the income-tax amendment in 1913, however, and particularly since the rise of needs for enormous war funds, the Bureau has become, as the Commissioner has said, "an arm of the Government reaching out to every citizen and establishing a direct fiscal relationship with every business enterprise in the United States." An inevitable consequence of this tremendous increase in prominence and power and responsibility has been a growth and improvement in point of view and in methods held and practised by the Bureau, and I think it but fair to say has brought about, directly or indirectly, a higher type and average of personnel in positions of responsibility.

As all are aware, the revenue laws in so far as they specifically touch on the matters which relate to the mining industry are nearly always brief and frequently not superlatively clear. These, like all other portions of the revenue laws, are made subject for their interpretation and application to regulations to be established by the Commissioner of Internal Revenue. It is most illuminating to observe how, as the importance and responsibility of the Commissioner's duties in this respect have expanded with the increasing magnitude of the taxes, the regulations with regard to the mining provisions of the law have with successive years reflected a greater and greater approach toward that two-sided fairness and equity which alone can be satisfactory and can lead to decisions and settlements that will hold with finality. It is true that of the older points of view and the past methods some that are imperfect still survive in the interior structure of the tax-collecting machine, but I assure you very confidently that Mr. Roper, the present Commissioner, and Mr. Callan, who, next to the Commissioner, is in charge of the entire income-tax administration, are thoroughly imbued with the desire and the determination to administer the taxation program in an absolutely fair and businesslike way. In substantiation of what I have just said, may I quote from the last annual report of Commissioner Roper, dated Oct. 15, 1918. In the concluding sentences, he said:

> An open-minded attitude must be maintained in the interpretation of the law, in the framing of regulations, and in the application of the law and the regulations to particular circumstances. Administration must be even-handed and impartial; the objective must always be to secure resolutely the full observance of the law, protecting equally the interest of the taxpayer and the Government. * * * With the conviction that this policy will be met halfway by the people, the Bureau approaches with

confidence the task of administering the pending revenue bill, which contemplates the collection during the current fiscal year of double the amount of taxes collected during the fiscal year 1918.

In further proof that this policy is actually being carried out in the intended spirit, you will be interested to know of a general order to revenue agents recently issued by Mr. Callan, head of the income-tax administration, which directs that no rating of employees for purposes of promotion shall be based on amount of additional taxes they recommend but solely on capacity, efficiency, and dependability.

Finally, coming closer to the matter of our own interests, I trust you will permit me to express the opinion that the recent establishment, in the Bureau of Internal Revenue, of the Sub-division of Natural Resources, whose business it is to give expert recommendation and to take final action in the formulation of guiding principles and in the actual settlement of questions relating to taxation of the mineral deposits, the timber lands, and other natural resources, is in itself an index of the conceptions and the motives of the Bureau. And when I tell you that effort has been made to bring into that division able and experienced engineers into whose hands the solution of these problems shall be placed without reservation or qualification, save such as the law imposes, and that every man in the organization from top to bottom is engaged and retained without the slightest influence of political expediency, I believe you will agree with me that there is much justification for believing that the general ideals or axioms I mentioned at the very outset are to be made the aim of the tax administration and that so far as the special matter of mine taxation is concerned, there is a good reason for hoping that this admittedly difficult subject may be handled sensibly, fairly, and conclusively. Specific instances may continue, though I trust with constantly decreasing frequency, to come to your attention which may seem to contradict what I have endeavored to establish. I beg of you to believe, however, that every earnest effort is being made to eradicate and abandon all those things which have caused confusion, exasperation, or alarm to the honest and well-intentioned taxpayer.

Assistance and Coöperation of the Industry Needed

With this part of the story, which is really introductory, I have occupied so much of your time for the reason that to bear this message from the inside is a foremost object, perhaps indeed the principal object, of this paper. It is, moreoever, a necessary preliminary to the successful attainment of the second object, which is the securing of your coöperation and your help in solving a job which is as much yours as the Government's and which is no more to its advantage than to yours to have solved correctly. For notwithstanding every reasonable effort which the

Government may make, the job that confronts us is so big and difficult, has so many ramifications and complications, that I believe the Government cannot master it single-handed, but must look to the specialized groups of taxpayers—in the case of mines, to the mine owners, mining engineers and geologists—for the advice and for the help with which to put the deal through.

The Government is in a frame of mind to trust you high-grade and reputable engineers, to trust not only your technical talents, but your motives and integrity; and you are invited and indeed urged to contribute, for the sake of the general welfare of the industry, such help as you can, by correspondence, by conferences with the Federal officials, and by discussion among yourselves and associates.

Any such policy of coöperation as this obviously necessitates that you shall be taken into the confidence of the Government, that you shall know what it is trying to do in order that you may suggest improvements and short cuts, may point out and correct mistakes, in a word, may have an effective and influential hand in the administration of the revenue laws as applied to your industry.

In what follows, therefore, while leaving numerous features quite untouched, I shall try to indicate, in merest outline, some of the principal problems involved, though most of you no doubt have by necessity become familiar with them through your professional connections. I shall also try to sketch briefly some of the methods of attack upon these problems which the Bureau is considering in a tentative way. These problems and these tentative methods constitute the target toward which it is hoped that you and others like you may direct your criticism and your help.

Fundamentals of Mining Economics

Risk

Since the subject of mine taxation cannot be approached intelligently without a thorough understanding of mining economics, we may well consider briefly some of the economic principles and factors that we all realize apply to the mining business.

The applications and relations of capital to the mining industry involve different principles and are on a different basis from those in other forms of enterprise or investment. Two essential factors serve to distinguish the economics of mining from the economics of other industries: first, the much greater risk generally involved; and, second, the fact that the life of the enterprise is fixed by conditions over which those engaged in it have but little control. If the hazards of mining could be brought down to the average level of other industrial undertakings and if the life of mines could be maintained indefinitely by application of a good average degree of intelligence and effort, mining would take its place on the same plane with the manufacture of clothing or the growing of wheat or the running of a hotel. As a matter of fact, however, neither of these things can be brought about; in consequence, it is necessary to consider mining as something different from a business in the ordinary sense and in a class apart from most other undertakings. The mine operator not only shares the risks common to other types of enterprise, such as the uncertainties of labor and the fluctuations of market; he must in addition bear certain unique risks peculiar alone to mining and equaled in magnitude in few if any other industries.

In ordinary business, investment of capital results in the acquisition of something of inherent value which, in case of failure of the enterprise, can be disposed of at a price that will go part, or all, of the way toward refunding the capital invested. For example, suppose a company engages in the shoe business or the milk business and finds, after due trial, that its costs of production are such as not to afford commensurate profit on the capital invested. In short, it fails. It can dispose of its tangible holdings, which represent the largest portion of its investment—its leather and machinery and plant, in the case of the shoe business, or its cows and its lands, in the case of the milk business—to successors who hope by better methods to turn failure into success. Even if a considerable loss is so involved, something at least is saved.

Hazards of Ore Supply.—But in mining, the raw material represents the chief investment and the raw material is the ore. In general, the success of the mining enterprise depends primarily on what nature has put into the ground. Over that no man has control and, ordinarily, no one may know what or how much until he spends much money to find out. And if one company's operations demonstrate that, in a given tract of ground, nature was not sufficiently generous to give man a profit, it would be difficult to persuade any one else to purchase the property—the value is next to nothing. For a mass of ore in the ground is not like leather or machinery or cows; it has no inherent and intrinsic value; its only value lies in its capacity to produce profit when mined and treated—if it cannot do that, it has no value whatsoever. In such an instance as I have outlined, therefore, every dollar added by the company in development work makes the situation worse and actually reduces the value of the property by gradually eliminating the possibility that a better state of affairs may be disclosed. In short, each addition to capital account has the paradoxical effect of lessening capital value.

The great majority of mining ventures are of the kind just indicated. They fail to return investment plus a reasonable earning upon it. Most of them fail even to make a pretense of returning the original outlay. It has been said[1] that on the Comstock lode, out of several thousand locations into nearly every one of which more or less money and effort were put, less than fifty found sufficient ore to justify extensive development and still fewer paid dividends. Yet the gross product of bullion from the comparatively few active mines was enormous.

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[1] R. H. Stretch: "Prospecting, Locating and Valuing Mines," 20. N. Y., 1909, McGraw-Hill Book Co.

Though the hazards of ore supply are probably greatest in the prospect and early development stages, they are by no means absent in the later stages of a mine's history. Even when promising deposits of ore have been found, and perhaps extensively developed, unexpected geological conditions, such as faults, pinches, change in rock, or erratic distribution of values, far too often turn hoped for success into dismal failure. The Alaska Gastineau is a recent conspicuous example among many.

In short, ore supply, whether to convert a prospect into a mine, or to assure the continued operation of a going mine, is in the majority of instances a most fickle and unreliable thing and involves a degree of risk far beyond that of most industries.

Hazards of Operation.—Nor are the risks peculiar to mining limited to the question as to whether or not a given piece of ground will be found productive and worthy of the establishment upon it of an active operating mine, or whether the mine so established will continue to find profitable ore. The history of mining is literally filled with examples of important and temporarily profitable mines which gradually or suddenly come to grief in consequence of some of the accidents that constitute the other special hazards that peculiarly attach to the mining business.

Finlay, who must be regarded as a conservative in the matter of mining investments, referred on the opening page of his well-known book,[2] to the Treadwell mine in Alaska as one of the really stable and valuable investment securities. In 1909, such a characterization was abundantly justified. Yet in 1917, the chief value of the largest part of the Treadwell mine, if it had any value whatever, was as an aquarium for salt-water fish. In 1905, the Atlantic mine in Michigan had the distinction of earning profit on the lowest grade of copper ore that had ever been treated, namely, but slightly over one-half of 1 per cent. yield. Yet, the very conditions which had made that enviable record possible, namely, the even and continuous mineralization of the lode with consequently high proportion of extraction to waste or pillars, was the very cause which in the following year led to the entire wrecking of the mine and the mine plant by caving, so that today, in spite of efforts made in the interval, the value of the property is essentially nil. Coal mines, flourishing today, are wrecked tomorrow by explosions of dust or gas. Mines in Pennsylvania, in Butte, in Jerome, in Shasta County have been on fire for years. Many a mine with good ore still in it has had to be abandoned, perhaps after years of successful operations, because water at last gained the upper hand and now stands high in the shaft. Every mine faces one or another or several of these and other catastrophies. In some respects, the greater the mine, the greater the menace.

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[2] "Cost of Mining." N. Y., 1909, McGraw-Hill Book Co.

It may be urged that accidents to mines are often due to bad management and might have been avoided by men of proper experience and ability. To this contention the reply may be made that in mining, as in everything else, we determine what is sound practice only by contrast with what proves to be faulty and unsound; we make progress through avoiding previous mistakes; we do not dread the fire until we have been burned. Moreover, in mining, especially, experience accumulates slowly, partly because a long time is required for working out a large mine, and partly because the variations in conditions from mine to mine are so great that the conclusions indicated in one instance may prove (perhaps only after costly trial) to be quite inapplicable to another. But even if the contention of bad management be accepted, it is nevertheless true that in few other industries must so heavy a penalty for mismanagement be risked as the utter wrecking of the entire enterprise.

Unusual Profits Represent Incentive and Insurance.—Clearly enough if the industry is to survive at all, the abnormally high risk in mining must be offset by abnormally high returns in those instances that turn out profitably. These views of extent of risk and compensation for it are not confined to mining men, who might naturally be regarded as biased in their judgments. They have long been recognized and expounded by professional economists who have no more prejudice in favor of mining than of any other industry or occupation. The case has been put by them like this. The majority of mining ventures are failures. A multitude of disappointments is relieved by occasional success. Were it not for the chance of great rewards, all the vitally necessary but mainly unremunerative work of exploration would not be undertaken. In contrast with nearly all other fields of endeavor, it is altogether unlikely that, on the whole, the gains in the successful mining ventures suffice to offset the losses in the unsuccessful. Under such conditions, a high return to the few fortunate ventures does not constitute a true surplus, and must be accepted without prejudice as a necessary and legitimate stimulus to efforts that inure to the benefit of society.

The argument is sometimes advanced that the great profits made by some individual mining companies should, in large part, be taken through taxation by the Government so that the people as a whole may participate in the benefits that the nation's soil and rocks afford. If such policy should be followed, however, there would go with the collection of these profits by the Government the moral and economic obligation to perform all that vast ratio of profitless exploration on most if not all of which depends continued discovery of profitable mines. It seems highly improbable that the Government would pursue such discouraging work with the vigor and persistence that characterizes individual effort when stimulated by the unfailing hope of hitting upon a bonanza. Yet if this obligation should be shirked, mining would certainly languish. The farmer who eats up his entire year's crop of grain without providing seed for the next year's crop will never repeat the operation.

Moreover, none of the great hazards peculiar to mining can be insured against. The manufacturer's plant can be insured against the elements, his stocks of raw material, his goods in process can likewise be covered by insurance. The orange grower can insure against frost or hail, the banker against defalcation. But up to date, no company exists which will write insurance that protects the mine owner against the unexpected exhaustion of his orebodies or against calamities which may bring upon the mine a termination of its operations. The only possible insurance against these risks in mining is comprehended in a correspondingly greater return upon capital which shall be sufficient to induce the operator to assume the risks and which in the long run will go toward balancing the actual losses. Mill, the economist, says:[3]

> The gross profit of capital may be distinguished into three parts which are respectively the remuneration for risk, for trouble, and for the capital itself, and may be termed insurance, wages of superintendence, and interest. After making compensation for risk, that is after covering the average losses to which capital is exposed either by the general circumstances of society or by hazards of the particular employment, there remains a surplus, which partly goes to repay the owner of the capital for his abstinence, and partly the employer of it for his time and trouble.

In other words, by those who invest in the mining industry it must be recognized that compensation for hazard is to be provided for, at least in a mental way, before the true net profit of the enterprise is revealed. Clearly enough, the greater the hazard, the greater is the portion of gross profit required to balance it.

Increase in Mine Values Not Unearned Increment.—One of the effects of the hazards of mining is shown in the prices paid for prospects and partly developed mines.

There is a disposition among many people to regard as dangerous and vicious that type of property appreciation which is often designated by the term unearned increment. This view finds frequent reflection in a variety of ways in the tax laws of various countries, states, or communities. Without entry into a discussion of the ethics or economics of this question, it may be admitted that there are certain justifications for penalizing, through the medium of taxes, those who bought land in the past and now, without any risk or effort on their part in the meantime, reap a disproportionate benefit from either its earning power or its sale. Since such people take for themselves a surplus which has come into existence only through the presence or the activities of the community or of society in general, it may truly be said that this appreciation is unearned by the beneficiary.

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[3] "Principles of Political Economy," **2,** Chap. 23, 153. Colonial Press Ed., N. Y., 1899.

In order that the contrast with increase in mine values may be clearly disclosed, let us set up two imaginary cases. First, let us assume that A bought a large corner lot in a country town 30 years ago for \$1000.[4] If that price was reasonable and fair at that time, the buyer incurred practically no risk. No one could run away with his lot, it could not burn up or cave in. Land values the civilized world over, and especially in this country, where the increase in population had been so persistent and so great, had been steadily increasing and would doubtless so continue; only in exceptional localities were declines registered and these probably for the most part but temporary. All A had to do was to sit tight. It was through no effort or design of his that at some later date a railroad was constructed through the town, that the locality was found advantageous for the establishment of certain important industries, and that in consequence of the great increase in population A now sells his corner lot, in identically the same condition as at first, for \$100,000 as a site for a department store. One might say that the \$99,000 surplus was a result of A's foresight; just as likely, however, his retention of the original purchase was a consequence of inertia or of outright good luck. But in any event, he gained the profit virtually without effort or risk.

Now let us turn to the case of B who bought, likewise 30 years ago for \$1000, a tract of what he hoped and believed would prove to be mineral bearing land. What would happen to him if he, like A, simply held on and did nothing? Even though profitable mines might develop all around him, and thus possibly cause an appreciation in the value of this tract, such appreciation would be only speculative. And as a matter of fact, few mines increase in value to any material extent through the non-activity of their owners. What B would almost certainly do would be, after paying \$1000 for the land, to spend a lot more money, plus much effort and time and anxiety, in sinking or tunneling to find ore. Labor, machinery, explosives, timber, building of houses and roads—these and other things have to be provided for, and isolation, privation, and hardship perhaps endured. A sum equal to the original \$1000 investment makes hardly a beginning; \$10,000 melts away before he knows where it has gone—provided he does not "go broke" before he gets that far. An encouraging development here, a flattering indication there, keep him going until his optimism outlasts his pocketbook and his ability to borrow—then he has to stop. Or perhaps, being endowed with a more conservative enthusiasm and greater resources he decides, after what he deems fair trial, that the property will not develop into a profitable mine. Under such circumstances he is more likely to have to abandon it outright or dispose of it for a pittance than to be able to sell it for anything reasonably approaching the stake he has put into it. With some remaining resources and with confidence probably undiminished (for only a confirmed optimist enters such a game) he buys another undeveloped tract for $1000 and begins the cycle all over again.

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[4] Suggested by discussion of W. O. Hotchkiss.

It may be a matter of opinion how many times such a cycle has to be repeated before a property is found that can be converted into a real mine; that is, a commercially profitable source of metal or other mineral product. But there can be no doubt that on the average the number is very great. The large exploration companies are offered literally hundreds of properties in various stages of development for each one that they take up. I am informed by a strong American mining company operating in the Andes, where the opportunities for mine finding are supposed to be better than in this country, that they investigated 143 properties and found only one that was good enough to buy.

It really makes no difference whether B has sufficient capital to survive through the many failures to the time when one of the properties makes good, or whether he reaches the end of his resources and withdraws from the game, defeated, to be succeeded by some one else. The essential fact is that a successful mine is built on a long string of previous unavoidable failures. Of course, there may be individual exceptions. A newcomer may draw a prize on his very first venture, but the next man may draw twice as many blanks as the general average. Of course it is the average that counts; and on the average the chances that a given prospect will become a profitable mine are only one to scores and scores—say 1 to 50, 1 to 100, or what you will, but certainly something of that magnitude for mines of the precious and semi-precious metals. For iron mines the ratio averages somewhat lower, and for coal mines of course, lower still, but in all cases the principle is the same.

Let us assume that B persists until he finds a real mine, which then he sells for $100,000. Is his $99,000 increase on the same footing as that gained by A with the corner lot? Obviously not. First because B had to spend much in development work, perhaps $20,000 or $50,000, to reveal the $100,000 value; and, second, because to find this one mine he had sunk $1000 plus the $20,000 or $50,000 time upon time in previous prospects that at the outset may have looked just as good as the final successful one, but proved to be failures. Yet each failure contributed to the final issue by yielding experience at least.

A's surplus of $99,000 was not earned; no surplus existed inherently in the property when he bought it, since his original purchase price paid fully for all the property eventually proved to be or to contain; no surplus was created by his efforts. On the other hand, the $99,000 received by B may represent no true surplus or profit at all—instead an actual deficit may be involved. But by whatever name B's $99,000 may be designated, there can be no doubt but that it is on a different footing from the gain realized by A. For the $100,000 received by B is to pay for ore that was always present in the property, that was always a part of the property. B's efforts and expenditures have merely revealed the true value that the property had at the beginning.

Of course it is wasted breath to tell such things to mining men. But since these are facts which others should know, it may be permissible to clinch them with one or two analogies familiar to everyone.

Suppose a new battleship goes to sea to try her big guns at target practice. The explosive and projectile for each shot cost $250. Assume the Navy average for vessels of that class is nine shots to one hit, and that our particular ship sustains that average. Here on the deck lay nine projectiles; one will hit the target, yet no one can tell which one, for they all look alike. The nine shots are fired; one hits; that one represents an expense of $250, yet it really cost $2250 to hit the target once. A prize of $1500, offered by the company that built the ship, if the crew would equal the Navy average of one hit in nine shots, might seem to yield a handsome return if one thinks only of the single shot that reaches the mark; but in reality, the prize, instead of affording a surplus, leaves a deficit.

Or, to take another example, a man in the egg business maintains his flock of hens by yearly additions from incubation. He buys fertile eggs, secures a hatch of 80 per cent., raises say 50 per cent. of these through the perils and vicissitudes of infancy, discards the half that prove to be cockerels and winds up with one effective pullet for each five eggs purchased. Plainly enough the one pullet came from only one egg, yet there was no escape from wasting the four others to secure the one pullet. There were no distinctive markings on the one egg that would finally yield the mature hen; it looked just like the other four. If pullets were worth $1 each, and disregarding the cost of incubation and raising, he could not pay more than 20 cents each for the eggs for incubation.

It is the same with mines. An individual or a company that buys a prospect or a tract of ground which later develops into a profitable mine obviously can pay for it at the outset only such fraction of its eventual value as is represented by its chance of making good, that is, by its risk factor. The complement of this fraction is represented by the investments of the same owner or of others sunk in those prospects which failed to make good. These failures, like the four eggs, are an inseparable and absolutely necessary part of the transaction, they must be paid for and they must be charged against those few which turn out successfully.

As a final illustration, the prospect or little-developed mine may be compared with a herd of cattle on a range. The owner may estimate that he has about 2000 head. A prospective buyer rides over the range, concludes, likewise, that there are probably about 2000 cattle in the herd; the two men compare estimates, find that they agree, and effect a sale on that basis. If later, on actual count, it develops that there were 2100 or 1900 cattle in the herd, must the new owner refund the excess in the one case, or can he collect for the deficit in the other? Of course not. He bought the herd, so the herd is his. Likewise, when B bought the mineral land for $1000, his deed gave him title to all the ore the land contained, irrespective of whether or not its quantity and grade were fully evident at the time. When, by development of the property, the actual count is taken, it simply measures and establishes the true value of what he bought at first.

Exhaustible Capital Value

It has also long been recognized by economists as well as by mine owners and mining engineers, though not by everyone, that mining is an industry of wasting assets and that the apparent income for a given year's operations is not true profit but part profit and part return of capital. The miner who sells his metal or his coal is not only selling his product, he is selling at the same time a part of his mine. The price he receives per pound or per ton must cover both. Failure to appreciate fully this situation has led to illogical and foolish action by many people whether mine operators or investors in the shares of mining companies. It is to be hoped that the general attention to this subject which the present heavy tax burdens are going to force, will greatly clarify the general understanding of it and will lead to a more sane and satisfactory basis for the conduct of the mining industry.

The entire conception, which some have seemed to feel is a complex and mysterious affair, becomes of utmost simplicity when the ore deposit is regarded merely as a great supply of raw material, the cost of which, along with the other costs, must inevitably be covered by the selling price of the finished product. Provision for this is made under the term "depletion," which, recognizing that ore, when once extracted from a mine, can never be replaced, aims to compensate for the inroads into the value of the mine thus occasioned by affording the money equivalent of the value, in the ground, of the ore removed. Thus at any time, value of ore remaining plus the depletion fund accumulated to date should equal a constant, which represents the value in the mine of all the ore originally estimated to be present. As is well known and will be discussed on a later page, the value of ore in the ground is the present value of the profit which that ore will yield when it shall be mined and treated. The true measure of depletion, therefore, is a function of the profit concealed in the ore. The subject of depletion is so well understood by professional mining engineers that further elaboration here would be quite redundant.

Mining Must Be a Continuing Industry

In emphasizing the idea of wasting assets, Finlay has stated that a mine is exactly like an account in a bank whose business is being wound up by a receiver. Hoover, Smyth, and others have implied, likewise, that the business of a mining company is to work out its mine as rapidly as it reasonably can. Even more fundamental, however, than the conceptions which underlie these statements and in a sense contradictory to them, is a principle which is becoming more obvious and important each year, namely, that the business of a mining company should be the mining business, in just the same sense that the business of a railroad company is the railroad business.

What I mean is this: The companies which are producing the largest part of the country's output of most of the metals have made enormous outlays in time, effort, intelligence, and money to build up organizations of great size and remarkable efficiency, which cover all steps from the ore in the ground to the finished metal. In large measure, it is because of the perfection and the efficiency of organizations like these, some of them great, others on the way to become great, that the world is buying its metal supplies at prices not greater than those which now prevail. Probably in even larger measure is the country's dominance in mineral production due as much to these highly perfected organizations as it is to the natural mineral wealth of the nation. This is demonstrated by the fact that Mexico, Chile, Peru, Russia, and other countries richly endowed with mineral resources made little utilization of them, when measured by modern standards, until American capital, enterprise and skill, singly or in combination—in a word, American ability for organization—took hold. To assume or to require that each of these great organizations must be torn down and thrown away when the particular lot of raw material, namely, the mine on which that particular company is operating, happens to be used up, not only would be rank folly, but would constitute an economic waste as direct and as deeply injurious to the public welfare as waste in the mining of coal, or in the efficient utilization of water power, or any of the other national extravagances or deficiencies which we are striving to overcome and eradicate. Evidences are constantly multiplying that large mining corporations realize that they must regard themselves as continuing organizations. They are providing, or attempting to provide, themselves with new supplies of raw material, that is, new mines to be worked up as soon as or before their present supplies give out. Since such a policy is sound and in accordance with public good, it must be both permitted and encouraged.

The policy of continuing enterprise is taken for granted in most industries. The shoe manufacturer, for instance, buys a six or twelve months' supply of leather. Out of the gross proceeds derived from working up this leather, he puts aside, as a matter of course, a sufficient amount to enable him, when that supply is exhausted, to purchase the next lot. Simply because the supply of raw material for a mining enterprise must of necessity be purchased in very large quantities which often suffice for many years, it is no less logical and fair that the mining companies ought, and should be permitted, to set aside out of the gross income from operations upon that lot of ore, a fund with which to purchase the next great lot. The only essential difference between the two illustrations is that in the case of the shoe manufacturer he can be practically certain that when his present supply is exhausted he can buy as much as he needs of a new supply without any doubt or risk and without any serious effort to locate it and at not very great difference in price. But the mining company, to replace its old mine, may have to spend much time and effort and anxiety in finding another and may have to pay far more per unit for the new one than it paid for its former supply which was purchased, perhaps, 10, 20 or 30 years ago.

Although timber comes under the head of wasting assets, the analogy with mineral deposits quickly ceases, for any given lot of timber increases in value through growth (less fire loss, which can be minimized by fire protection measures) and there is always the possibility of reforestation, either natural or artificial. Mineral deposits, however, are of fixed and determined magnitude (in so far as concerns human utilization) and when once gone are gone forever.

Perhaps the closest analogy to the wasting assets of a mine is afforded by a farm which, after having been worked for a number of years without fertilization, begins to lose fertility through impoverishment of the phosphate and potash of its soil. Artificial fertilizer is then applied to replenish or replace those exhausted elements and the farm resumes its former productivity. Assume that the land originally cost $1 per acre. That figure has absolutely nothing to do with determining the cost of fertilization now, which may be $1 or $2 per acre. Then, due to increasing demand for potash and phosphate for fertilizer use, the cost may mount to, say, $5 per acre. But this may be justified by the increased yield. Surely, no sensible farmer would fail to charge the item of fertilizer expense into current cost of production of his crops, and no one could criticize his logic and his right in so doing. What he would be doing is providing for replacement of exhausting mineral constituents of the ground.

Then why should the miner not make similar financial provision for replacement of his great, long-time purchase of ore, his mine, which he is in process of exhausting. In short, I believe that before long mining companies will set up a fund, not merely to return initial capital, not merely to refund the original cost of the current great stock of raw material purchased years ago, but a replacement fund—a fund which will truly fulfil the spirit of the sinking fund idea by making sure that the company constantly and fully maintains its power. This replacement fund ought to be large enough to include an insurance factor to cover the extra risk involved in assuring to the company its new supply of raw material.

If the average life of a successful mine is say 30 years, and if the cost of finding and securing such a mine doubles every 30 years, which is, I believe, a conservative assumption, it is easy to see how the policy of refunding only original investment will inevitably carry a company from a position of independence and power into a position of insignificance and impotency.

Moreover, the establishment and retention of a replacement fund by the active and reputable mining companies is the best remedy for what, in my opinion, is the present unsound and improper policy of doing business on borrowed money. As the tendency toward Federal regulation of private enterprise continues and grows, as it almost inevitably must, I believe industries will be obliged to support themselves mainly out of their own revenues, instead of invoking that overworked and, to my mind, eventually fallacious notion of depending so largely on credit.

It must not be concluded, from the fact that in the last 50 or 60 years many great mine discoveries have been made (California, Australia, Transvaal for gold, Michigan, Montana, Arizona, Chile, for copper, etc.) that such discoveries may be counted on to continue either at a growing rate to keep pace with general growth in population, industry, consumption of staples, etc., or even at a fixed rate. For these reasons:

1. Mining, although an old occupation, is in its modern developments essentially a new industry, just as the varied applications of steam and electrical power are new industries, and for the reason that all industrial development and advance started on a wholly new curve of progress during the 19th century. Although it is dangerous to set any limitations to the capacity of human intelligence, it nevertheless seems fair to assume that the advance in scientific, industrial and general material progress in the last few scores of years, say in the last 100 years, and particularly in the last 40 or 50 years, has produced a break or jump in the curve, not only greater than at any previous time, but greater than will again be achieved. It is unnecessary to enlarge upon this. I believe brief reflection will convince any one of the probable soundness of the assumption.

The result was no doubt emphasized and hastened by the exploration and settlement of a new continent inordinately rich in ores and other natural resources, coming at this critical time in industrial development. Coinciding also as it did with a period of revolutionary advances in transportation and communication, the effect was to stimulate prospecting in all corners of the world, and mining engineering became the most roving of professions.

What may be termed, then, the acceleration in understanding of material matters and in incentive to progress has attained a maximum in the past few decades, and, in consideration of the span of human history, has attained that maximum with almost unbelievable abruptness. Indeed, one may almost say that all of a sudden, civilization turned its attention to industrialism, involving the use of metals and minerals and, therefore, involving the search for metal and mineral deposits. No such sudden addition of attention to mine hunting can again be expected. Consequently, no future epoch of mine finding can hope to equal that of the three-score period of years beginning with 1849.

2. In many respects the saying that "the early bird catches the worm" is true for mine finding. Imagine 100 gold dollars scattered promiscuously through a large haystack. Someone who realizes the value of a dollar happens along, sees one of the gold pieces practically at the surface of the stack and is naturally impelled to search for more. In a few minutes, he may find as many as ten of the gold pieces which were near enough to the surface either to gleam through the thin covering above them or to be revealed by a relatively slight scratching of the surface. To find the next ten, however, may require hours of patient and diligent lifting and sifting and search. And the last ten may elude discovery for what is relatively a long, long time.

What is true for bright gleaming gold pieces in a pile of hay 10 or 20 ft. in diameter is, plainly enough, immeasurably more true of generally dull, inconspicuous deposits of ore buried to depths of possibly hundreds or thousands of feet in solid rock, and scattered through a mountain range or a continent. In short, most of the mining regions now known were relatively easy to find. Moreover, in view of the intensity of prospecting to which near and remote corners of the world have been subjected, it is safe to assume that of the total number of orebodies easy to find, most have already been found. In any event, the finding of new orebodies is becoming more and more difficult. The time has passed when the old-time prospector, generally unenlightened and superstitious (though not to be despised nor his accomplishments belittled) can, with the assurance of a grubstake, be relied upon to replace the mines that are being exhausted at such a rate as present enormous scales of production imply. Otherwise, there would not have come into existence in the last 10 or 15 years the great exploration companies who are endeavoring to utilize every available means for mine finding; yet even these, notwithstanding their resources, are by no means meeting with the degree of success that could be hoped for. In a comparatively short time, the question of finding the new supplies of ores to meet the world's needs is going to become pressing, if not acute. In a word, the next great purchases of raw material for the mining companies are going to cost far more than the supplies they are now using up.

Prolonged failure to take account of these underlying factors of risk, exhaustible assets, and necessity for continuing enterprise in the mining industry must inevitably force into existence one or the other of these outcomes, either that mining will languish because it cannot be accomplished with profit, or that the business of mining will be continually passing into new and inexpert hands, who, because of their lack of experience, optimistically rush in where the experienced companies found it impossible longer to tread. Either of these consequences would be most unfortunate and serious.

Tax Laws as Applied to Mines

Importance of Mining Industry

It is of course unnecessary before an audience of mining engineers to emphasize the importance of the place held by the mining industry of this country, whether measured in money value of output, in relation to the country's other sources of wealth, or in terms of international power which possession of such vitally important resources has afforded the nation. But merely to fix in our minds in some approximately quantitative way the magnitude of the mineral industries of the country, the following facts may be cited:

In 1917, the latest year for which data are available, the value of the mineral production of the country, as computed by the United States Geological Survey after eliminating duplications, amounted to $5,011,000,000. This represents probably a larger portion of the estimated gross national income of $68,000,000,000[5] for that year than was contributed by any other single industry save the railroads. Indeed the only reason why the income of the railroads somewhat exceeded that of the mineral industry is because, as the Director of the United States Geological Survey has recently pointed out,[6] the mines and smelters, oil wells and refineries, quarries and cement mills furnish to the railroads more than 4,000,000 tons of freight a day, or nearly twice as much as all other freight combined. The dominant or highly important position of the United States as a producer of most of the common and fundamental mineral products is too well known to require even tabulation. As for the effect of this, we need only contrast the present position of this country with that which it would occupy if it possessed only world-average quantities of coal and oil, iron, copper, and other vital metals—if it were in the same condition with respect to coal, iron, and copper that it is in regard to tin.

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[5] *The Annalist* (Jan. 6, 1919).

[6] "The Strategy of Minerals," 321. New York, 1919.

The 1917 mineral output is divided in round numbers as follows:

		PER CENT.
Principal metals	$2,059,000,000	41.1
Coal	1,523,000,000	30.4
Oil and gas	703,000,000	14.0
Miscellaneous	726,000,000	14.5
Total	$5,011,000,000	100.0

Thus metal and coal mining account for 71.5 per cent. of the total mineral production of the country.

Magnitude of the Tax on Mines

As would be expected from the gross income of the mineral industry, the portion of the total Federal taxes which this composite industry bears is very high indeed. Comparative figures are set forth below for the calendar years 1916 and 1917. The latter, approximate as yet, have been kindly supplied by Edward White, head of the Statistical Division of the Income Tax Unit. All figures are given in round millions of dollars. In the case of mines and oil and gas wells, the figures taken directly from the returns of the companies themselves may reasonably be regarded in the aggregate as minimum figures.

TAXES PAID BY MINERAL INDUSTRIES

Official Designation	1916		1917	
	Amount	Per Cent. of Total Corporation Tax	Amount	Per Cent. of Total Corporation Tax
Extraction of minerals	$16,000,000	9.2	$207,000,000	9.5
Smelters and blast furnaces	10,000,000	5.8	126,000,000	5.8
Petroleum, refined	3,000,000	2.0	24,000,000	1.1
Stone, clay, and glass products	2,000,000	1.0	16,000,000	0.7
Total	$31,000,000	18.0	$373,000,000	17.1

TAXES PAID BY METAL MANUFACTURING INDUSTRIES

	1916		1917	
	Amount	Per Cent. of Total Corporation Tax	Amount	Per Cent. of Total Corporation Tax
Iron, steel and non-ferrous metals	$23,000,000	13.5	$393,000,000	18.0

TAXES PAID BY ALL CORPORATIONS

Industry or Group*	1916		1917	
	Amount	Per Cent. of Total	Amount	Per Cent. of Total
Mineral and metal industries, as above	$54,000,000	31.5	$766,000,000	35.1
Business and finance	20,000,000	11.4	444,000,000	20.4
Agricultural products	26,000,000	15.0	345,000,000	15.7
Chemical industries	10,000,000	6.0	123,000,000	5.7
Transportation	22,000,000	12.7	99,000,000	4.5
Public utilities	8,000,000	4.9	48,000,000	2.2
All other corporations	32,000,000	18.5	359,000,000	16.4
Total corporation tax	$172,000,000	100.0	$2,184,000,000	100.0

* The grouping in this last table is my own, but I believe it fairly represents the relations between the several industries.

Outline of Tax Laws, 1909–1918

Although the Federal tax laws of recent years, as they apply to mines, have not taken into account in any direct way the excessive risks involved in mining, they have lately given some recognition to the second difference that characterizes mining, namely, that its assets are exhaustible. This recognition comes under the head now designated in the laws as "depletion." It may be advisable in this connection to outline very briefly the revenue laws of the past 10 years, touching on each only in its particular application to the mineral industry.

Although various taxes were levied by the federal government on mining companies and on mineral production during the Civil War, the first federal tax in recent years to affect the mining industry was the corporation excise tax of 1909, which was, to all intents and purposes, an income tax and was given its particular name simply to escape possible unconstitutionality of an income tax pure and simple. It levied a tax of 1 per cent. on the net income above $5000. This law provided, as a deduction from gross income to arrive at taxable net income, "a reasonable allowance for depreciation of property, if any," but did not specifically refer to depletion of mineral deposits. Many mining companies, however, in making their returns under this law, made claims for depletion. The Bureau of Internal Revenue allowed these claims, wholly or in part, until 1913 when the Supreme Court, in a much-discussed decision, held that depreciation as used in the law does not apply to exhaustion of mineral deposits and that, therefore, no deduction on account of such exhaustion could be allowed.

In 1913, however, after a special constitutional amendment, finally ratified on March 1 of that year, had validated the principle of taxing incomes, the 1909 excise act was superseded by a new law. This levied a 1 per cent. tax on net income, which should be computed after deducting, among other things, "a reasonable allowance for the exhaustion, wear and tear of property, arising out of its use or employment in the business, not to exceed, in the case of mines, 5 per cent. of the gross value at the mine of the output for the year for which the computation is made." Thus the principles of both depreciation of physical property and depletion of mineral deposits were put upon a firm legal basis.

The 1913 revenue law was superseded, in 1916, by one which levied 2 per cent. on the net income of corporations and provided tax-exempt deductions as follows: "a reasonable allowance for the exhaustion, wear and tear of property arising out of its use or employment in the business or trade." In the matter of depletion, oil and gas wells were for the first time specifically distinguished from mines. The oil and gas wells were granted, "a reasonable allowance for actual reduction in flow and production," while to mines was given, "a reasonable allowance for depletion thereof not to exceed the market value in the mine of the product thereof which has been mined and sold during the year for which the return and computation are made."

The 1917 income tax law applied a 6-per cent. tax on corporation incomes with exactly the same terms as to depreciation and depletion as did the 1916 law, but to meet the heavy additional expenses of war, there was also placed upon corporations a war excess-profits tax which levied from 20 to 60 per cent. on portions of the net income as determined in relation to invested capital. There was also inserted in the income-tax law of 1917 a provision that an additional 10 to 15 per cent. be levied on that portion of the year's income undistributed 6 months after the end of the year, except such parts of that surplus as are actually invested and employed in the business or are retained for employment in the reasonable requirements of the business or are invested in Liberty Bonds.

Finally, the revenue law of 1918 levies a 12-per cent. tax on corporation incomes for the year 1918 and 10 per cent. for subsequent years. In this law, depreciation is covered by "a reasonable allowance for the exhaustion, wear and tear of property used in the trade or business, including a reasonable allowance for obsolescence," while with respect to depletion, the law provides,

> In the case of mines, oil and gas wells, other natural deposits, and timber, a reasonable allowance for depletion and for depreciation of improvements, according to the peculiar conditions in each case, based upon cost including cost of development not otherwise deducted; provided, That in the case of such properties acquired prior to Mar. 1, 1913, the fair market value of the property (or the taxpayer's interest

therein) on that date shall be taken in lieu of cost up to that date; provided further, That in the case of mines, oil and gas wells, discovered by the taxpayer, on or after Mar. 1, 1913, and not acquired as the result of purchase of a proven tract or lease, where the fair market value of the property is materially disproportionate to the cost, the depletion allowance shall be based upon the fair market value of the property at the date of the discovery, or within 30 days thereafter.

There was also imposed a war-profits and excess-profits tax which provides for the collection on various fractions of the income of amounts that range in 1918 up to as much as 80 per cent. of certain portions, and up to 40 per cent. for subsequent years.

Summed up, therefore, these several laws have provided for taxation of income at steadily increasing rates from 1 per cent. up to 12 per cent. for 1918. They have levied heavy excess-profits tax in 1917 and still heavier in 1918. Both income and excess-profits taxes are continued indefinitely for years subsequent to 1918, though at rates somewhat lower than for 1918. Allowance for depreciation of physical property has been provided for in the income tax of each of these years and has been extended to cover obsolescence beginning with 1918. Depletion has been treated in a different way with nearly every law. Allowances for depletion, specifically unauthorized by the first law, were granted until pronounced illegal by the Supreme Court. Then, in 1913, a 5 per cent. allowance on the gross value of the year's product at the mine was granted; in 1916 and 1917, an allowance not exceeding the market value of the year's product in the mine; and, in 1918, a reasonable allowance for depletion according to the peculiar conditions in each case, based generally upon market value as of March 1, 1913, but allowing for revaluation in case actual discovery subsequent to that date shall have materially increased the value.

On the whole, therefore, increasing taxes have been accompanied by increasing reasonableness and liberality as to tax-free deductions for depreciation and depletion, though of course the increase in the allowances has been of no such magnitude as the great increase in taxes.

Administration of Tax Laws

Survey of Situation

It is clear that whether the existing laws be reasonable or senseless, whether they be fair or unjust, whether they be mild or oppressive, the activities of the Bureau of Internal Revenue are necessarily confined to the limitations that the laws impose. If the laws are found to be bad, it is to be hoped that their shortcomings may be so clearly pointed out that Congress will see fit to remedy them for the future; but unless relief, if found to be needed, shall subsequently be given and made retroactive, which seems unlikely, the procedure which must be followed for the present is already outlined in the laws.

The country being at present committed to tax upon incomes, we are freed or prevented, as one may care to regard it, from considering whether the modern idea of determining tax by ability to pay rather than by amount of service rendered to the taxpayer by the Government is ethical and sound; and from trying to decide what methods of taxation are fairest. In the matter of mine taxation for example, we cannot be materially concerned with policies for taxing idle mineral lands or unprofitable mines, since neither class earns income. All we have to do, all we can do, in the present connection is to take the laws as they are and try to apply them fairly.

The regulations which according to the laws are to be promulgated by the Commissioner of Internal Revenue for the interpretation and application of the tax laws have been written for each of the laws through 1918. But in their case, opportunity for revision and improvements is not excluded, for the desire and intention of the present Commissioner are, while adhering to the limits set by the law, to administer the tax program in the light of facts and equity. It is for the purpose of assisting in the formulations of rules and the making of decisions relating to minerals and related products that there has been gathered together a group of engineers to constitute the recently established Sub-division of Natural Resources of the Income Tax Unit. These engineers are, therefore, to assist in revising the regulations relating to natural resources products for the years back through 1916, wherever necessary to bring them into accord with the true intent of the law and with the present policy of the Bureau.

Obviously enough, the main job of this Natural Resources Sub-division is one of valuation. Initial value and the decline in values resulting from time, wear, and exhaustion, fix the depreciation and depletion deductions; valuation is the essence of the test of discovery; actual value is the one true measure of invested capital.

The handling of the timber problems has been begun and is being carried out under very auspicious circumstances with a most cordial and effective relationship established between the Government and the forest industries. The oil and gas program, which has been under way for something like a year, is well advanced toward completion. The metal- and coal-mining problems, however, have been but slightly touched as yet, in so far as concerns settlement of those features that depend on valuation, for the determination of the taxes in 1917 and for the review of the taxes for 1916, although for the preparation of the questionnaire that has been sent to all mining companies and for some other preliminary work, the Bureau had for some months the assistance of R. C. Allen, whose experience with mine taxation in Michigan was a most valuable asset.

It is the plan to have 1917 and earlier years out of the way before undertaking 1918 cases, but the settlement of valuation principles and the rules for their application will be made once and for all as soon as possible; and when these objects shall have been accomplished and the 1917 cases settled, it is hoped that operations thereafter may become in considerable measure a matter of routine.

Point of View

Equity and Finality.—Reverting to a topic already mentioned, may I outline again, this time a little more specifically, the point of view which it seems to me should guide a proper application of taxation laws to the mines.

The two fundamental objects to be achieved are, first, impartial balance as between different taxpayers in a given industry and as between one industry and another and, second, finality of the decisions. If proper balance is attained, finality will be assured. If, on the other hand, the decisions are not sensible and just, there is likely to be no end of revision and unsettling of results, so, after all, balance is the prime requisite. The Bureau of Internal Revenue will endeavor to act in the joint capacity of trustee for the taxpayer and for the Government and will seek to harmonize the interest of each. It will try to be equally as eager and prompt in refunding tax that may have been collected in excess of what is properly due as in collecting all that ought to be paid.

The latest law clearly recognizes the wasting industries. The mining industry is thus among the few to which distinctive treatment of any kind is applied. The idea has gained hold in the minds of some engaged in other industries that mines are thus accorded special favors or advantages. This view, however, is entirely erroneous. For while certain allowances are made because of the exhaustible character of mining assets, they are only such as fairly provide for including cost of raw material in total cost of production, and serve only to remove what, in their absence, would be a harsh and unfair discrimination against mines and other wasting industries. Moreover, not all the consequences of the exhaustible character of mines are provided for in the law, as, for example, the establishment of a replacement fund. Neither is special consideration afforded to cover the extra hazards involved in mining. For example, in the average business, the largest part of the investment is protected by insurance, and the expense of that insurance is properly charged into working costs. The largest part of the mining company's investment, however, is represented by the ore in the ground; the great and varied risks that attend the extraction of that ore cannot be insured against. The only insurance protection is afforded by the increased return to the investor. Yet high rate of interest return results in the computation of low valuation, hence low depletion allowance, and consequently, high tax. And then instead of deducting from income his heavy insurance expense, much heavier than in most other industries, the miner pays tax on it as if it were true profit. Furthermore, no recognition is given of the difference between unearned appreciation in land values and the increase in mine values revealed by development. Nor are the extreme fluctuations of income from year to year, which particularly affect the mining industry in consequence mainly of its hazardous character, afforded any equalization in the law, which regards the calendar year as the independent unit of computation. On the whole, therefore, it would appear that no valid objection can be raised to the law as in any sense specifically favoring the mining industry.

Normal Industry Should not be Upset.—Perhaps the requirement next in importance to equity and finality is that every effort be made to avoid setting up regulations and procedures that will furnish strong incentive to the mining company, in the hope of lessening its tax burden, to adopt practices or pursue policies contrary or foreign to the dictates of good sense and the well-founded customs of the industry. Such a policy would mean bad tax-collecting, bad mining, and therefore, in reality, bad morals. I do not refer to deliberate evasions of a culpable kind, but to acts, which, though legitimate in the sense of not being illegal, are wrong because artificial, indirect, and probably subversive of the public good.

As an example, there readily comes to mind the case of many ore deposits in limestone where, because of the commonly irregular distribution of the orebodies and the heavy character of the ground when the altered rock is exposed to the air, it is customary to carry development work not far ahead of current extraction. In the matter of valuation on the basis of exposed or proved ore such mines are clearly at a disadvantage with extensively developed mines like the porphyry coppers. If, to overcome this disadvantage, one of these limestone mines should greatly expand its underground development and should thereby discover a proportionate amount of new ore, it might gain a higher valuation and thus, through greater depletion deduction, pay a lower tax. But the money thus saved to it (and lost to the Government) would be partly offset, perhaps eventually more than offset, by long-time interest charges on the expense of the idle development and by the likelihood that when finally needed for extraction purposes, the workings would require heavy repairs or be useless altogether because of caving and sloughing during the long interval of disuse.[7]

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[7] A case opposite in detail though similar in its bad consequences is the well-known restriction of development in parts of the Lake Superior iron region as a result of applying State property taxes to valuations based on exposed ore.

Neither should there be temptation, through complicated schemes perhaps not yet discovered, to sell mines which ordinarily would not be sold. No matter how bona fide and complete the change of ownership might be, the policy would be wrong, for in general a mine is worth most to those who have become familiar with it and experienced in mastering the particular problems it presents; it should, therefore, properly remain in the hands of those to whom it is worth most.

Falling in somewhat the same category is the conviction in my mind that, in view of the comparative recency of the income tax as a matter of prime consequence, and of the prevailing uncertainty as yet regarding what the Government desires and requires, it will be improper to refuse allowances, to which a company would otherwise be entitled, simply because its books are not kept in a specified way.

Methods of Mine Valuation

As already pointed out, the latest revenue law provides three ways of arriving at the value of a mining property: (1) The cost, if purchased on March 1, 1913, or later. (2) The fair market value as of March 1, 1913, if purchased earlier. (3) Revaluation in either of the other cases, provided since March 1, 1913, or since purchase subsequent to that date, new discovery in unproved ground results in a value disproportionate to the value prior to discovery.

Since most mining properties now yielding profit and, consequently, concerned in the income-tax program have not changed ownership since March 1, 1913, we are confronted mainly with the problem of establishing the value of the mine on that date, and perhaps in a considerable number of cases with the clause relating to discovery since that date.

Regulations of the Bureau of Internal Revenue for earlier years have outlined the various ways in which fair market value as of March 1, 1913, may be reached. Most of these methods, however, are methods of indirection. The method or methods which I trust may be followed in the present attempt to apply the tax laws to the mines should be methods as direct and professional as possible, *i.e.*, engineering methods.

We have progressed beyond those primitive modes of exchange when beef was sold by the "side," wheat by the "bin," or metal by the "chunk." Nowadays, we insist upon knowing and take deliberate pains to find out how much beef or wheat or metal, and what quality, before we complete a transaction. Likewise, ore is not sold at so much "per mine," nor are mines generally sold at so much per block of ore. Instead, here again, elaborate effort is expended on ascertaining as accurately as possible how much ore is present, of what character and grade it is, and above all, how much profit it can be expected to yield, since in that lies its value.

While valuations gaged by transfers of nearby or essentially similar properties may be reached with some reliability in certain cases of oil and coal lands, since these in recent years have been subject to rather frequent changes of ownership, it is doubtful if methods of that sort will, except in occasional instances, yield any safe basis for valuation of metal-mining properties. Stock-market quotations may, under some circumstances, afford a rough check on valuations arrived at by more direct and reliable methods, but in themselves can ordinarily be given little weight, for a variety of reasons too obvious to require enumeration. And the par value of security issues is likely in most instances to be far too fictitious to be accorded serious attention.

Present Value of Eventual Earnings Method.—The pronouncements of economists and the dictates of common sense leave no doubt whatever that the true measure of value of fixed property like farms, city lots, or mines is a capitalization of their income. Lands afford prospect of yielding a practically perpetual income. In their case, therefore, the method of capitalization is a simple division of yearly income by the income rate. Mines, however, as they are of diminishing value in proportion as they yield return, must be valued in a different way, namely, by estimating the total expected earnings and the total life and from these working back to such a valuation as will afford a demanded annual rate to cover profit and risk and will be itself returned at the end of the estimated life by a sinking-fund accumulated from annual depletion increments. This present value of eventual earnings method is the method which has been practised for scores of years and within the last decade illuminated particularly by the writings of Hoover[8] and Finlay.[9]. So far as I am aware, it is the only definite method of mine valuation expounded in textbooks on mining or in treatises on mining investments. Of necessity, this present-value method of mine valuation must be the corner-stone on which the work of the Natural Resources Sub-division of the Bureau of Internal Revenue must build. The fear implied in some of the earlier regulations of the Bureau that such a method of valuation would permit true income to escape taxation is devoid of economic foundation and is as groundless in the case of mines as it would be of a corner lot on Broadway. Indeed, the provision for profit is the vital feature of the method, and that profit, like any other, is subject to tax.

Mines vary so greatly in character and conditions that no single basis of valuation can be applied indiscriminately to all without grave error in the case of many. This is true, not only as between coal mines on one hand and gold mines on the other, but as between different coal mines to some extent and as between different gold mines or different copper mines to a greater extent. Either each mine must be considered in unlimited individual detail or else classes must be established into which mines of essentially similar character and circumstances shall be grouped and these classes handled as if individuals themselves. The former alternative, if feasible, might give the fairer result, though I am not confident that best balance would be secured thereby. The latter scheme, however, is likely to afford the best practical solution because the single-mine method would require a degree of detail and an amount of time which would render the problem almost hopeless. This does not mean, however, that due consideration of special circumstances in any case will be denied.

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[8] "Principles of Mining," N. Y., 1909, Hill Pub. Co.

[9] "Cost of Mining," New York, 1909.

Unquestionably the fairest and most defensible method of mine valuation is the present-value method, *provided the essential factors that enter into the computation are reasonably well known or can be arrived at with fair reliability.* Most coal mines, many iron mines, the porphyry coppers, some of the Lake copper mines, and perhaps a few others can be valued directly and satisfactorily by this method because more of the necessary information is available for them than for any others. For them, any other method is likely to be less reliable and less fair. Obviously, also, valuations so determined for such mines will be the most reliable group of valuations of all that will be established. Therefore, it seems to me logical to value by the present-value method those mines to which it will directly apply, in order to establish certain standards, or yardsticks, by which to determine valuation of those mines for which less and less of the necessary data is available.

Let me illustrate why this method of comparing with a standard is desirable and use, by way of illustration, examples from the copper industry with which I am most familiar. A given porphyry-copper deposit may be producing 100,000,000 lb. a year at a gross profit of 6 cents a pound and may have an indicated life at that rate of 30 years and a past history of successful production behind it of 8 or 10 years. A limestone mine may likewise be producing 100,000,000 lb. of copper per year at the same gross profit of 6 cents a pound; it may have a successful producing record behind it of 30 or 40 years, yet have actually developed ahead only, say, 3 years' ore supply. Is the porphyry mine to be valued on the 30-year life and the limestone mine on the 3-year basis? Obviously not. How, then, is the limestone mine to be valued? Will not its true value be best arrived at by applying certain properly determined factors to the value indicated for the porphyry mine—factors that will take into account its greater risk in ore continuance or life persistence and that will make needed provision for such other changes or differences as experience has shown are likely to arise between limestone mines on one hand and porphyry mines on the other?

If the conclusions to which these arguments point are sound, the present-value method will be the very keystone of the valuation section's work and a series of modifying factors must be established which will bring other mines into proper balance with those to which the present-value method can be applied directly and without modification.

Right here lies the very keynote and crux of the entire valuation job. If a series of factors, or indices, can be established that are practical, effective, and fair, the problem is going to be simplified immensely. But such an outcome will be quite impossible of attainment except with the closest possible coöperation and assistance of professional engineers and geologists, who collectively possess the vast fund of specific information which such a series of factors must comprehend and codify. By all means, therefore, let the mining companies contribute, through their technical staffs, all the help that they possibly can. The Revenue Bureau hopes to settle this and related matters with the help of conferences to which it trusts those who can assist will come and give all possible help.

If the present-value method is to be extended to the great majority of cases of valuation, as just indicated, there arises a multiplied obligation that it be applied in an absolutely logical and proper way, and with all reasonable exactness. Inasmuch as the method, at its very starting point, relies on factors, such as tonnage and yield of ores, which are necessarily but estimates and approximations at best, it might seem natural to conclude that all subsequent steps in the process could, with entire permissibility, be of only corresponding accuracy—that attempt to introduce greater refinements than those imposed at the outset would be futile and absurd. That is the conventional theory of error. Perhaps it is justifiable and sound when the object is the attainment of a proper average, for the probability, or so-called law, of averages will ordinarily balance individual low results with corresponding high ones. But in dealing with this tax problem, we cannot be content with a fair average. We cannot satisfy A's complaint by proving to him that "B was let off correspondingly easy, so the thing evens up after all." The essential and reasonable accuracy of each individual result is important.

If inaccurate methods are applied all the way through, some cases will inevitably arise in which the sign or effect of the error is the same in most or all of the steps, and the magnitude of the total error thus accumulated may be large—far too large to be knowingly permitted.

The simple logic of the case, then, seems to be this: If estimates that are necessarily low in accuracy have to be used in the early steps, the obligation is all the greater in the subsequent steps, to confine the total error, so far as reasonably possible, to that thus introduced at the start—in a word, not to deliberately permit inaccuracies to enter into the result. When, to a considerable extent, the accuracy of many of the steps in the computation can be determined by choice of method, it is proper to choose those methods that give the most accurate results—unless the working out and application of such methods is too costly in time and effort to be justifiable. There must obviously be a balance struck between degree of refinement and rate of accomplishment. But to choose sloppy methods because some other steps must be elastic is, in this particular instance, illogical and unpermissible.

While it is unnecessary to defend before this audience the present-value method of valuation, it may be well, in view of the responsibility which it may have to bear, to analyze with care what its consequences will be if used in the taxation program, and to indicate some of the considerations that, in my personal opinion, ought to be given appropriate attention in such an application.

The valuation engineers of the Bureau, as I understand it, are to answer the question of valuation that is comprehended in the imaginary situation of a prospective buyer, competent to measure mine values and actuated by the hope of profit to be derived solely from mine operation, making an offer for a mine to the owner who is likewise competent to measure mine values and who is under no obligation to sell except such obligation as arises from his belief that the offer made is advantageous to him.

The first question that such a wise prospective buyer would want to determine is the security of his capital. He realizes that mining is a hazardous and uninsurable business. Therefore, since in all forms of investment risk is reflected in the interest rate, he demands a higher return on his money than he could obtain if he invested it in an ordinary manufacturing business or in a good mortgage. Since the interest rate must thus include both actual profit and compensation for risk, the determination of the rate cannot be based on the mining industry in general, as compared with other industries, when the degree of hazard varies greatly among different classes of mines; different rates must be established for different classes. On these determinations, as well as of proper interest to be credited on the sinking funds, much thought may well be expended.

Let[10] us assume that our prospective buyer concludes, in a given case under consideration, that he should have 10 per cent. on his money each year, in addition to a depletion installment calculated eventually to return his capital sum. As a matter of fact, regardless of whether he spends it outright or invests it in something else or places it, as theoretically intended, in a sinking fund for the positive redemption of capital, his annual depletion installment, if actually received, continually reduces his stake remaining invested in the risky mining enterprise; yet if everything goes well and his original assumptions on the basis of which the valuation was reached prove to be justified, he will be receiving a 10 per cent. return on his entire initial capital through the last year when only a small fraction of that capital is invested in the 10 per cent. risk and the major part has been used in some other way. In reality, therefore, he receives on the average distinctly more than 10 per cent. on the capital that is at risk.[11] The question now arises whether this greater interest return, concealed in the apparent flat rate of 10 per cent., is actually required to offset hazard, or whether on the other hand, an even 10 per cent. on the decreasing sum actually at risk is enough, in which case, the present value of the mine would increase.

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[10] The following paragraphs on the relation of interest to capital at risk are the result of discussions with my colleague, Doctor Hance, and may, we hope, be elaborated further at another time.

It seems highly probable that in the average mining operation, the greater return concealed within the flat statement of rate is not too much to compensate for hazard, and also that the valuation computed on the flat return of say 10 per cent., as by present methods, represents a safer and fairer basis on which to buy and sell a mine than the higher valuation that would arise if the return were limited to an exact 10 per cent. on such parts of the capital as remain in the hazardous enterprise.

Since the money still in the enterprise derives greater and greater interest return with time, it could, therefore, afford to carry greater risk during the late than in the early period of the investment. Clearly enough, in a going mine purchased on the basis of proved and prospective ore, the early years would indeed involve less risk, while, as viewed from the beginning, the later years, necessarily relying on ore less positively assured, would entail greater uncertainty and hazard. The method and the facts thus appear to coincide in direction, and although it may be doubted if the risk mounts at the very end at so rapid a rate as the percentage of return is found to do, or if, on the other hand, the risk of the late middle years is sufficiently covered, it seems likely that no more accurate coincidence can be established save by complex formulas, which would be quite unjustified. Acceptance of the flat rate has behind it, furthermore, the support of accepted custom. Still another advantage is that, because of the rapidly increasing spread in the later years of life between, say, an 8 per cent., a 10 per cent., and a 12 per cent. flat rate of return, justification is afforded for employing, in the computations of present value, rates of interest that do not differ very greatly and yet make provision for a wide range of risk, as for example, a porphyry-copper mine and a pockety silver mine.

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[11] On the assumption that money in an ordinary, good investment is entitled to 6 per cent., it is found that in the case of a property of 10-yr. life, valued to give 10 per cent. on initial sum and amortized at 4 per cent. compounded annually, the ratio of total profit assignable to amount remaining at risk in the property, divided by the total of the amounts at risk during each of the 10 yr., works out to be not 10 per cent. but 13.25 per cent. Similarly, for a 10-yr. life, an apparent 8 per cent. return is found to give an average of 9.71 per cent.; and a 12 per cent. return to average in reality 16.79 per cent. on the money at risk. In the three cases, 8 per cent., 10 per cent., and 12 per cent., these actual average returns are respectively 21.4 per cent., 32.5 per cent., and 40 per cent. higher than the flat rate to which they are related.

Since this paper was sent to the printer, my attention has been called to prior recognition of such disparities, by W. W. Whitton and D. B. Morkill, who wrote on "Formulas for Mine Valuation" in *Min. & Sci. Pr.*, May 18, 1918, and Aug. 31, 1918, respectively. Neither one, however, views the situation exactly as implied above.

Another phase of the present-value method which appears not to have been clearly set forth in any of the discussions thus far devoted to this subject is that all the assumptions on which this method has been based and all the tables that have been printed as aids in applying it arise from the idea of equal annuities and imply that a given property will mine a fixed tonnage of constant average grade of ore each year throughout its history. All the computations as to life and all the interest factors involved rest on such assumption. As a matter of fact, these assumptions may apply to certain classes of property. They probably do pretty well apply to operations upon a given seam of coal tributary to a developed industrial district. It was to such a condition, I believe, that this fundamental method of valuation was first applied, namely, the English collieries.

But the assumptions of constant grade of ore and a fixed tonnage per year surely do not apply to many other mines in which this method could otherwise be advantageously used. Many of the mines like the porphyry-coppers, which have great reserves of ore established by exploration, will through choice, and must indeed of necessity, work richer ores during their early years and depend on leaner and leaner ores as their life progresses. As the richer ores yield not only more metal per ton but also more profit per pound or per ounce of metal, this mining of the richer ores first will tend in a double way to bring realization of eventual earnings into the interest market at an earlier date and will increase the present value of the property. Likewise, such mines, possessing enormous ore reserves, are likely to continue in future, as they have done in the past, to increase their scale of operations and their rate of output so that the actual life of the mine will be shorter than is indicated by any present rate of exhaustion. In this way, also, the realization of the eventual profits will come at an earlier date and the present value of the mine will be enhanced.

Perhaps some justification or support should be given for the conclusions that have just been stated. First as to the hypothesis of declining grade of ore with time. Almost every influence combines to cause mining of a great deposit to begin on about the richest ore it contains. Some of these influences are:

(*a*) Most deposits of the class embraced in the long-life-ahead mines decrease in value outward and downward. The decline in value with depth is especially striking in those deposits affected by the geological process of secondary enrichment, but downward decrease is by no means confined to these.

(*b*) Most such deposits are explored first in their upper central portions, *i.e.*, where the ore is best and where surface indications were probably most attractive, and only with time are the explorations pushed to the lateral and bottom limits of the deposit.

(*c*) It is natural that actual operation, *i.e.*, ore extraction, shall begin approximately at the point where exploration began. Indeed, commonly, extraction has already been going on there before the marginal portions of the deposit are completely developed. It takes time to extend the mining operations laterally outward to the margins, where the lower-grade ore generally lies, and in most instances it takes time to extend operations to the bottom of the deposit, where likewise the grade is lower.

(*d*) In several instances already passed into history, the figure regarded as the basic economic limit of ore grade a few years ago has now been so lowered by improvements in methods, processes, and experience that what was waste then or of only prospective future value is profitable ore and is being mined now. This tendency will unquestionably persist.

(*e*) Regardless and independent of improvements implied in (*d*) there will always be a tendency, perhaps more or less unconscious, to make sure now of at least the better grades of ore and to go more slowly with the lowest grades, in order that still further improvements may be developed, which will simplify the problem, or that the difficult job may be passed to someone else.

The foregoing considerations, which are of general character, are amply confirmed by experience in case after case. The same conclusion is indicated by the general changes in ore grade for the country as a whole. For example, definite statistics as to average yield of copper from all copper ores produced in the United States were first compiled by the United States Geological Survey in 1906 and are now available through 1916. These show that the tonnage of copper ore mined has increased from 18,000,000 in 1906 to almost 58,000,000 in 1916, and that in the 11-year interval the yield of copper from this ore has declined from 2.5 to 1.7 per cent. If the great tonnage of low-grade native copper ore from Michigan is excluded, because it has stood nearly stationary in percentage yield (declining in the period only from 1.25 to 1.08 per cent.) the remaining copper ores have fallen from a yield of 3.56 per cent., in 1906, to 1.86 per cent. in 1916, a drop of almost one-half. Changes of similar nature are indicated for the other principal metals.

Turning now to the second question, that of increasing rate of production, here again we have abundant evidence, both general and specific, most of which need not be entered into here. It may be pointed out, however, that if declining grade of ore must be handled, as I have just endeavored to establish, that disadvantage generally implies the necessity of so increasing the output as to bring down unit cost to a proper level. An influence of probably even greater importance lies in the fact that in the 21 years between 1896 and 1917 (the only span for which reliable statistics are available) the copper consumption of the world has increased from 766,000,000 to about 3,150,000,000 lb., or at an annual rate of slightly over 7 per cent.—that is to say, the consumption in 1896, if compounded annually at 7 per cent., would give the consumption for 1917. This increasing demand has caused increasing production. There being no prospect of decrease in the rate of consumption growth, production must likewise expand in the future. This growth in production will come, in part, from newly discovered mines, but beyond question much of it will come from those mines now existing which have great deposits developed and to which there is strong financial incentive, or what may be called economic compulsion, to produce and sell their product as rapidly as the world's market will allow. As a matter of fact, for the 36 years back through 1882, when reliable copper statistics were first compiled and when the American copper industry and the applications of electricity were both just becoming important, the output of copper in the United States has grown at a rate just about equal to an annual compounding at 9 per cent.,[12] that is, from 90,000,000 to 1,908,000,000 pounds.

The importance of these factors of declining grade of ore and increasing rate of production becomes very real in the case of mines of assured long life like the porphyry coppers. Let us suppose that a given mine has developed an immense tonnage as follows:

	Tons		Per Cent.
	20,000,000	expected to yield	2.00
	40,000,000	expected to yield	1.75
	60,000,000	expected to yield	1.50
	80,000,000	expected to yield	1.25
	100,000,000	expected to yield	1.00
Total	300,000,000	of average yield	1.33

Let us suppose further that this deposit is now being mined at the rate of 5,000,000 tons per year, indicating a life at that rate of 60 years. Assuming an 8 per cent. return on the purchase price and redemption of capital by a sinking fund compounded annually at 4 per cent., the indicated present purchase price of $80 worth of eventual total earnings is $15.84. Such would be the present value indicated by the tables of Inwood, Hoskold, and others.

But if we assume that the richer ore is mined first and the lowest-grade ore mined last, the indicated present value becomes $19.77 or an increase of 25 per cent. If, in addition, we assume that the present rate of output of 5,000,000 tons a year will be increased each year, say, by 4 per cent., which indicates a doubling of mill capacity in about 18 years, and a tripling of capacity in about 28 years (surely a conservative assumption for porphyry deposits), the life of the mine changes from 60 years to about 31½ years. The present value, or indicated purchase price, then becomes about $22.40 by increased capacity alone, or about $24.75 by combination of increasing capacity and declining grade, or respectively 41 per cent. and 56 per cent. greater than the value indicated by the tables now in general use. Still further increase in present value would be afforded if we considered the greater profit per pound yielded by the higher-grade than by the lower-grade ore.

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[12] The disparity between this figure and the smaller rate of the world's consumption is, of course, not due to accumulation of stocks but to the fact that all other producing countries have not fully contributed their share to the total increase.

These factors, it seems to me, will have to be given serious consideration and statistics will have to be compiled to show how mines of various classes are likely to change their rates of production and how the ore that they mine is likely to change in grade. Then certain groups will have to be established to which a scale of factors indicated by these past-history statistics must be applied.

In arriving at fair market value as of March 1, 1913, the Bureau of Internal Revenue has hitherto followed the policy of taking into consideration only the facts that were known on that date or that could have been then known had one endeavored to learn them at that time. The question may, therefore, arise as to whether it is fair to arrive at a 1913 valuation by applying to the standard present-value method of mine valuation modifications which have not been proposed until 1919. In my own opinion, there is sufficient justification for using those modifications that are consequences of taking into account changing grade of ore and changing rate of production. The justification is this: In 1913, and since that time, there was no actual method followed for valuing mines for purpose of sale, for the reason that mines of the size to which these newly proposed factors would apply are rarely, if ever, sold. In arriving at a market value as of 1913, therefore, we are dealing with a wholly hypothetical question. If there had been occasion to value such mines for sale in 1913, it is unbelievable that factors so important as the change in grade of ore and the likely change in rate of production would have been ignored. In other words, if a market value had been required at that time, it would have been reached in accordance with the methods now proposed.

In arriving at valuations by the present-value method, certain of the data must originate with the mining company, namely, tonnage and grade of ore, percentage of extraction, and cost and rate of production, for these various factors either are direct consequences of the nature of the mineralized ground that the company owns or are results of the policy and methods that the company chooses to apply. Of course they will all be subject to check by the engineers of the valuation section of the Bureau. There is one necessary factor, however, which must come from other sources than the company concerned, namely, the probable selling price of its product over the period of indicated life of the property, for the reason that the selling price is something over which the individual producing company does not have more than slight and indirect control. That the selling price is a factor of utmost consequence in the computation of present value has been emphasized by previous writers; therefore careful and detailed research must be put upon this subject when so vital a job as valuing all the profit-making mines of the country is to be faced. It will not do simply to sit back and say the best index of the future is what has happened in the past and then make a flat arithmetical average of prices of the past 10, 20 or 30 years and assume that the same flat average will persist for a corresponding period to come, especially if the average of the past is made up of a low price at the beginning of the period and a higher price at the end, or vice versa. In such cases, the trend rather than the average is important, yet it must be ascertained whether the trend is in itself but part of a minor fluctuation that is a component of a greater trend.

In endeavoring to reach an idea as to the price that copper may be expected to bring for the next 10 or 20 years from 1913, I have prepared a sort of syllabus that might contribute information toward the answering of that question and will sketch it only partly and in merest outline to indicate the extent to which it is possible to go with any of the mineral products, provided such degree of refinement proves justifiable.

Aids in Determining Future Price of Copper

1. General methods of economists in predicting future prices of commodities, and their predictions.
2. Price trend of principal commodities for say 20 years prior to 1914.
3. Price trend of principal metals for say 20 years prior to 1914.
4. Price trend of copper for say 20 years prior to 1914.
5. Copper-price predictions of others.
6. Details of copper-price fluctuation.
7. Curves showing relations of copper production and consumption to price, for the United States and for the world.
8. Copper consumption per capita in principal consuming countries during past years.
9. Ratio of bulk to price for copper is such that transportation factor is subordinate. Therefore, production of any locality competes in the world's copper market. Contrast with coal, oil, and iron which have a higher bulk to value ratio and, therefore, more localized realms of competition; and on the other hand, with precious metals having a much lower bulk-value ratio and, therefore, affected more by whim and caprice than by the steadier effects of industrial or economic balance.
10. Probable future cost of copper production.
11. Substitutes for copper.
12. Growth in accumulation of old copper.
13. New uses for copper.
14. Increase or decrease in known uses of copper.

Each of these items is capable of considerable elaboration and expansion, as may be illustrated, for example, below in the case of No. 10, "Probable future cost of copper production." This matter of cost of production is so intimately bound up with the problem of available copper supply as to make it a question which is the more fundamental. But in any event, it is clear that the price at which copper must sell in the future must yield a profit above the marginal cost of production for the minimum quantity required.

10. *Probable Future Cost of Copper Production.*
    - (*a*) Sources of supply.
        - (1) Deposits now known and worked. Statistics of reserves and of exhaustion up to date.
        - (2) Deposits now known but not worked. Remote deposits; red-bed type.
        - (3) Deposits not yet discovered. The outlook in various regions; expenses of exploration; advances in geological methods and skill; availability; transportation.
    - (*b*) Effect of increasing depth. Statistics.
    - (*c*) Recovery from tailings, waste dumps, pillars, fillings, etc.
    - (*d*) New or improved methods or processes.
        - (1) Mining.
        - (2) Treatment: leaching in heaps or in place.
    - (*e*) Utilization of byproducts.
      Potash? Sulfur, sulfuric acid, fertilizers.
    - (*f*) Wages.
    - (*g*) Other costs: fuel, power, timber, supplies, etc.
    - (*h*) Average spread in past years between cost of production and selling price.
      Constancy or fluctuation? Trend?

A matter on which all possible illumination is desired from mining engineers and geologists is the proper handling of probable and prospective ore in the computations of tonnage. As Hoover has so aptly said of this matter of prospective value, no engineer can approach it with optimism, yet the mining industry would be non-existent today were it approached with pessimism. My own feeling in this connection is that the decisions reached must arise from and be in accord with a thorough understanding of ore deposition and occurrence rather than rest on any arbitrary or conventional limitations and rules.

Discovery.—The clause in the latest tax law that permits revaluation of property if discoveries made by the taxpayer have caused a real change in value, is clearly going to be difficult of application. Without endeavoring at this time to indicate either the present attitude of the Bureau on this subject or my own opinions regarding it, I wish to emphasize that suggestions from the mining industry as to the application of this discovery clause will be very welcome in Washington and will undoubtedly aid in the solution of this difficult and important question. Particular attention may well be directed to these questions: What is meant by discovery of a mine, what constitutes discovery according to that meaning, and when is such discovery achieved or completed?

Depletion Rate.—Question has been raised as to how or at what rate depletion is to be allowed, *e.g.*, whether coincident with the exhaustion of the deposit, or at a faster or slower rate. This is another of those matters upon which professional advice will be welcome. In the meantime, I may confess that the plan of allowing depletion in direct proportion to the actual annual impairment of the mine value by extraction of the year's ore seems to me most equitable and probably simplest of application. In this connection, it may be pointed out that if such coincident rate of depletion is adopted, the actual depletion allowances should be computed not on number of tons of ore, assumed to possess average grade, mined during the year, nor on weight of metal actually produced, but on the value in the ground of the ore extracted during the year.[13]

Depreciation

The remarks thus far made with respect to valuation are related primarily to the relation between mine values and depletion deductions. The matter of proper depreciation allowances on physical property must also be decided. In this connection effort will be made to determine the intent of the law with respect to those three main factors ordinarily comprehended in the inclusive term depreciation, namely, physical wear or decay, obsolescence, and inadequacy. Also, effort will be made to settle on a sound principle of depreciation; that is, to decide as between one of the several methods now current, such as the straight-line method, the reducing-balance method, or some other. Due consideration ought to be given to the fact that the usable life of equipment may outlast the life of the property on which it is to be used and a correspondingly rapid rate of depreciation, therefore, allowed from the start. The idea has been expressed that such an outliving of a mine by its equipment will fairly be cared for by finally charging off the remaining balance of value in the equipment against the final year's income when the property shall be wound up. This theory fails, however, when confronted with the practical fact that in the final year of wind-up, there is ordinarily no income or not sufficient income from which to deduct large items of liquidating depreciation. It will also be recognized that the salvage value of specialized equipment in remote districts may range from low to nil.

Excess-profits Tax

The excess-profits tax raises problems that all of us might well wish to escape, but that nevertheless must be faced and solved. No effort will be made in this paper to go into its details, but I may express the conviction that the assistance which the valuation engineers will afford in arriving at the fair quantities to be taken for invested capital, paid-in surplus, and earned surplus for the mining industry will necessarily rest upon the true conceptions of the nature of investments of capital in mining enterprises and on the sound and accepted economic theory that in this hazardous industry, occasional great gains to lucky or shrewd investors must be accepted with equanimity; a policy of control that is too grasping overreaches itself.[14] For quite outside of any ethical questions that may be involved, this practical situation has to be faced; that taxes which take up to 80 per cent. of net income must be applied with due consideration of the necessity that profit-making industries must survive in order to supply the heavy taxes that will apparently be required for years to come. To whatever extent is proper, also, account should be taken of the fact that the pre-war average of earnings of the mining companies was on a higher percentage level than the earnings of ordinary enterprises because, as emphasized several times in this paper, those earnings include cost of insurance in the one instance but not in the other.

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[13] Suggested by discussions of Paul Armitage and A. D. Brokaw.

Effect on Local Taxation

Quite apart from the direct considerations of the subject of federal taxation of the mines is a matter to which many of the mining companies have already, no doubt, given attention, namely, the effect of the federal tax policy on state, county, and local taxation. Deliberate valuations must, of necessity, be applied by the Government to most of the profit-making mines and other producers of natural resources in the country. To no other group of industries will such official and conspicuous valuations be applied. It may easily follow, therefore, that the true values set on mining property, which are likely to be higher than most previous values for local taxation purposes have been, will result in the mines of those regions where the property tax prevails carrying a disproportionately heavy part of the local taxes, for the reason that all the other property except mines will not be brought up correspondingly to its true market-value level. In securing the benefit to be derived from full and fair valuation of their properties for the federal income and excess-profits taxes, it is to be hoped that the mining companies will not suffer injustices arising nearer home, and it is to be desired that the local tax commissions may be brought to see this matter in the proper light.

DISCUSSION

R. V. Norris, Wilkes-Barre, Pa.—The taxation of coal properties, *i.e.*, their valuation, is not in many respects similar to that of other mining properties. Coal mining is more nearly in the class of manufacturing than is the mining of metals, with the difference of a wasting and irreplaceable raw material. The tonnage available on any property is subject to reasonably accurate estimate; the grade of mineral is constant; the character of the deposit is regular, except as to the Pennsylvania anthracite and some few western coal areas; the extent of deposits is generally known or readily ascertainable; and the mining problem is that of a low-grade material of wide distribution and relatively low cost, but involving the handling of huge tonnages, where the cost of operation rather than the varying value of the mineral is the important factor.

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[14] F. W. Taussig: "Principles of Economics," **2**, 101. N. Y., 1917, Macmillan.

Methods of Valuation.—The author has struck the keynote in insisting on valuations based on the present value of prospective earnings, which is unquestionably the only proper method of valuation of coal properties, *i.e.*, as going operations; but he has not touched on the question of the valuation of coal properties under lease at tonnage royalties, yet a very large portion of the country's coal is mined from so-called leased properties, on a tonnage-royalty basis.

Prior to the income tax law passed in 1919, no recognition was given to any equity to the lessee in such so-called leased properties, but in this law such possible equity was recognized. In the case of a lessor who has a constant tonnage royalty, regardless of the profits or losses of the lessee, the returns can and should be divided into interest on the value of the property and depletion to amortize its value in its probable life; and, as these returns are practically independent of mining profits, the legal rate of 6 per cent. should be used in calculating the value of this property. In the case of term leases, which are rare, the value of the property should be calculated, taking into consideration the probable going royalty of the region at the time of release or renewal, rather than calculating for the entire life of the property, at the royalty of the lease, which will probably be far lower than the new royalty obtainable.

The mining risks, as applied to coal, include, as a very small consideration, doubt as to the continuity of the deposit. This fact is subject to reasonably accurate estimate, so the life of any coal property is determined by the rate of output. This, in turn, dependent on market conditions, is largely under the control of the operator. Hence the "hazard of ore supply" of the author is, in the case of coal, very slight. The hazards of operation in coal mining are gas, water, dust and squeeze. Of these, the greatest hazard is explosion, either gas or dust or both, but these hazards should be taken care of by a "reserve for mining hazards' carried as an operating expense, rather than by an abnormal interest rate used in calculating present value.

Profits.—The profits of coal mining per unit of output are very small, the earnings being dependent not on a high profit but on very large output, with a small unit profit. The 1910 census returns showed less than 2 per cent. return on invested capital for all bituminous-coal operations, not including depreciation, and only 4½ per cent. on anthracite operations under the same conditions.

The author's statement that the capital invested must be returned by reserves for depletion and depreciation is absolutely sound and fair. Such reserves should be sufficient to replace the exhausted coal at the going price at the time of exhaustion, not merely to amortize the cost, or the estimated value March 1, 1913, the date at which the first income tax law happened to be passed.

Future Reserves.—That the coal business is and should be a continuing industry is, of course, unquestionable. The author indicates that this continuation of the invaluable organization must be by the purchase, from time to time, of new mines at the then going price. In the case of the coal industry, this long and indefinite life is more reasonably attained by the control of territory or tonnage not available at the present time, either by reason of its distance from shipping facilities, its location in relation to possible development, or its depth rendering its present exploitation financially impracticable. Such reserves should not be considered as available coal, which in fact they are not, but should be carried as "reserves unavailable." It is a fact that, considering local taxation and carrying charges, coal reserves that cannot be mined within a reasonable time, 40 to 45 years, are of no present value and are not properly subject to depletion charges. They, therefore, should not be considered in calculating the present value of coal property.

There is but small probability of new discoveries of coal of sufficient amount to have any considerable influence on the total known to be available in the United States. Enormous reserves of coal, not now of any economic value, are known to exist, though unavailable by reason of location beyond the probability of transportation within a reasonable time, by reason of depth, and by reason of the quality being too low to allow competition with other known coals available to the same territory.

Such deposits have no present value and if held as future reserves by present operating companies, such holding should be considered as a public benefaction and should not be discouraged by the imposition of taxes. Coal now commercially unavailable held by well-organized operators promises the continuation of a valuable organization to the ultimate benefit of the body politic.

While the author shows that even in 1917 coal comprised over 30 per cent. in value of the total mineral output of the country, he fails to point out that, as a low-grade product, it supplied over 80 per cent. of the total tonnage of all mineral products, and actually provided nearly one-half of the total tonnage transported by the railroads. The proper encouragement of coal production is, beyond all other, vital to the prosperity of our transportation systems and, hence, to our country as a whole.

Questionnaire.—The author refers to the questionnaire sent to all mining companies. May I suggest that this, while probably excellent for metal-mining conditions, is unsuited to coal. It does not speak the "coal language;" many of the terms used are incomprehensible to the coal miner. The requirement of maps showing operations is preposterous. A single company whose operations were recently examined by the writer would be required to furnish a map, on the standard scale of 1 in. equals 100 ft., 36 by 64 ft.; and in the case of the anthracite operators mining a great number of beds, even more extensive maps are in use. There would hardly be available room in the Treasury Department to file the maps that might be furnished, and no living engineer or combination of engineers could properly and scientifically study all of these. The request should be, at the most, for small-scale maps showing the areas worked over, exhausted, and the areas of solid coal remaining.

Interest Return.—The author very properly states that the high rate of interest return results in low valuation and low depletion allowance, hence high tax, and the miner pays tax on his insurance expenses (mining risk) as a profit. Hence it is clear that, as far as coal mining is concerned, a high rate of interest in figuring the present value of a coal-mining property is not warranted.

I would respectfully suggest that the present value of coal properties be calculated on the base of 6 per cent. interest plus the percentage required to amortize the value of the property at its exhaustion; and, further, that the date of exhaustion be taken with extreme conservatism. Experience shows that the last few years of life of a coal property are more likely to produce tonnage than to produce profits. The author states that, in his opinion, there is no valid objection to the law as favoring the mining industry, in the matter of assuming a high rate of interest return (and resulting low valuation). I would suggest that while this is absolutely correct, there is a very valid objection due from the coal-mining industry; such a high rate of interest is an unjustifiable discrimination against it. Six per cent. plus the necessary percentage to amortize the property value is all that is justified in calculating present values in the case of coal mines.

The author states that, in the case of coal land, valuations may be reached with some reliability by sales. As sales of operating and successful properties are rare, this is assuming that the value of such proved, opened, and successfully operated properties may be fairly determined by sales values of adjoining lands, perhaps unproved and unopened. I do not believe that this is sound. The value of an operating property should properly be based, as the author elsewhere states, on its present and probable future earnings, in which earnings, organization and management are a dominant factor, or in the case of properties when such values are not available I believe that calculations based on the going rate of royalties for the region in question will give reasonable though probably too conservative values.

As an example, in an operation once examined by the writer there was found a consolidation of two companies, A with two-thirds the total investment and a losing proposition and B with one-third the investment but making reasonable profits. In the consolidation A took one-third and B two-thirds, and justly, as the consolidation made good money for both. A was a losing proposition only because of the management; the physical conditions of each company were identical. I do not believe that competent and successful management should be penalized by taxation, hence I agree with the author that district averages should control.

The author properly comments on the necessity for accuracy in calculations, as far as such are subject to mathematical treatment; all estimates of value based on future results are necessarily approximations, and it is manifestly improper to further add to the approximation by methods involving unnecessary errors. The author is absolutely correct in urging that no "sloppy" methods be used, and that only such matters, distinctly noted, as are matters of estimate or judgment be treated with other than mathematical accuracy.

Interest Rates.—As has been previously indicated, as regards the coal business a present value based on the legal rate of 6 per cent. interest, plus a depletion allowance, is proper. The interest rates, suggested by inference, of 10 to 12 per cent. are too high for this highly specialized business, which depends for its profit on great output and not on high unit profit.

Such elements of profit as stores, lumber, byproducts, etc., not essentially mining, should not be included in either costs or profits, but the colliery settlements, houses, schools, churches, clubs, parks, amusement halls, and the like are essential elements of the business, and their financial result, be it profit or loss, is properly included in operating expenses.

Referring largely to metal mining, the author states that the early risk is less than the later. This does not generally apply to the coal business, in which the risk in opening a property, proving the coal, and finding a market is largely in the early years of any operation. Once opened, proved, and established in the market, the future of a property, barring labor troubles, is fairly assured, and such establishment of an operation is properly credited in any estimate of value.

Tonnage.—The author's proposition that a given property will insure a fixed tonnage, equal year in and year out, is incorrect as applied to coal mining. The history of practically all mines shows a small tonnage at the beginning of operations, increasing relatively slowly in the early years, a rapid increase when development is completed, a certain life at maximum tonnage, a rapid fall during the obsolescence of the property, and a stoppage of operations when, due to the large area mined, the cost of operation becomes financially prohibitive. Hence, in estimating the present value of coal properties on the basis of present and prospective profits as is most proper, it is erroneous to use an average output to exhaustion. Careful studies should be made of past and future development, the future output should be carefully estimated, and the calculations of present value based on this varying output, with increased costs for smaller than normal tonnage.

Valuation.—In studying present value as of 1913, as indicated by the author, it is not proper to apply directly data of later years. But, in the opinion of the writer, it is proper to study the trend of conditions up to 1913, and to predict future conditions on the basis of past variations. Further the writer believes that, except for war years, war conditions are not a proper base for calculations, but in calculating values as of 1913 he does not hesitate to use the actual results available of years subsequent to that date, using, however, for the future figures based on general conditions up to March, 1913; that is, abnormal profits or losses during the period of the war, are, in the opinion of the writer, properly considered in obtaining values as of March 1, 1913. It seems preposterous to make calculations on data that, to the engineer, are known to be far from the facts, and the writer declines to thus stultify himself; but for future probable costs and profits, war conditions should be eliminated but the general upward trend of costs and prices should be considered.

In determining profits as of the future, Mr. Baer's theorem must be considered; "The price of the entire supply of anything necessary to a community will be regulated by the cost of production of that portion of the necessary supply that is produced at the greatest expense." The price of coal is not fixed by the lowest-cost producers nor by the average cost of production, but by the cost, with a reasonable and minimum profit, of the high-cost necessary coal. This method of price fixing, which is the law of supply and demand, was used generally by the United States Fuel Administration in the so-called "bulk line" principle and also by the Price Fixing Committee of the War Industries Board, on which Committee the writer had the honor of serving.

As a matter of fact, under normal conditions, competition in the coal industry is so keen that only a minimum profit is obtainable for the higher-cost operations (and none for the very high-cost producers); hence increases in price are primarily and practically only due to increases in operating costs incident to labor increases, or to higher costs of necessary supplies. It is believed that, owing to this intense competition, the margin between cost and selling price will remain fairly constant. Hence, a valuation based on such margins averaged on recent years will fairly represent the probable future profit and this forms a reasonable basis for valuations either for sale or purchase, or for the purposes of taxation. This is believed to be reasonable, as the supply of coal in this country, estimated at over 1000 years, is for the present generation practically inexhaustible; hence the output and price is regulated by the demand, not by the supply

The author suggests that depletion is best calculated on a per ton basis. The writer can conceive of no other reasonable method and has recommended depletion on a tonnage basis calculated from the value of the property and the tonnage probably recoverable within a reasonable time, omitting from the calculation both the value (usually negative considering carrying charges and taxation) and estimated tonnage of lands probably containing coal not minable within a reasonable time (40 to 50 years).

Depreciation.—Depreciation charges should be based on the life of the property (obsolescence) when all improvements become of only scrap, or nominal, value; or, in the case of very long-lived properties, on the probable useful life of such improvements. As an example, shafts, slopes, or main openings may be properly considered to have a life equal to the life of the property or bed opened by them, but not to exceed 45 years; local tunnels, the life of the coal tributary; buildings and structures, not to exceed 20 years; machinery 15 years; and animals, based on experience, 5 years. All these maximums must be reduced so as not to exceed the estimated life of the colliery properties.

The author also calls attention to the danger that proper valuations for federal taxation may result in undue and unfair local taxation. In general, the local taxation of coal properties is reasonably fair, in some states as Virginia, and West Virginia, notably so. These states base values on the sliding scale of values, thus:

A. Coal immediately available, taxed at a high valuation = the acreage mined out the preceding year

B. Available coal 10 years; A times 10, about one-half of the A valuation.

C. Available coal deferred, A times 10, at a fairly low valuation.

D. Unavailable coal, all other holdings, at a nominal valuation.

In sharp contradistinction some of the anthracite counties of Pennsylvania are attempting to impose a valuation for taxation purposes based on $500 per foot-acre; or on the best average lands containing 60 ft., of coal, a base of $30,000 per acre. This is in the face of sales rarely exceeding $3000 per acre for the 60 ft. territory. Of course such excessive valuations are being resisted in the courts, and thus far the decisions have very materially reduced this preposterous valuation.

R. D. Patterson,* Dayton, Ohio (written discussion).—The relation of capital to the mining industry differs from its relation to almost any other undertaking. The industry, on account of the many hazards incident to the business, the many conditions over which it has no control, the rapid deterioration of physical plant and equipment, and the constant wasting of its mineral assets, necessarily must show a greater return on investment than that of an industry in which these factors do not prevail. Furthermore, reserves to provide for possible hazards are a proper element of cost and should be treated as such in making our returns to the Treasury Department. This is a non-insurable risk, yet an explosion, fire, or some such hazard which frequently occurs in the mines may wipe out all profit accumulation of former years in an attempt to restore the property to its original condition, or else the capital may be impaired by the operator having to borrow money to put the mine in a workable condition.

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* President, Weyanoke Coal and Coke Co.

CLINTON H. CRANE,* New York, N. Y.†—In order to determine a fair basis for depletion, first, a fair market value of the property as of March 1, 1913, must be determined. Having determined this, the annual sum necessary to return this fair market value to owners at the exhaustion of the mine is a matter of simple arithmetic based on the same factors that are used in determining the market value of the mine. The four basic factors to be determined are: Average market price of product; average costs of product; the life of the mine; the allowable interest rate.

The average market price would be the price that the producer might reasonably expect to receive for the mine's product during the life of the mine. In determining this average price, the history and average market price over a period of years past must be considered. Such a price, which we may call the normal price, has generally been used in all estimates in valuing a mine. For instance, the average price of copper for 30 years prior to 1915 was 14.06 cents, the average price of tin was 27.63 cents, the average price of lead was 4.25 cents, the average price of common spelter was 5.36 cents.

The average costs would vary for each individual mine. It would be the average cost for producing a given unit of output in normal times. This average cost might be per pound of copper, per ton of ore, or per ton of concentrates, depending on its application to value as to whether the metal itself was marketed or whether the ore or concentrates were marketed. It could be determined with reasonable accuracy for any year by dividing the annual profit by the total number of tons or pounds produced and subtracting this profit per ton or per pound from the average market price per pound of that year. This would make no allowance for interest on investment nor for operations other than mining that might be carried on by the company in question, but such refinements of cost could be readily allowed for and justified by each individual company doing a more complex business. For the purposes of this memorandum refinements in regard to interest on capital investment, and on more or less development work, the fact of varying costs through the life of the mine, due to variations in efficiency and wages, need not be considered.

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* President, St. Joseph Lead Co.

† This paper, written by Mr. Crane in 1917, was submitted as part of the discussion, with his permission.

The life of the mine would be the total number of tons of ore in the mine divided by the annual output. For the purpose of this discussion the annual output is assumed to be constant. The total number of tons of ore is the total estimated tonnage as of March 1, 1913. The difference between the average price and the average cost would be the average profit per pound or per ton. This sum multiplied by the annual output would represent the average annual income expected. This annual income is like an annuity for a certain number of years; that is, an income for the number of years that the mine has life. The present-day value of this income is a matter of simple actuarial determination varying only with the amount of interest allowed. For instance, an annual income of $100 extending over a life of 20 years which will produce 2000 actual dollars in that time is only worth $1146.99 if the interest rate is 6 per cent. The same income for 10 years is worth today $736.01. That same income received for 40 years would be worth today $1504.63. This present-day value of the mine or mining property is the capital value that should be allowed for depletion under the terms of the act. Now let us see what sum it is necessary to set aside annually so that at the exhaustion of the mine the present-day value of the mine may be returned to the mine's stockholders. This would obviously be an annual payment that will produce, in the number of years that the mine is to last, a sum of money equal to the present-day value.

To return to our examples: The present-day value of the $100 a year income lasting for 20 years is $1146.99. The annual payment to a sinking fund that would be required on a 6 per cent. basis to produce this money at the end of 20 years would be $31. The present-day value of the $100 a year income lasting for 10 years is $736.01. The annual payment to a sinking fund that would be required on a 6 per cent. basis to produce this money at the end of 10 years would be $56. The present-day value of the $100 a year income lasting for 40 years is $1504.63. The annual payment to a sinking fund that would be required on a 6 per cent. basis to produce this money at the end of 40 years would be $9.72. This, in each case, is a definite per cent. of the annual income.

Now as we have shown that the present-day value of the mine is dependent on the annual income and the number of years that this annual income will be paid, it is obvious that the per cent. of this annual income that should be set aside for depletion is a variable depending on the life of the mine. As the life of the mine becomes greater, it is a smaller and smaller per cent. of the estimated average income. In the case of a mine having only 1 year of life, a fair amount to be allowed for depletion would be 94 per cent. of the annual income, while in the case of a mine with a 40-year life, it would be only 10 per cent. of the annual income.

The allowable interest rate would be 6 per cent. per annum. The present value, the estimated life, the average annual income, and the sinking-fund payment necessary to return the present value at the exhaustion of the mine having been determined, all these factors being based on the law of averages, it would then be fair, in the case of any individual year, to allow for depletion the same part of actual income as the ratio that the annual payment to the sinking fund bears to the average annual income. For instance: In the case of a property with a 20-year life, the annual sinking-fund payment would be 31 per cent. of the average annual income and such a property would be allowed to charge against its current earnings for depletion 31 per cent. of its earnings. As shown, before, these percentages will vary for each mine.

W. O. HOTCHKISS,* Madison, Wis.—A fact to which I would call attention is the difference between the business and the economic, or economists', points of view as to what a mine is. The economists have properly, from their point of view, classed a mine as a wasting asset, but the average metal mine is not a wasting asset if it is a real mine. The economists are correct in a broad sense in that every ton of ore taken out diminishes the orebody by that amount. When it comes to the consideration of an individual mine from the business standpoint, the future of that mine rests on unforeseeable events—the discovery of new orebodies or the development of old bodies beyond the point of probable expectations. Every going mine must develop new ore each year to replace that extracted, and, during all but the last one or two years of its activity, actually does so. Instead of being a "wasting asset" such a mine, from the practical standpoint which must control valuation, is an "increasing asset." From the businessman's point of view, when an ordinary moderate-sized mine reaches the stage when the owner knows the full amount of tonnage available and can figure the life of that property within reasonable limits, it is on its last legs and is no longer a going proposition. This statement is true of the vast majority of mines other than coal mines. There are a few exceptions in the largest metal mines, but they number only a few score out of the 35,000 mines of the country. We can make district totals and range averages and all that sort of thing with fair accuracy, but when we come to apply this method to a particular mine it is absolutely beyond human ability to say that a particular mine is going to produce a certain definite percentage of what you can reasonably prophesy for a whole district. Any method of valuation, therefore, which is based on the estimation of future production and future profits is going to fit properly only coal mines and a very few of the largest metal mines. When the smaller metal mines—the vast majority—are considered, the one great factor in their valuation is going to be, not ore reserves, not profits, not the risks incident to the working of a particular property, but the judgment and experience of the particular employee of the Bureau of Internal Revenue in charge of the work for that particular mine.

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* State Geologist of Wisconsin.

I want to call your attention further to an important matter that I believe will come up for consideration. Suppose a man buys a lot on the street corner for $1000, on which, by the expenditure of a moderate amount of energy and money, he builds a house. Because of a real estate boom, at the end of the year he is able to sell it for $100,000; his income tax is based on that selling price minus what he actually invested in that property. On the other hand, should he expend on the property a moderate amount of energy, time and money and find an orebody worth $100,000, and then sell the property for $100,000, this law permits him, when he has discovered that orebody, within 30 days to revalue that property and to have back as return on capital its full present value. There is a dangerous parallelism in those two cases; I do not know whether this deduction for capital, based on revaluation every time a discovery is made, is going to be able to withstand the light of good business judgment.

Another matter to which I want to call your attention is the effect of interest rate on valuation. This law, carried to its logical conclusion, and assuming that you can accurately measure and revalue a deposit within 30 days after you have found it, means that the discount rate you assume is the rate of profit that this company is going to make, and the rate on which its income taxes are going to be based.

Mr. Norris does not believe the coal industry should be burdened with a high rate. In the iron country they are demanding a high rate of discount, which means a low present value of capital for real property assessment. Then the profits are based on a low capital which means a larger percentage of profit and that commands a larger rate for income taxation. On the other hand, if the discount is made at a low rate it means a larger present value and a lower percentage of profit (although the total amount of actual profit is the same in both cases) and consequently a lower rate of income tax. Now the difficulty comes from the fact that we cannot estimate the value of any orebody 30 days after discovery, and consequently there is going to be much difficulty in the application of this law.

WILLIAM P. BELDEN,* Cleveland, Ohio.—The legal profession, for the gold mines, the copper mines, and some of the iron mines, sought to establish in the courts the proposition that the entire return for the sale of ore, or metals, constituted the return of capital; but the courts rejected that theory and have adopted the one the engineers developed—that ores or metals in the ground have a certain unit value, analogous, if you please, to the stumpage value of the timber in the forest, and that the amount received from the sale of that ore, or from these metals when marketed, is a combination of the tonnage value of the ground plus the cost of operation, with a fair profit to producer. The mining industry owes a debt of gratitude to the engineers who have advised the Federal Administration on this question. The law of 1909 allowed no deductions and the law of 1918 has this whole scheme worked out.

The point that I wish to make in connection with this paper is rather by way of comment than of criticism. The tax law may be divided into three divisions: the income tax, the excess-profits tax, and the capital-stock tax. The question of valuation enters into all three and the principal debatable questions in federal taxation relate to the deductions we may make from income.

With reference to the income-tax law, the question is: What deduction shall we make for depletion? Nothing can be said in addition to what the author has said on that subject. On the capital-stock tax law, the basis of valuation is the fair value of the stock, but the author is in error in his comment on the excess-profits tax. He says, valuation underlies the determination of invested capital for the excess profits and war profits tax. I wish it did.

The regulations issued by the Department say, "In the case of the war profits and excess profits tax, invested capital is based on the actual investments of the stockholders of the corporation, irrespective of the present value of its assets."

This regulation operates unjustly and practically penalizes conservative companies. For example, in northern Michigan, many years ago a property was purchased for $3000, on which iron ore was discovered prior to March 1, 1913. That property is now assessed by the state of Michigan, for purposes of taxation, at $600,000. For purposes of depletion, that sum is used as the basis; but for the purpose of determining the invested capital of the owner on which to compute his 10 per cent. exemption he may only include the $3000. How does this work out?

Two companies, A and B, were organized some years ago, each with $5,000,000 worth of stock. In the course of time they developed additional ore deposits, and by 1913 their properties were each worth $10,000,000. Company A issued stock against its additional developed value so that it now has a capital of $10,000,000. Company B did not do that and retains its original stock at $5,000,000. In figuring the exemption on invested capital, Company A takes out 10 per cent. on $10,000,000; while Company B is limited to 10 per cent. on $5,000,000, because the regulations provide that appreciation in value will not be included in invested capital.

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* General Counsel, Cleveland Cliffs Iron Co.

Invested capital as defined in this Act includes cash paid for capital stock, also stock paid for with tangible property, and to a limited extent with intangible property. It also includes paid-in and earned surplus.

It is our contention that developed value which has accrued prior to the effective date of the tax law, through the discovery and development of additional orebodies, has become an addition to capital analogous to surplus and should be included within the meaning of that term, in determining the amount of invested capital. Unless this is done the law creates an inaccuracy between taxpayers engaged in the same business which, as the author has said, is bound to create unrest and dissatisfaction.

Paul Armitage, New York, N. Y.—The paper has covered the subject of the valuation of the mines very completely, but it has not touched the rate of depletion. At what rate shall we deplete? We find literature pertaining to economics and engineering replete with discussions as to the rate of depreciation. We find all sorts of suggestions as to the rate at which we shall take depreciation and as to how it shall be measured. We find the straight-line method, the reducing balance of cost method, the annuity method, and the unit cost method. But when we come to depletion we find that only one method has been proposed and adopted by the government, and that is what the author has aptly termed "the coincident method." It is the method suggested by the Treasury Regulations with which you are familiar. I may call it the "stock-in-trade" method or take-as-you-go method; *i.e.*, an allowance based on the quantity mined each year. That method is not the only one or the only correct method that should be applied to mines. It is not the only one that is adapted to all the complexities of the various kinds of mining industries with which the Treasury Department will be confronted in its attempt to give us a fair deduction for depletion. Before we approach the question of rate, two things must be settled: First, the value of the property and, second, its term of life.

The question of the rate at which we shall take depletion does not emerge until the two factors have been settled. My discussion is not concerned with either the value of the mine or its life. We have given, therefore, a mining property of value known or agreed upon and its life. We know that at the end of its life, that value will be zero. We further know that under the Treasury Regulations we are permitted to amortize that value. How shall it be done?

The regulations say that the purpose of depletion is to amortize capital. I will briefly quote the section:

> Art. 201. The essence of this provision is that the owner of such property, whether it be a leasehold or freehold, shall secure through an aggregate of annual depletion and depreciation deductions a return of the amount of capital invested by him in the property, or in lieu thereof an amount equal to the fair market value as of March 1, 1913, of the properties owned prior to that date.

It is not necessary to amortize this capital value by depletion rates that coincide with the rate at which the property has changed in value. Now, is it essential that the only way to do that is by this "take out as you mine" method of the Government? I say, "No!" The Government should adopt the same attitude toward the rate of depletion that is adopted toward the rate of depreciation that we are allowed.

The Government is very fair on that; it says on depreciation we may take any standard method, so long as we are consistent.

> The capital sum to be replaced should be charged off over the useful life of the property either in equal annual installments *or in accordance with any other recognized trade practice, such as an apportionment of the capital sum over units of production.*

Why should not we be allowed to deplete, "according to the peculiar conditions in each case?" It may be answered that the law compels this to be adopted. I answer, the law does not compel anything of the sort. A broader, fairer, or more liberal law cannot be imagined. The text of the present law of 1918 is "a reasonable allowance for depletion according to the peculiar conditions in each case." The Treasury Department says, "Only one method shall be allowed to deplete." I ask, gentlemen, what opportunity is there for applying that law if we are going to be fettered hand and foot by this single definite method?

If we are going to apply this law reasonably, "according to the peculiar conditions in each case," we must allow different rates of depletion. For example, in the short-lived zinc mines of the Joplin district, we should allow what I may term a "concave curve," a curve at which the greatest depletion is taken out the first year, and a lesser depletion in the later years. In a porphyry mine where the content is known, where it is more or less of uniform grade or quality, probably the "take as you mine" method of the Treasury Department is fair. But attempt to apply this rule to a deep-veined limestone mine, where we have only 2 or 3 years in sight and a large value of prospective ore and highly different grades of ore, ranging from 43 per cent. down to 7 per cent. copper.

Let us see where the Treasury will leave us. We will first assume—and it is a fair assumption—that the rich ore will be mined first as it is reached first; it is the more profitable, therefore we will get the larger profit on it; the poorer ore we will reach later. Assume that the copper in this mine is 1,000,000,000 lb. of which the high-grade ore averages two-thirds, or 666,000,000 lb., and the low-grade ore amounts to, approximately, 333,000,000 lb. The profit on the high-grade ore is 10 cents per pound, or $66,000,000 in round figures, and the profits on the low-grade ore is 5 cents per pound, or $16,000,000. Adding these figures gives a total value of $82,000,000. Assume a life of 10 years; this gives an expected annual income of $8,200,000. The present value of that income would be in round figures $66,500,000+; amortized over 15 years it would be $59,000,000+. Applying the Treasury rule to those figures for depletion, you must divide the $66,500,000 by the total number of pounds of copper, or 1,000,000,000, and be allowed a depletion of 6.65 cents per pound; or on a 15-year rate, 5.9 cents per pound. In the first years, when we get a 10-cents profit, all is well, we readily obtain the depletion of 6.65 cents+; but in the latter years of that mine's life, when we only have a 5-cents profit on that ore, how can we take nearly 7 cents out as depletion? We must have a different method of provision, or we will lose during the wealthy years and during the later years there will be no depletion.

That, gentlemen, is an illustration of what this rule means. Any fair-minded man will say, and very properly, that we should take more depletion in the earlier years and less in the later years—that would be a reasonable depletion according to the peculiar circumstances of the case. The Treasury rule prohibits it. Therefore, I say the Treasury should amend its rates and give us the same ruling it has in the case of depreciation, and the reasonable depreciation should be allowed, and we should take it in equal installments or in accordance with any other recognized trade practice.

L. C. Graton.—I should like to say that my paper was written, or the ideas, at least, were gotten together from the standpoint of copper-mine taxation, which is the subject for which I am specifically held responsible. It turned out it would not be feasible to have the whole field covered by specialists just at this time, so I presumed to spread the principles of the copper mine over the whole field simply in order to spread the target. I should apologize, therefore, to many of you who are interested in other things than copper for confining my illustrations and, no doubt, for the narrow scope of many of the principles of my application to copper in particular. I am simply one of several engineers down there who jointly will have to take the care of the job.

You will be interested to know that, except for a man to take care of coal, our major organization has now been perfected. Gold and silver are to be under the supervision of J. C. Dick, a mining engineer and operator of long and varied experience. Iron will be handled by E. C. Harder of the U. S. Geological Survey. Lead and zinc are to be in the hands of C. E. Siebenthal, another representative of the Geological Survey. All this work is centered in the Sub-division of Natural

Resources, a reorganization of the national resources work after the departure of Ralph Arnold, who had been acting head of the entire group and had specific charge of the oil and gas program, already well advanced. The senior engineer in oil, who was likewise the senior engineer in the whole group of natural resources, J. L. Darnell, has been put in charge of the new Sub-division. We hope to secure a high-grade coal man, and then we shall feel pretty well prepared to accept and put into practice, so far as good common sense and the law jointly will permit, all the suggestions that have been given and that today's discussion will no doubt increase.

R. C. Allen, Lansing, Mich.—This idea of a specialized administration in the Bureau of Internal Revenue is new. It was attained with a great deal of struggle and trial on the part of the Commissioner and his assistants. This unit is known, I believe, as the Natural Resources Division and is the first of these special units to be developed. Grouped together in this unit are timber, oil, gas, and mines. The problems of taxation of incomes from wasting resources in essence are the same whether dealing with exhaustible timber, exhaustible oil, or exhaustible minerals. The principles are the same. In detail the problems are a little different. Therefore, they have divided the unit into three sub-units, viz., mines and minerals, oil and gas, and timber. At the head of each of these sub-units there is an expert, who is surrounded by his assistants who are peculiarly qualified for the work they have to do. The taxpayer now has a great deal to hope for from the increased efficiency of the administration of the taxes on wasting resources industries which will develop under this specially organized division of the Bureau of Internal Revenue.

The discussion shows some of the trains of thought that may be set in motion by a consideration of this subject. But, in order to block out the limits of profitable discussion, perhaps it might be well for us to consider the subject in this wise: The revenue law of 1918, as applied to mines, leaves certain matters to be determined within certain limits in the discretionary authority of the Commissioner of Internal Revenue; certain other matters are defined and fixed by the law itself. It will not be profitable to discuss amendments to the law. But those parts of the law which are expressed in general terms, leaving considerable latitude for the exercise of discretionary authority by the Commissioner of Internal Revenue, should be thoroughly discussed.

The author gave an excellent outline of some matters that are within the discretionary authority of the Commissioner. The determination of the value of a mining property, for instance, is an act of human judgment. Therefore, the methods of determining the value of mining property, I should say, will profitably and advisably come within the range of this discussion. A second question is the determination of the rate at which the value or the cost of a mine may be written down and charged off as return of capital tax-free; in other words, the rates of depreciation and depletion. A third question is the determination of invested capital in those cases where the books of account do not clearly reflect it.

In exercising discretionary authority, the Commissioner must act through his agents appointed for that purpose and some of them have come to discuss with us ways, means, and methods of approaching and solving some of these complex problems; therefore, we should confine our attention to those phases of the subject wherein we may contribute something of assistance, importance or value. Another of these difficult subjects is the application of the provision of the law that permits the discoverer of a mine to set up in his accounts the value of his discoveries for depletion purposes. Doubtless all of us have been struggling with the problems of determining what constitutes discovery, the date of discovery, and the value of a discovery. There are present today representatives of all of the great mining groups, and I would like to ask the author to explain what is the present opinion within the Bureau of Internal Revenue of what constitutes discovery and what methods have been considered by which the date of discovery may be determined.

L. C. Graton.—I would be glad to give you something definite and specific, but on that topic no final decision has been reached, and, because of differences of view, I do not feel quite free to discuss it.

E. F. Brown,* Iron Mountain, Mich.—The subject is very interesting to the man who has to make out the income-tax statement. The properties in which I am most particularly interested were developed to a considerable extent prior to March 1, 1913. The question arises with us as to the date when the value of that property is to be established as a part of our invested capital and surplus. The corporation, being organized in a very conservative manner, is in a peculiar position perhaps, although no doubt there are many mining corporations in the same position.

The question of exhaustion of resources is another of the problems—the question of how much depletion is allowable in a given year and on what basis that depletion can be calculated. There is nothing in the Internal Revenue regulations, nor in this paper, that gives definite instruction or a definite method to follow in computing either depletion or a percentage allowable from annual earnings for the perpetuation of a corporation's industrial life. The method of ore valuation that Mr. Finlay instituted, and which has been followed in Michigan by Mr. Allen, appears to offer some solution of the depletion proposition, some possibility of solution. That method, however, is imperfect if we consider that the value of a property which is now and has been in the same hands long prior to March 1, 1913, had its value as of March 1, 1913, regardless of the exact amount of minerals at that time developed or that could be estimated.

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* General Manager, Pewabic Co.

R. C. Allen.—Most of the mining companies doing business in the Lake Superior country are somewhat in the position of the company that Mr. Brown represents. They were formed prior to March 1, 1913, and the properties now in operation were in operation on that date. In many of these cases the value of the property as of a date prior to March 1, 1913, will affect "invested capital." However, many mines and corporations now operating in the Lake Superior country have had their birth since then. Perhaps we may cite a case to get at the heart of the matter.

In the usual case the history of the development of a Lake Superior iron mine is this: Some concern or explorer selects a piece of ground where he thinks there is a possibility of the occurrence of ore. He gets an option for a lease and explores the property. If he is successful in locating ore he sinks a shaft and installs a mining equipment. In other words, he discovers and develops a mine. It seems to me that the law specifically extends to that operator the privilege of depleting the full value of his discovery. It says in effect, if the value of the discovery is materially in excess of the cost thereof, the excess value over cost may be added to the cost and the whole sum may be set up in the accounts for purposes of depletion. The cost of the discovery is always returnable through depletion and depreciation. It is the excess of value over the cost that we are talking about. Ordinarily the drilling proceeds continuously until the operator is satisfied that he either has or has not a mine. If ore is discovered and developed by the addition of one drill hole after another, there is a continuous process of discovery but somewhere in the course of the operation some approximate date of discovery of the deposit will probably be suggested. In any case, the date of discovery should perhaps not be advanced beyond the time when the operator elects to take a lease on the property.

Personally, I have had to do with this problem in recent months in the Miami zinc district of Kansas and Oklahoma. This district was developed during the war period; it was not known in 1913. This discovery clause is of transcendent importance to the operators of this district and many of them are making claims for depletion on account of the discoveries they have made. It becomes necessary under the law to fix the date and the value of each discovery with respect to the peculiar conditions in each case.

Here is one test that was applied in determining dates of discoveries in the Miami district. In that district it is necessary to mill the ore; if an orebody is found, a mill is erected. We have in no case advanced the date of discovery beyond that of the beginning of the erection of the mill to treat the ore, taking it for granted that an operator would not erect a mill unless he thought he had discovered a mine. Extension of known orebodies by drilling after the date of beginning construction of the mill were not believed to constitute discovery but ores discovered prior to the erection of the mill were included in the discovery value. That is merely one test that has been used. It would not apply to the iron mines of the Lake Superior country. You would have to use other tests there In fact, each district has its peculiar types of orebodies and mining operations, so that a test of discovery in one may not apply to another.

William Kelly, Vulcan, Mich.—The date of discovery in most mining operations cannot be definitely fixed. In such mines as the zinc mines that lie near the surface, as has been suggested, the date perhaps can be determined; but where an orebody is found by drilling, even after drilling has continued a long time, there may be doubt as to whether there is a commercial body of ore and it is necessary to sink a shaft or extend drifts to open up the body. It may take a long time before a commercial orebody is definitely determined. Why is it necessary to limit the time of determining the value of a discovery to 30 days? After a period of 30 days, in certain cases, there may be a determination of value but in the majority of these cases it will only be partly determined so that some time later the value of the discovery will have to be redetermined. I cannot see how there can be a limitation of time of discovery in the great majority of cases.

L. C. Graton.—There is one point in connection with this matter that all interested in mining might well consider. I understand that the discovery idea, and the features entailed with it in the law, like the 30-day period, are the results of a strong effort, made at the time the law was framed, by representatives of the oil industry to see to it that their industry was afforded such necessary provisions as would give it a square deal. Metal and coal mining, I understand, were not so strongly and persistently represented. One apparent consequence is that the law, as it stands, is going to be rather easier of application to oil and gas wells than to mines.

R. C. Allen.—Do you think that clause applies to the discovery of mines with exactly the same meaning and with the same force and same effect that it applies to oil?

L. C. Graton.—That question has not been at all settled. My own inference from the law is that it intends to accord to the owners of mineral deposits concealed in the ground certain benefits consequent upon the disclosure and proving of those deposits, and that it intends that mines on the one hand and oil and gas wells on the other should share in these benefits in an absolutely equivalent and equitable fashion. But this does not mean that the means for attaining this object must be identical in both instances; it cannot have such meaning without ignoring the actual differences between oil or gas wells and mines and without violating common sense. Equivalent, not identical, treatment is required for such cases.

As already intimated, the language of the law is more the parlance of the oil man than of the miner. But it is our duty, as I see it, to make the evident intent apply equitably to each. The actual discovery of an oil well, of course, is a thing of perhaps only an instant. The drill reaches the proper point—and out rushes the oil, commonly in great volume. Experience has shown that from the initial flow of an oil well, or from the flow as it settles down after a few days or weeks, the value of the well may be deduced with a good degree of accuracy. So for that, the 30-day period is satisfactory. Ore and coal, however, do not trip eagerly out to meet the miner the moment he taps the margin of their deposit. To ascertain how much is present and what it is worth, he has to go into the deposit and he has to spend time and money to get there.

Personally, I see no way of making the discovery clause effective except by saying that to "discover" is to "determine." That, it seems to me, is what the law really means, what it must mean. If you determine that you have a mine, then you have discovered a mine; but unless you have reasonably completed that determination you do not know whether or not you have a mine or what its value is. Then, as I take it, after you have made the determination, you have 30 days of grace to think it all over, put in an extra round of shots in a place or two that look particularly good, figure up the tonnage, grade, and value, and then make your showing.

I realize that this is heretical doctrine from the standpoint of the older regulations, but I believe that it is what the 1918 law intends. I realize also, however, that there are things which we, as engineers, might like to read into the law but which, in fact, we cannot do. Even so, the case may not be hopeless, because while the present law is presumably to flow on indefinitely in its terms and its rates of levy, we may feel confident that if injustices or inconsistencies are revealed in it, they will be remedied. At the worst, the industry will have to bear them only for the time required to disclose and demonstrate them.

Probably this matter of education on both sides is going to be a rather slow job. I say education on both sides, for I believe that mining men themselves need to take a new and better grip on their understanding of their business. They have got to go to the bottom of what they are doing, analyze the entire situation thoroughly, and acquire a complete and masterful understanding of the fundamentals. For instance, it is my belief that we are selling a lot of our mineral products at prices that do not return true cost. If that is so, then there is only one answer—the prices must go up. We cannot find real remedy in legislation, in depletion allowances, or in any other artificial means if we are selling our products at prices that include only certain costs without taking into account the fundamental cost of the stuff itself.

Take, as an illustration, the example to which Mr. Armitage referred. If the mine he mentioned is going to face the latter half of its life without enough profit in the ore to cover the cost (or value) of the ore to the company, then only one of two things can happen. Either the company will quit business—but that cannot be for the copper from that mine and others like it is needed by the world's industries—or the selling price of the product must be raised to the point necessary to cover all costs plus reasonable minimum profit. If the mine cited were unique as to its position in this regard, it might indeed have to close down, but I believe it is not unique, but rather representative.

I am certain that millions of pounds of copper, and of other metals as well, have been sold in the past without the slightest regard for their inherent cost or value and without a single provision for replacing them—in other words without providing for continuing enterprise. This is unquestionably unsound economics. Our American producers of mineral products are entitled to no credit for underselling the rest of the world if they have been able to do so only by using up the best of the deposits which nature so bountifully placed within our borders and which, while the country was young and knowledge of these things imperfect, could be secured for a song. Such a policy is unsound and must be as short-lived as that of the profligate who squanders both income and capital in riotous living.

If the present heavy taxes, by forcing mining companies to examine and analyze minutely their true situation, have the effect of driving home these lessons and of instituting a sounder basis for the conduct of the mining business, I believe the game will have been worth the candle. Indeed, for my own part the appeal and inspiration of watching in at the game while such an outcome as I have implied is taking shape is one of the bright prospects in what will be a trying, and perhaps thankless, job of helping to administer the tax laws to the mines.

Let me not be understood as implying that the situation to which Mr. Armitage referred has the only application or consequence I have implied above. As a matter of fact, I for one have gained from his example a clearer understanding of what depletion really is and how it ought, in fairness, to be computed, than I have had hitherto. He has shown us, I think, that one pound of copper in a mine is not necessarily the same as every other pound—that the extraction of certain pounds of metal, because being contained in rich ore they represent greater profit, inflict a greater impairment of the value of the mine than the exhaustion of an equal number of other pounds which cost more to produce because contained in lower-grade ore; and of course, no impairment of mine value would result from the removal of countless pounds of copper if contained in stuff of too low grade to yield any profit.

R. C. Allen.—The question that all of us have in mind is simply this: Do discoverers of mines benefit by the clause in the law which apparently in its wording permits them in case of discovery of a mine subsequent to March 1, 1913, of a value materially in excess of the cost, to set up in the accounts the value of that discovery to be returned tax-free out of the income of the operation of the discovered mine? That is, as I take it, the heart of this discussion. Does the mine operator benefit by this clause in the law and if he does, to what extent? This is a matter of enormous importance to a great many miners in this country now and if the clause stands in the law, it is going to be of great importance in the future.

A. D. Brokaw, New York, N. Y.—While I was in Washington, I had considerable difficulty with this matter of date of discovery and, unfortunately for you and for me, it fell upon me to throw very hastily together that document known to you as Form A revised, Questionnaire on Mines. By scurrying about in the Bureau of Mines and among my mining acquaintances in New York, I decided that the best I could do was to say the date of discovery of the mine is the date upon which it could have been shown by an experienced competent engineer that there was sufficient material in sight to justify operation on a commercial scale.

I may say that I have had the temerity to set up a claim for revaluation on the basis of discovery in the case of a coal mine in the New River district. This falls not under the clause of new discovery, but under the clause of the regulations that allows valuation in the case of a mine that has been abandoned or sold as essentially worked out. I do not remember the exact wording. This particular working had never been profitable. An entry had been driven some thousand feet back in coal of indifferent quality, when what is called a "swag," a channel in the coal, was struck. This was a sharp dip of perhaps 12°, the coal rapidly thickening up to 6 ft., and then rising on the other side and pinching down to 6 in. This coal had been mined out and it was necessary to drive entries and carry on exploration work in order to find out whether the mine could be operated at all. I may say the mine was sold on a basis of something like 85 per cent. of the actual appraised value of the equipment in the mine. By driving about 1200 ft. of entries beyond the break, or fault, they got into coal of workable thickness. In doing that, many hundreds of feet of work was done that did not yield any result.

It did not seem fair to me to place the date of discovery and the values at the moment coal of approximately 40 in. thick was found. In setting up this value, which was a retroactive one, I took a date on which the company had blocked out sufficient coal for it to know that it was going to make money and eventually pay back the original investment in that equipment; which, if the mine turned out to be worthless, was nothing but junk. I think that the thing that we have to rely on in setting up these valuations is the absolute fairness of the men of the Bureau of Internal Revenue. In other words, if we make our case and show that we are fair, the Bureau will be entirely ready to meet us half way.

Mr. Armitage was discussing the necessity of some depletion rate different from the one determined for the total, or for the mine. If that problem were for me to meet, I should not hesitate to do it in this way. In making a valuation of a mine, an engineer will always place valuation on the various blocks of ore according to the quality of the ore and the probable cost of extraction. If these are not grouped before they are discounted, but are discounted to present value independently, according to some logical plan of operation, I believe that there would be no difficulty in placing values on the ore in each block or in each stope, if necessary, and then with proper records kept, it would be perfectly possible to apportion the return of capital according to the actual difference in the cost of the ore. If we distribute the unit cost throughout the whole body of ore evenly, it is a fictitious unit cost, because we do not pay as much per unit for 2-per cent. as we do for 5-per cent. copper ore.

One other matter of great interest to me is the replacement fund. It is good economics and naturally we all want it. I think, however, it is necessary to consider, if we accept the dictum that a mining organization must be a continuing organization, what it is right to expect the government to allow us to perpetuate. Is it the actual value of the assets of the company? If so, the present regulations meet the situation. Is it the earnings? In that case, the necessity of a replacement fund depends, not on the increase of cost of new deposits but on any change in ratio of income to costs of new deposits. Thus, if you buy a deposit now for more than it would have cost 10 years ago, you are no worse off because it costs more if it yields you a return proportionate to the increased cost. The third thing we might desire to perpetuate is the productive capacity of the organization; that is, the output of metal. If that is desired, it is necessary to establish some sort of replacement fund. That, however, involves almost insuperable difficulties in formulating an administrative regulation. It might possibly be done by allowing a revaluation each year and depletion on the basis of such revaluation with the idea that the newly deducted depletion allowance would be immediately invested in reserves. However, the regulations specifically say that there shall be no revaluation after once the valuation has been made and we need hardly waste our breath in trying to modify that, because there is a rather rigid determination to hold to that point; and, after all, is it not true that this increase in replacement costs is faced by practically every industry that we have?

Our railroads are facing the replacement of equipment at costs very much greater than they were 10 years ago; should they be allowed a fund to meet this? Should it be a depreciation on the basis of replacement cost rather than on the basis of actual cost? If that is done, should they be allowed to capitalize the depreciated cost, or should they not? There are a good many complications of the sort. The manufacturer is in the same position. He purchases raw material this year, manufactures it, and pays a tax on the difference between what that raw material cost him plus the cost of manufacturing, and selling price. He does not pay a tax on the basis of what it is going to cost him to replace that stock he used during the past year and I really question whether it is entirely fair to other tax-paying industries for the mining and oil industries to be allowed special consideration in the way of replacement funds.

R. C. Allen.—Mr. Brokaw's last remark suggests to me the importance in all discussions of this sort of keeping in mind the fundamental principle of income taxation. An income tax levies a tax on gains and profits. Gains and profits of mining are taxable just as the gains and profits of other industries and activities are. But capital is not taxable. The theory is that any thing one owned prior to Mar. 1, 1913, is not taxable, only the subsequent gains or profits are taxable. From the income of the mine a certain portion representing a portion of the capital value on March 1, 1913, is deductible each year as depletion. But this clause that we are discussing goes beyond the idea that only the income in excess of cost or of value as of March 1, 1913, as the case may be, should be taxed. Congress realized, after years of study of income taxation, that mining is a peculiarly hazardous business and that mines should not be taxed on precisely the same basis as banks, for instance. This is a special relief extended by Congress to the mining and oil industries in recognition of the peculiar hazards in those industries. The extent of the relief and the manner of its applicaton are left largely to the discretion of the Commissioner of Internal Revenue.

Those of us who were in the Bureau of Internal Revenue at the time the law was passed and who had to do with the interpreting of the meaning of that clause of the law tried for weeks to define the meaning of "discovery of mine," in such a way as to include everything that could be fairly interpreted as discovery and to exclude everything else. Pages were written on it and eventually consigned to the waste basket. You have the results of our combined labors in Article 219, Regulations 45. Perhaps by the time the experts in the Bureau have determined the taxes on some 1000 or 2000 mining corporations under the Revenue Act of 1918, they will have a much clearer idea of how that discovery clause may be applied equitably and fairly and will know more about what it means. We have not reached the bottom of this subject yet.

Paul Armitage.—The purpose of the clause in the 1918 law relating to discovery is to extend to the mining industry a help and benefit in certain emergencies; those conditions are the discovery by a taxpayer of a mine. The subject is very much clouded by a certain preconceived idea that we all have about the discovery of ore sufficient to justify the location of a claim. That is not what the law refers to at all. The Act says "In the case of mines, oil and gas wells discovered by the taxpayer." So, the taxpayer must discover a mine. He has got to discover a mine; nobody has a mine because he has discovered ore. The two words used are "discover" and "mine." Let us see what the meaning of those words is, for in order to understand the law those words must be given their ordinary meaning.

The word "discover" in its primary meaning means to uncover, to expose to view; that is the ordinary meaning. The second meaning is the act of finding out or bringing to knowledge what was unknown; a third meaning is "exploration."

Let us take up the word "mine." The first meaning of the word mine is "an excavation for digging out some useful product for ore, metal or coal." That is not the meaning used in this Act; in that sense, no mine is ever discovered. We never discover the excavation made for the purpose of digging out the product. The other meaning of the word mine is "a rich or productive source or supply;" in that sense, as when we say "a mine of wealth," it means a valuable deposit, a large deposit. This is the sense in which that word is used in the law.

It can mean nothing else than that for two reasons: In the first place, the law says that the value of the mine must be "materially disproportionate to the cost." To apply that, we must have discovered something which is of great value. How can we assert that it is disproportionate to the cost until we have uncovered something of value?

The second reference in the law is that the property must not be acquired as "a proven tract." It distinguishes between a proven and an unproven tract. Carrying forward these meanings, let us turn to the Act.

If we put these two words into the law, what have we got? We have the uncovering, the exploration of a valuable source or supply of ore; and until we have those things, plus the fact that it is materially disproportionate to the excess of the cost, we have not discovered the mine, in any sense of the word. Those things must exist before you can apply the law.

The only remaining question is: When have we reached a point of time when we can say, we have now discovered and explored a large or abounding source of ore materially disproportionate to the cost?

I do not really think in its practical application there is going to be any difficulty in determining that particular point of time in the discovery of any particular mine. As I take it, this is what usually happens in the case of discovery of a body of ore. First, a person comes up on the ground and discovers ore and locates a claim; there is no discovery as yet. Then the capitalists, the mining men, purchase this unproven tract and start in to discover a mine; that is, explore the ore in that mine. That the law permits, because it says that until they purchase a proven tract the discovery provision is open to them.

The exploration proceeds continuously in almost all cases to a point where the operators sit down and say, "Now, we have a sufficiently large body of ore to start an operation, to build a smelter, to construct workings, to organize our force and go ahead." Until they have reached that point, they have not discovered the mine. In the vast majority of cases, that point is readily ascertainable. That is the time of discovery. In valuing that mine, you are not limited to the ore that you have exposed, but you can add to that ore a reasonable estimate of probable ore. It may be true that you may underestimate but, at least, you get not only the value of your discovery, but your discovery plus the probable ore. The sum of these will be the value for the purpose of depletion.

If the Bureau of Internal Revenue will apply the law in this way, I think it will justly administer the law, and benefit the mining industry and give us the full benefit that Congress intends. I think it has already interpreted the regulations as meaning that. I do not think the question, unless the Bureau wants to modify or change its regulations, is really an open one. Art. 219 says,

> The discovery of a mine or a natural deposit of mineral, whether it be made by an owner of the land or by a lessee, shall be deemed to mean (*a*) the bona fide discovery of a commercially valuable deposit of ore or mineral of a value materially in excess of the cost of discovery in natural exposure or by drilling or other exploration conducted above or below ground.

There is a definition exactly in accordance with what I believe the contentions of Congress were, as we interpret it.

The second part is that, "the development and proving of a mineral or ore deposit which has been abandoned or apparently worked out, etc." So I say that right in the regulations is the very definition of this discovery that we need. If we apply common sense to that in the history of each case, in determining a fair valuation of the property, I think that the law will be readily administered when the facts are laid before the Commissioner in each case.

R. C. Allen.—Assuming that Mr. Armitage has a fair interpretation of the meaning of the words "discovery" and "mine," after the valuation of the discovered orebody has been made, it follows, does it not, that the value of further additions to that ore in the ordinary course of mining is not to be added to the value of the property for purposes of depletion. Suppose that on Dec. 31, 1915, an orebody consisting of 1,000,000 tons has been developed and valued as a discovery. As years go by, the orebody is extended in the ordinary course of mining and 5,000,000 tons of ore are taken from it before it is exhausted. The amount allowable under this clause for depletion is the value of 1,000,000 tons and not 5,000,000 tons, is it not; in other words, a mere extension of known ore does not seem to constitute discovery within the meaning of the law. The regulations insist that no revision of the value shall be made after a property has once been valued for depletion purposes. There is a limit to the depletion allowance. You may prove that the orebody was much more valuable than was thought at the time it was valued but the valuation of anything as of a given date must take account only of those things affecting and determining value that were known at that date.

Take another case: Suppose that an orebody is discovered, measured, and valued, and the amount set up in the accounts. At the time of this valuation, no other orebody is known to be on the property, but later a second orebody is discovered, quite apart, separate, and distinct from the one that had been earlier discovered and valued. Does the second orebody constitute a discovery of a mine? It may or may not be so located in the ground that it may be exploited from the plant already installed. It may be necessary to sink a new shaft and put in a new plant. It may, in fact, be a new mine in the ordinary meaning of the term. Is it a discovery in the meaning of this law? While it seems to me that the taxpayer is debarred from considering as discovered ore a mere extension of a known body, if he discovers a body of ore in no way connected with a body that had been earlier discovered and valued, if there is nothing in and about the first orebody or the property that would in any way suggest the occurrence of the second one, he has made a new discovery. The question is, is he entitled to the value of that second orebody for purposes of depletion?

Let us take another case: In the case of an adjoining property, suppose that one operator has developed the ore up to his boundary when the second operator extends his neighbor's discovery into his own property. Is the second operator entitled to discovery? Personally, I think not. The first operator's discovery extended to the boundary; the second fellow extended the other fellow's work. That, in my opinion, is not a bona fide discovery, for the law says that the taxpayer must make the discovery in order to benefit by it. So, it seems to me that if we start out with a clear understanding of what seems to have been the intention of Congress, to reward only the bona fide discoverer, and then consider the facts and circumstances of each individual case, it will be possible to come to some fair, reasonable, and equitable explanation of this clause with respect to the circumstances of individual cases.

If you will permit, I would like to raise another point in the law. Some of the speakers have referred to the unit method of determining the rate of depletion. I heard no commendation of that method; I heard some very severe criticism of it. Having established the sum that we may be permitted to set up on the books to be returned tax-free out of earnings, at what rate is the sum to be returned? How much may we deduct in each year until the whole amount is exhausted? The regulations say that the best way of doing that is to determine the number of units in the property and divide the whole value by the number of units to arrive at the value of each unit. Multiply the value of each unit by the number of units mined during the year and you have the sum that you are permitted to subtract from your income of the year as depletion.

It may be that the income of the year may not be sufficient to take care of the depletion allowance for the year. Can you carry the balance over into the next year? May you write off on your books merely your income, carrying forward into the next year the amount of depletion that your income does not take care of in order that you will be assured the return of the whole amount providing the property will pay it to you? It seems to me, gentlemen, that the law does not intend that the taxpayer shall be assured of the return of his capital provided only his property will pay it out. Many of you have a different idea. Therefore, we ought to discuss this matter.

It is necessary in the computation of an income tax to consider an accounting period. The law specifies the year of 12 months as the accounting period. The tax is assessed with reference to gains in that period. The regulations specifically instruct the taxpayer that he may not write off losses of one year against the gains of a succeeding year. It seems to me that if the income of the period of 12 months is not sufficient to take care of your costs, including depletion, you have lost part or all of the depletion allowance for the year. The deficit cannot be carried forward and subtracted from the profits of the succeeding year as expense or cost of the operations of that year. The law does not assure the keeper of a store that his capital invested will be returned to him. He is permitted and required to write off depreciation and other costs of the year, and no more, irrespective of the amount of his gains and losses. If his income is not sufficient to take care of his costs he cannot carry forward a deficit to the succeeding year and write it off as a loss.

May the rate of depletion be varied from year to year? I think it ought to be under some circumstances, and I think that the regulations partly cover the subject when they say that if a mistake has been made in the estimation of the number of units in the property whether it be an overestimate or an underestimate, a revaluation of the unit of depletion may be made; such revaluation of the unit affects the rate of depletion. But this revaluation of the unit does not affect the total allowable sum for depletion. It is necessary, of course, to have some measure of the amount of depletion that may be written off from year to year. If the unit method of figuring depletion is not a good method, then what is a better method?

F. W. Sperr,* Houghton, Mich.—I agree with the author that the value of mining property for taxation should be based on the price at which an expert valuator of mines would recommend the property for purchase to his client; for that makes it possible that mining property shall be fundamentally on the same basis as all other property as to method of valuation for taxation, which is as it should be. The tax assessor is presumed to know the value of property coming under his own particular assignment. In the mining business he is called an expert. His basis of value is always, whether consciously or unconsciously, the price at which we would recommend the property for purchase to his client; and the valuation returned for taxation is usually at some fixed ratio to this price. Then for the same tax this ratio should be the same for all kinds of property, and all questions of differences should go before a special board of review for each particular tax, so that all property interests can feel that they are on the same footing in the matter of taxation. But, in the proposition to group mines for special methods of valuation, if I understand it correctly, I can hardly agree. It seems to me necessary to leave the valuation wholly to the assessor, the expert, if you please. It may be necessary and advantageous to classify mines and assign special assessors to special classes, but each assessor must be competent to pass on the values of the properties in his own assignment.

Hoskold's method of mine valuation was undoubtedly old among mining financiers long before it found its way into literature, and I have never known any fault to be found with it as a fundamental proposition; but as to the details of its application, there are sometimes greatly diverse opinions. Smock used the method in New Jersey many years ago, Finlay used it in Michigan more recently, and it is still being used in Michigan, for there is nothing else to be used. You cannot get away from it any more than you can get away from the method in use for finding the value of a perpetual annuity. But there should be uniformity in the application of details to each class of property. Then, no doubt, under the conviction that true valuations are at least fairly well approached, differences of opinion and charges of unfairness will largely disappear.

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* Professor of Civil and Mining Engineering, Michigan College of Mines.

Furthermore, I believe that under such a system of valuations the anticipated difficulties with local taxation will be greatly minimized, if not altogether eliminated, because it will be possible to fix the ratio of tax valuation to actual price value the same for all property subject to the same tax.

C. H. BENEDICT,* Calumet, Mich.—One point that has not been brought up is the question of the rate by which we shall establish the amount to be set up for the depletion account. I think some of us would rather question whether a mine, which has once been established and for which a proper depletion account is set up, would be considered more hazardous than the ordinary business enterprise in which a man pays the tax on the profit of his operations and is not permitted to set up a capital account. Everyone understands that if you consider mining a hazardous operation you should be entitled to a higher return on your capital account. You are faced at once with the fact that if you are going to capitalize your earnings at a higher rate, you are going to have a lower amount to deplete.

I should like to know how they are going to steer the good ship between Scylla and Charybdis in this case. It is my contention that a rate of 6 per cent. on the capital is an ample return, in addition, of course, to such an amount set up that at that same rate of interest, 6 per cent. or 4 per cent. will finally return your capital. It is not my belief that it is necessary to establish a return rate of, say, 8 or 10 per cent. on the original capital. It appears to me that this is well worth discussing because on this is going to hinge the depletion value that you are going to be able to set up.

One example that I may give, where the hazard is a minimum, is a certain mine that is located in a lake. That mine is entirely developed, the records show exactly the tonnage of ore available, it has not been disturbed, and the values that may be recovered are fairly well known as well as the cost. What the selling price of copper is going to be is not known. I should like to know in a case of that kind, after correctly estimating the cost and the selling price over a period of 25 years, what would be done as regards a profit over and above the original property value?

L. C. GRATON.—I believe I understand Mr. Benedict, yet it seems to me that the Calumet & Hecla Mining Co. would not endeavor to reclaim and rework those old mill tailings unless a profit resulted from the operation. If they do derive a true net profit, it will of course be taxed. And in arriving at the present value of that tailings "mine," we must insert into the computation a reasonable rate of profit. I thoroughly agree that this body of tailings, because of the definite knowledge of its copper content and of the minimum of risk attached to its recovery, should be valued on the basis of a relatively low interest rate, a rate perhaps lower than any other mining property that now comes to my mind. If money is worth 6 per cent. in a good sound investment, it certainly cannot be worth much over 6 per cent. in this case.

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* Metallurgist, Calumet & Hecla Mining Co.

We have not endeavored to make up our minds as yet regarding proper interest rates for various classes of mines. We are gaining ideas on which to base decisions from this discussion, as I thought we would. If the Board of Directors of the Institute will appoint the special committee to confer and coöperate with the Bureau of Internal Revenue, I feel confident that reliable results will be reached. So, for the present, at least, do not be alarmed lest we shall adopt interest rates that are too high to be just.

On the other hand, must not we who, I suppose, will have to make the final decisions, take properly into account that in the case of a mine, say the Calumet & Hecla, since it has been cited as an example, which has a long, unbroken profitable record behind it, its owners and operators will be likely to regard it more highly than most other people do? Will not their estimate of its worth and of the security of its capital presumably be tinged by their intimate knowledge of its dependability in the past? Will not its unusual and persistent prosperity, with which they have been so closely in touch and principally engrossed be likely to make them more optimistic toward mining in general than is justified by the average outcome in mining?

Clearly enough, the valuations that must be set up cannot reflect solely the owner's appraisal but must endeavor to reflect that price at which the owner and an intelligent prospective buyer would strike a bargain. On such a basis, I am inclined to believe that the average buyer of mines, unable to efface entirely from his mind the memory of many a failure, would insist on a risk factor above the 6 per cent. standard in the case of even the best of mines. I do not attempt to say how much above 6 per cent.; and I realize that the argument I have just set forth must not be carried too far, for, as I have sought to establish in my paper, it must be recognized that the average profit-paying mine is naturally, and necessarily, worth more to its owner than to any one else. That this is actually true is attested by the fact that the owners of such mines rarely dispose of them. Therefore, it seems to me, a certain liberality must flavor the valuations.

C. H. Benedict.—I wish to add that I did not wish to be understood as saying that the Calumet & Hecla Co. did not expect to make a profit out of this operation. We did expect that the return would be no greater than what is represented by a tangible asset, as of Mar. 1, 1913, and therefore should not be taxable as income, but returnable as capital.

W. O. Hotchkiss.—Mr. Benedict's example illustrates the point I endeavored to bring out, and that is a property which can be completely estimated. Let us make a few assumptions and make a concrete example of this. Assume that $50,000,000 will be the total profit in recovering that copper, and that it will be received in ten yearly installments of $5,000,000 each. At 6 per cent. discount with 4 per cent. sinking fund rate, the present value of this future profit is approximately $35,000,000. At 12 per cent. discount, its present worth is approximately $25,000,000. According to the discount rate chosen, one of these sums would be the "capital" on which a "depletion charge" would be based, and the remainder of the $50,000,000 would be profit on which income tax would be paid. If $25,000,000 is taken as present value, the rate of profit will be 12 per cent. and the rate of income tax will be higher than in the other case where the capital is taken as $35,000,000. In the first case, total profits (in addition to capital return, or depletion charge) will be $25,000,000, which will pay a higher rate of income tax than the $15,000,000 total profits in the second case. Now the actual profits of the company are going to be the same no matter how we may juggle with "depletion" and "discount rate" and other abstract terms. The point is that it is of a great deal of importance to the company whether 10 or 6 or 3 per cent. is the discount rate taken to arrive at the present value. Furthermore, it is of interest that the higher discount rate resulting in the payment of the greater income tax is of advantage to the company when the matter of real property tax is considered, as the higher the discount rate, the lower is the present value.

R. C. Allen.—It seems to me that the decision with respect to the rates of interest to be used depends entirely on whether you choose to treat hazard as a function of interest. Most writers on the subject of mine valuation so treat hazards; but inasmuch as the valuation of a mining property, under the general method which is used by almost everybody who has to do with this subject in a practical way, is in the end a result of mathematical calculation, it is possible, if we choose, to eliminate the hazards in the weighting of the other factors and to use an ordinary rate of interest in reducing expected income to present worth.

The Michigan Tax Commission concluded that, with respect to the iron mines of Michigan, it is more logical to eliminate the hazards in the estimates of tonnage, cost, price, and life so that a uniform and moderate rate of interest can be used in all of the calculations of value. If in the calculation of value we treat all hazards in mining as a function of interest, it is necessary to establish a sliding scale of interest rates, to be applied to groups of properties and even individual properties in a group, in accordance with the actual hazards in the group or mine.

In the case of the Calumet stamp sands we have an illustration of the extent to which the hazards may be reduced by the precision with which the controlling factors of value may be estimated. The tonnage of sand is known; the copper content of the same is known; through years of experiment there has been developed a standardized method of extracting copper from those sands at a cost that may be estimated with precision. So, as Mr. Benedict says, the factors of valuation are well established. As a matter of fact, all of the factors that enter into the calculation with respect to those sands may be determined with a relatively fair degree of precision. But compare the recovery of copper from Torch Lake stamp sands with the mining of gold in Cripple Creek. In the estimation of the value of a Cripple Creek mine all of the factors are uncertain save the price of gold.

The value of a mining property is, after all, not easily determined by arithmetic. The estimate of the value of a mine is an application of human judgment and it seems to me that the Bureau must be free to exercise its judgment in each individual case as well as with respect to a group of similar mines. No one set rule or method may be devised that will apply equitably to all mines even in a single district or large group unless it be so elastically constructed that in its application it amounts to the use of alternative methods based merely on the common general principles that underlie the estimation of value of all expectations of income from wasting natural resources whether it be from a mine, oil well, tract of timber, bed of marl, or clay bank.

E. P. Griffitts, Chicago, Ill.—It seems to me that your discussion of this matter has possibly been from a narrow point of view and that you are forgetting some things that are allied to this matter. You say the mining industry is one of hazards and, yet, are you the only ones who received the government questionnaire? I spent an afternoon about two weeks ago with the president and the secretary of the American Pulp Association, who had this matter on their minds in behalf of the industry and were about to appoint a committee to consider the same subject. That same evening I spent several hours with a finisher of very fine textile fabrics, and he had the matter on his mind. He said "There is nothing standard in our business. Overnight the converters may get it into their heads that there must be a different kind of finish for fabrics." You have been talking about hazards of the mining industry. I do not see that your business is any more hazardous, generally speaking, than almost any other business. If you furnish 50 or 75 per cent. of the tonnage to the railroads, you are putting the railroads in a precarious business, if your business is hazardous. One of the speakers said he thought the matter ought to be handled the same as any assessed value, or like real estate. I think he did not fully realize that in this matter we are in a much more favorable position. We are going to be asked to assess ourselves, to assess our own property. If the Government furnishes anything, it will be a formula for us to apply.

W. O. Hotchkiss.—Most of the discussion has concerned itself with what, in point of numbers of mines, is of importance to an exceedingly small minority of the mining people in the country; therefore our experience in Wisconsin may offer some suggestive points and lead to further discussion.

When the Legislature turned over to the Geological Survey the task of determining the values of the mines of the state for assessment purposes, we undertook to apply to the zinc mines of Wisconsin the same methods of valuation as had been applied to the iron mines in the northern portion of the state. But in one year insurmountable difficulties were encountered. We would prepare our figures of valuation in the springtime—our assessment date is May 1—but by June a given property might be exhausted and have no value, while another might have run into a new orebody, and doubled its value or perhaps increased it ten times. That one year's experience proved to us the impossibility of applying the method of present value of future profits to the small, short-lived property. It is absolutely impractical.

The development of an orebody there began with the drilling for the zinc, which was carried only to such an extent as to develop a body of ore that would, with reasonable certainty, guarantee the cost of erecting the mill to treat that ore and sinking the shaft. Any profit received came from pure good luck in the extension of that orebody farther than the drilling had shown. In almost no case was there a mine in that section of the state which had visibly before it six months of life, and yet the average life of those properties would be between four and five years.

It was absolutely impossible to value those mines by discounting to present value the expected future profits, so it was necessary to draft legislation providing for a certain percentage of the revenue derived from the operation of that property of the preceding year as the basis of taxation for the succeeding year. In the vast majority of mines that are now going to be taxed under this federal income-tax law, that same difficulty will present itself.

The small mine never has an assured tenure of life ahead of it that will warrant a valuation such as you may expect from the basis of averages in the district. Such district averages and forcasts can be made with much accuracy in many cases, but it is wholly unjust to apply to any individual mine an average figure for the whole district. That mine may come up against a bar 20 ft. ahead of the breast of the ore at the time you assess it and the orebody may absolutely disappear. On the other hand, it may go for ½ or ¾ mi. No one can tell which it will do.

Any logical viewpoint of the valuation of the property of that sort for purposes of purchase or assessment is going to mean that the taxing official must put himself in the frame of mind of a possible buyer of that property. What is he willing to say is a fair sum that he could risk on that property individually and obtain his capital back with interest? To do his job rightly he should be blessed with a foresight that would make him the envy of the whole mining profession.

I do not see how the present federal income-tax law can be made to apply logically to that type of mine. When we began our studies we considered as absurd the methods of taxation adopted in many of the western states, in which they took a certain proportion of the yearly gross, or net, proceeds from the property as a basis of the tax. But after the most careful study we were driven to adopt the same plan. We still continue to value the iron mines of the state by the method of present valuation of the expected profits, and it works to the satisfaction of everybody concerned. These other properties, for a considerable number of years, have been taxed on a certain percentage of the preceding year's income.

James J. Forstall, Chicago, Ill.—There is one question in connection with the income tax, materially affecting mine owners, which has not been touched on here. It is the question of the taxation of increases in capital value. Under the Sixteenth Amendment, can you tax, as income, an increase in the value of capital merely because it has been turned into cash or some other form of property? That is of great importance to mining concerns where property has been bought for a small amount and sold at a tremendous increase.

Take the case of a mine that is put in trust, the income to be given to certain people and the remainder, after the death of these people, to go to some one else. If that property was worth $100,000 at first and was later sold by the trustee for $200,000, it is perfectly plain that the additional $100,000 would have to be kept intact as part of the principal of the trust estate and would not be income. If such a case came before the Supreme Court it would certainly so hold. Could that court, then, in the same breath, say that the Government can tax this additional $100,000 as income under the Sixteenth Amendment?

A. D. Brokaw.—One of the greatest objections of the present law, and more particularly the 1916 and 1917 laws, has been the prohibition of sales on account of the enormous taxes involved, particularly in the case of sales by individuals in process of consolidation. I am interested, indeed, to know that there is even a question as to the constitutionality of the income-tax law in assessing tax on incomes derived from increasing value of property.

Paul Armitage.—Is all property taxable where part of it had arisen before March 1, 1913? Suppose a man purchased property for $100,000, 10 years before 1913, and sold for $200,000 5 years after 1913; is that all profits which he has earned since the income tax went into effect, or would two-thirds of it be considered as coming in before the income-tax law and one-third after?

A. D. Brokaw.—That is covered in the regulations, by an appraised valuation.

Paul Armitage.—I think if the property was purchased after March 1, 1913, and sold at a profit, that profit would be taxed. There might be a question as to what rate it would be taxed.

R. C. Allen.—Suppose you buy a lot today for $500 and a year from now it is worth $5000; there has been an increase of $4500. That is an increase of capital value. In essence, it is the same thing as putting back your current earnings into the equipment and development of your mine. In both cases there is an increase in the capital value. But in the latter case the earnings of the year are realized in money and taxed before they are reinvested. In the former case no tax is paid on the appreciation in value from year to year because the appreciated values are not realized in cash or equivalent income. Under the income-tax law since March 1, 1913, the road to earned surplus runs through the income account and no addition to earned surplus can be made except from income. Therefore, in the former case no addition to capital account is permissible on account of appreciated but unrealized value.

W. O. Hotchkiss.—How would that policy agree with the policy of revaluation on new discoveries in the mine?

R. C. Allen.—Formal claims have been made in connection with tax returns at Washington for the addition to capital invested of the value of current discoveries of ore. If the appreciation in value is not income, it cannot be capital, because before it can be capital it must be income. For the tax purposes it is neither income nor capital. Perhaps I can make myself clear. We will start at the beginning of the year with 10,000 tons of ore. In the course of the year we mine out 5000 tons but discover 50,000 tons. Is the newly discovered 50,000 tons to be considered as capital invested? There is an appreciation in the value of the property equal to the value of 45,000 tons of ore; but that value cannot be taken into surplus until it is realized in money, passes through the income account and pays a tax.

The point of view of the administrative authorities seems to me very logical; it is that the discovered ore is neither capital nor income. It may be added to capital only through income, *i.e.*, when the income is realized from the exploitation of the discovered ore. I have never been able to understand how you can add to capital in this way except through the income account. It seems to me that unless we abandon the basic principle of the income tax, we have got to consider the appreciation of value of property as income if we wish to consider it as capital. A more logical view is that it is neither income nor capital for tax purposes until it is actually realized.

The practical method of measuring the extent of gain or loss is through a transaction involving change of ownership. That is the ruling of the Department—it is a very reasonable thing. I would like to cite a case. If I remember correctly, an Alaskan mine was flooded by the sea a few years ago and is worthless today. A gentleman who owned considerable of the stock in that mine came into the Department one day. He told of the destruction of the mine and said that he had tried to sell his stock in Europe and in this country but that no one would buy it. "I have sustained a loss and want to write it off in my income-tax statement," he said. He was told that he must sell the stock even if he had to sell it for 25 cents before he could determine the extent of his loss. Is not that a reasonable ruling on that matter? It may not appear to be at first thought, but careful consideration will convince one that as a practical matter in tax administration the ruling is wise and necessary. Without it endless claims for indeterminable losses would be made.

William Kelly.—Is it necessary to make a sale before you can compute income or loss of income? You cannot consider income as a matter of appreciation of value unless it is sold?

R. C. Allen.—In the case cited the fact of depreciation is of course apparent before a sale of the depreciated property takes place but the amount of the depreciation in value may be measured only when the property is sold.

L. C. Graton.—One of the questions which interests me particularly is whether it will be possible, in order to handle the thousands of mines to which values must be assigned, to treat them in groups or whether we must burn the midnight oil, in every sense, and take them up individually. Some appear to feel that the group plan is unworkable. I believe if we look the matter fairly in the face we will agree that the group method is likely to be the one adopted.

We all realize that settlement, decision, of these taxes for back years is becoming, if anything, more important than the size of the tax. Many things that ought to be in motion are drifting or at a standstill because of present uncertainty regarding taxes. Now would you not prefer to have us group you together in classes and then put you by handfuls approximately where you belong, rather than to pick up each one of you separately, turn you over and over, determine and describe all your individual differences—and all of us die before the job is anywhere near completion? The group method then, maintaining necessary elasticity and with a willingness always to pursue special conditions as far as they ought to be followed, must be the procedure adopted.

One more word about interest rates. There are other ways of providing for hazard than through the rate of interest return; yet I am inclined to favor this last method, for this reason. In this complicated problem of taxation, we are liable to wander far from truth and sense if we do not apply this test to practically every step we take: is this the business-like, normal way of doing this thing? Now in the matter of investments, there is one standard, conventional, and thoroughly understood manner of reflecting risk, and that is through the interest return. We consider nothing so secure, so certain to protect principal and to pay interest as a Government bond, consequently nothing yields so low a rate of income. A good mortgage on real estate returns 5½ to 6 per cent. A mining company's bonds, as Mr. Brokaw has said, ordinarily yield less than the same company's stock because the bonds are better protected. I fear that if we were to depart from this recognized method, we might soon find ourselves adrift in unknown surroundings, and in any event our findings would be meaningless to those not conversant with the details of mining.

We can all sympathize with Mr. Hotchkiss over the difficulty of placing valuations on small, erratic, short-life properties, like the zinc mines of southern Wisconsin. Yet such properties change hands—indeed they are among the commonest types of mining property to be bought and sold. And each sale indicates that buyer and seller agreed on a valuation. How do they arrive at it? In this way: each tries his best to estimate what the property will yield, then each capitalizes that expected return; finally they dicker until any difference between them is eliminated. This method of capitalizing income is always the basis, and the aim toward which all valuation of fixed property approaches.

Mr. Hotchkiss, with his legislature geographically and objectively available and amenable, was able to avoid the difficult job of valuation by securing laws applying taxation on a different basis than that of value. But the federal government is probably too deeply committed to the principle of tax on incomes to be diverted from it by a difficult, though I believe not insoluble, problem in a single industry. Furthermore we must not forget that the conditions to which Mr. Hotchkiss has referred are not confined to mines of small output and short life. There are numerous great mines of long history, and no doubt long future life, which raise identical problems. I refer particularly to deposits in altered limestones.

H. S. Cooper.—How are you going to develop your valuation of them?

L. C. Graton.—It will be necessary, I believe, to put them up alongside mines whose value can be measured with greater certainty, then factors must be established that will harmonize the valuations for the two classes. It is difficult, but I believe not impossible.

Mr. Griffitts says that mining is no more risky than many other businesses. Although the mining industry is big, we might easily become narrow-minded if we confined our views to its limits. We want to keep in balance, not only within the industry, but in proportion and perspective to the other industries and activities of the country.

I want to say to you in all sincerity and seriousness, that if there be any merit or advantage in the plan of the Bureau of attacking the problem of mine taxation in a specialized, engineering fashion, there is one man more than any other who is responsible for it and to whom the credit for it is due; that is Mr. Allen. It is to be regretted that he could not have been persuaded to remain in the Bureau and run the ship.

H. M. La Follette,* La Follette, Tenn. (written discussion).—The far-reaching effect of the recent extraordinary application of the income tax, including the graduated rates applied as "excess taxes," involves many possibilities of indirect stimulation of both development and abandonment of large fractions of the values involved in mining operations. The dangers involved in its application to incomes from mines and other self-depleting industries are of gravest concern to all who appreciate the far-reaching effect on our country of the proper preservation, replacement, and complete reclaiming of the full values in these industries which are, at best, constantly sacrificed or largely lost by reckless or incompetent operations, without the added motive of avoiding excessive taxation or increasing the immediate output at the expense of the ultimate production.

Income produced as the fruit of mechanical invention, as well as of the wise and skillful administrative ability of the citizen far more than from invested savings or other funds as principal previously acquired, are necessarily subject to all income taxes, as much as are fixed incomes from vested funds or estates; but that fact is an added reason why "excess income taxes" need to be placed with caution and due care not to carry such levies to a confiscatory limit. The only justification of "excess taxes" is the absolute necessity of the state for self-preservation. While the principle of taxation of incomes is possibly the most just rule applicable to taxation and, by the adoption of the Sixteenth and Eighteenth Amendments, is to become permanently the foundation and chief source of our Government revenues, there is reason to believe that some tax legislation and administration are being subtly advocated and applied not so much for the true and wholesome needs of the state and the protection of constitutional rights and liberties as for the establishment of socialistic fundamentals and doctrines of virtual confiscation of property.

I believe it is a question of supreme moment to patriotic Americans whether the socialistic tendencies of a small but vociferous minority, promulgated through what Col. Henry Watterson has so well described as "our chattering ragtime press," are not in danger of securing through tax laws, passed under the guise of patriotism and a claim of "social reform," a misapplication of the ends of the state and creating political conditions in direct conflict with the letter and spirit of our constitution and the fundamentals of our republican government. We must never forget that our Government and Constitution were established for the equal protection of the lives, liberties, and properties of all our people, that the Fifth Amendment expressly declares that none "shall be deprived of life, liberty, or property without due process of law; nor shall private property be taken for public use, without just compensation;" while the Fourteenth Amendment explicitly makes it the duty of every state to protect equally the life, liberty, and property of every citizen. So, in the application of the income tax to mineral production, there must be considered the general principles involved, the urgent and reasonable necessity of the revenues to be thus derived, and the grave danger that excess taxation laid upon mineral production may either destroy, to some extent, the source of income or lead to the permanent loss of extensive deposits by not conserving them as a whole and, so to speak, making the rich sections pay for the mining and saving of the lean sections which in many if not in most cases, comprise the larger part.

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* President and General Manager, La Follette Coal, Iron and Railway Co.

Substantially all conditions that govern assessments of mineral property for direct or local taxation should be considered in determining the method of applying the income tax, as the author has pointed out. Distinction must be made between the ownership of land and "mineral rights." In seven states coal-leaseholds are specifically exempted from taxation; in eight states they are, by peremptory statutes, made a base of taxation more or less specific. In Wyoming, the law fixes tonnage values of 60 to 95 cents based on quality of all production. In Kentucky, the only direct tax laid on oil property (production and improvement) is one-half of 1 per cent. of the market value of oil as shipped, payable to the county. Thus at the rate of its current receipts, Lee County will receive within this calendar year over $100,000 of oil taxes. In two states, laws providing specific taxation of standing timber had to be repealed to prevent the complete denuding of their forests. The assumption that coal is of uniform thickness and quality, and that an average may be reached by estimation, has often proved outrageously unjust. In the anthracite region of Pennsylvania, assessments per foot-acre have varied from $9.84 to $250 and even $300 for direct taxation of faulty territory that in some cases yielded gross less than the cost of mining. If the enterprise of operators in drilling and testing coal land invites excessive assessments, the inevitable result is that such enlightening tests are omitted and the operator relies on the almost universal legal rule that the lawful value of untested mineral land is the cash price at which the land can certainly be sold.

My own experience has been chiefly in the field of iron ores and bituminous coal. The great bulk of our iron ores consists of hematite ores, red and brown, which have always shown wide varieties in quantity and quality, especially the brown ores which notoriously fail in a majority of cases to yield on the average more than a small fraction of the quantity indicated by the surface exposures.

In the case of coal values, the rule generally accepted and practised in a majority of the coal states is that laid down by the Supreme Court of Pennsylvania (Pa. S. C. R. 229, p. 470) as follows:

> Its market value is its fair selling value for cash, not payable as royalty strung out through a long series of years, but payable at the time or as soon thereafter as the value can be determined * * * * The question is not what earning power coal lands may develop in the future, but what they are actually worth in the market at present.

But while unpaid royalties are not a reasonable basis for direct or local taxation, in determining income taxes whatever royalty deduction is allowed a leaseholder should be credited uniformly and be deducted from the "net income" of the operating owner in fee.

That the best ultimate interest of the public and of the future supply and consumption of any mineral will be served by virtual omission of attempted valuation of invisible mineral believed to remain in the ground, and by the assessment in some form (whether for local taxation or federal income taxes) on the year's output at actual sale prices received at the mine, with deduction for full mining cost, depletion, depreciation of equipment, and other deterioration, if any, would seem to be self-evident.

All must agree that excessive assessment or overtaxation (whether direct or upon income) of many extensive deposits of coal and iron ores, which should ultimately yield an enormous production to the great benefit of the country at large even if with small profits to the operators, must inevitably lead to the destruction and permanent loss of the greater portion of such deposits, by the mining of the most valuable portions and such fractional parts or veins as can be most cheaply mined, leaving the great bulk of leaner mineral in the ground under conditions that prohibit profitable mining.

In applying income taxation to products of such properties considerations must also be given to the exceptional provisions in the constitutions and laws of the particular states. Thus in the constitution of the State of Tennessee there is a specific prohibition of taxing any Tennessee product while it remains in the possession of the original producer. This exempts from taxation virtually nearly all personal property on the farm as well as pig iron, ores, coal or coke on the yard, felled timber, manufactured lumber, etc., so long as owned by the producer. Combined with the further exemption to all taxpayers of $1000 of personal property, of whatever kind, these provisions in the original constitution of 1797 have made it impossible for "modern reformers" to displace that historical instrument, though frequently attempted, except by minor amendments that do not affect these provisions that favor all producers, miners, manufacturers and farmers alike. After all, it is not a bad constitutional provision! But it must be recognized as a general rule that only the most fair and liberal application of heavy income and excess profit taxes upon mine products will avoid the grave danger of driving the mine operator in many cases to "hog" his mine, by working out only the richer sections and leaving all leaner material in such shape that it will be permanently lost.

Alan M. Bateman,* New Haven, Conn. (written discussion).—It is gratifying that the necessity of the application of geological principles has been recognized by the Bureau of Internal Revenue both in the appointment of geologists to its subcommittee and in the recognition that the value and life of a property depend not only on the measurable ore gaged by conventional limitations and rules, but also on the mode of occurrence of a deposit, its behavior, and its possible change in grade of ore. It is so evident that the value to the owners of a particular property rests largely on the ore that may be expected beyond the limits of workings, that this feature should justly be allowed in the valuation of a mine. To estimate properly the amount of ore that may be expected beyond the limit of workings necessitates a knowledge of the occurrence and behavior of an individual deposit, and for this no set rules will apply.

The author has set forth clearly the special features, from a taxation standpoint, that characterize the mining industry. If, however, it is admitted that the peculiarities and risks inherent in mining call for a larger return on the investment than in most other industries, and that this larger return includes compensation for insurance for risk, it is unfair to tax that portion of the return which is to compensate for risk or insurance. If it is the desire of the Bureau that compensation for risk be given to the mining profession, this compensation should be real, and not an apparent one subject to taxation. This object might be gained by allowing a certain percentage of the annual income, or "per cent. risk," to be deducted as the cost of insurance or risk, in much the same way that depletion is deducted. Then, only a normal interest return need be taken, which would result in the computation of normal valuation with normal depletion allowance and consequently normal tax. The insurance risk or compensation would then not have to be taxed, and fairness to both parties would ensue. Thus, the fundamental interest return for mining would not have to be different from that of other industries, and the recognized peculiarities of the mining industry would be offset by allowable deductions such as depreciation, depletion, and compensation for risk. When the Bureau fully appreciated the merits for depletion in mining, allowance for it was incorporated in the tax laws, and now that the merits of compensation for risk are recognized, it also might be incorporated as an allowable deduction.

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* Editor, *Economic Geology*.

The meat of the paper is the method to be employed for mine valuation, and the suggested "set of factors" to be applied to those mines that cannot be directly so valued. The present-value method of computation for mine valuation, so strongly urged by the author, is certainly the most direct professional procedure. Much difficulty and hardship, however, would attend its application to many mines, which are just being initiated as producers or whose life cannot be determined. In the large low-grade disseminated deposits or the porphyry coppers, the method is directly applicable. But in the case of mines whose ore supply comes from irregular replacement deposits in limestone, or contact metamorphic deposits, or others of known irregularity and indeterminate periods of life, no set rules could apply.

The suggestion to establish a series of modifying factors for such mines, so as to adjust them to those that can be valued by the present-value method, would necessarily involve a certain grouping of mines to which a common factor would apply. Obviously, by this method, a factor could not be established for each individual mine. Inasmuch as no two mines are alike in all respects a given factor might work hardship on any one of a group. The ore deposits of two mines of a group may have similar modes of occurrence and yet have sufficient minor differences that the factor might not equitably apply to both, and if one of them suffers a hardship the procedure is bad. I greatly fear that any such set of factors would be sure to work injustice to some and for the continuance, encouragement, and stimulation of the mining industry such a condition should be avoided. Who could tell but that his particular property might be so affected? One would need to be fairly sure of the nature of a new property before investing considerable money in it. As most mines are developed from prospects which commence shipment as soon as can be arranged, the tendency might be to discourage somewhat the opening up of small properties that, by development, mihgt yield large profits. This might apply more particularly to those prospects upon which greater chances must be taken of their development into mines. Another tendency, though remote, might be to promote a higher estimation of ore tenor than would otherwise be done, which if not justified by more complete development would shake the public confidence in the mining industry.

If the tonnages, tenor, and length of life, as given for the determinate porphyry mines, are to be accepted for their valuation, why could not the same data, obtained with the greatest accuracy possible, also be accepted (instead of arbitrary factors) for the irregular deposits with their less definite data but greater risk?

The author gives no idea as to the nature of such factors or whether, when established, they are to be yardsticks known only by the Bureau of Internal Revenue and applied by it, or are to be public information and of such determinate character that an engineer would be able to use them as the basis of his opinion in the examination for purchase of mining properties or prospects. Obviously the taxation of a large number of properties is to hinge on such factors, and little can be known of the tax laws as applied to them until these factors are clarified.

If modifying factors are established, care should be exercised that, in addition to amount of development workings, length of life behind a property, depth, and other similar features, cognizance is taken of the geological features. It is a matter of considerable importance in the expected life of a property whether it is an irregular replacement deposit in limestone, contact-metamorphic deposit, fissure vein, or sedimentary bedded deposit, etc. Also, the prospective life of a tungsten mine occurring as an irregular replacement deposit in limestone differs essentially from deposits of copper or gold or lead of similar mode of occurrence. Or again, the grade of the ore in different mines of the same metal and same mode of occurrence makes considerable difference in the prospective life. Copper mines in irregular replacement deposits in limestone with smaller high-grade orebodies may have their life or value suddenly increased or decreased by the accession or termination of high-grade orebodies. Such features must needs be considered if the factors are to be fair and impartial.

R. B. Brinsmade, Ixmiquilpan, Hgo., Mexico (written discussion).—In my discussion of the papers of Doctor Raymond[15] and Messrs. Gibson[16] and Godson,[17] I have endeavored to demonstrate that our system of unrestricted private ownership of land is incompatible with equality of economic opportunity and, therefore, subversive of our democratic political institutions. In Mr. Graton's paper, the same land problem is important, though camouflaged by the failure of the federal income-tax laws to distinguish between land and capital, in the requirements for valuation, or between earned and unearned incomes in the levying of taxes.

I disagree with the statement, on p. 1194, as follows:

> The majority of mining ventures are failures. A multitude of disappointments is relieved by occasional success. Were it not for the chance of great rewards, all the

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[15] Our National Resources and Our Federal Government. *Trans.* (1912–13) **44,** 633.

[16] Principles of Mining Taxation. *Trans.* (1919) **61,** 709.

[17] Uniform Mining Law for North America. *Trans.* (1919) **61,** 712.

> vitally necessary but mainly unremunerative work of exploration would not be undertaken. In contrast with nearly all other fields of endeavor, it is altogether unlikely that, on the whole, the gains in the successful mining ventures suffice to offset the losses in the unsuccessful.

In other words, the author holds the popular view, acquired by small investors in wildcat companies at a ruinous cost for the experience, that mining is not self-sustaining but a sort of industrial mendicant that must be supported by the profits of the other industries like agriculture, commerce, or manufacturing. But, if so, how could the mineral industry pay from its own profits a federal tax of $766,000,000 in 1917, and have increased its output to a total of $5,000,000,000 in 1917, as compared with less than one-half that value in 1900? Investors are not accustomed to throw, for very long, good money after bad or to continue to increase a production that must always be sold at a loss. Next the author says:

> The argument is sometimes advanced that the great profits made by some individual mining companies should, in large part, be taken through taxation by the Government so that the people as a whole may participate in the benefits that the nation's soil and rocks afford. If such policy should be followed, however, there would go with the collection of these profits by the Government the moral and economic obligation to perform all that vast ratio of profitless exploration, on most if not all of which depends continued discovery of profitable mines.

This sounds plausible until one considers that the Government must live and that its only means of support is taxation. So that the problem is not whether mining companies shall lose their profits or keep them but as to which companies shall pay the share of taxation to be necessarily borne by the mineral industry. If taxes are not collected in proportion to profits, they must be collected on some other basis; such as, gross output, number of workmen, or capital expenditure. Is anything more discouraging to exploration than the addition of taxation to other expenses? Is anything more stimulative to the necessary search for new orebodies than the relief from all taxation until the ore has been found and is being sold at a profit? No! the present huge profits of a lucky few who own the rich orebodies of the country mean a corresponding burden of costs, under the general property and indirect taxes, on the many enterprises which are either in the development stage or producing at little or no profit.

Value of Unearned Increments.—The author, on p. 1196, in comparing the possible increments of gain on a town lot bought by A and a mineral tract bought by B, each at a price of $1000, says:

> What would happen to him (B) if he, like A, simply held on and did nothing? Even though profitable mines might develop all around him, and thus possibly cause

an appreciation in the value of this tract, such appreciation would be only speculative. And as a matter of fact, few mines increase in value to any extent through the non-activity of their owners.

While frankly acknowledging that if A's town lot increases in value by the growth of the city from $1000 to $100,000, the difference of $99,000 is an unearned increment,[18] the author asserts: (1) that mineral lands differ from town lots in having practically no unearned increments in their value; and (2) that even when their value does appreciate through the activity of the community, such an increase would be merely speculative. As to the second assertion, it is clear that if B can cash in and get $100,000 in real money for his mineral tract, his gain of $99,000 is just as genuine and just as unearned as the $99,000 A gains from his town lot. For the negation of the first assertion, it is only necessary to consider the outstanding facts in dozens of big mineral districts. The clearest cases are where the landowner does positively nothing but "sit tight;" while the would-be operator, at his own expense and risk, must find the mineral, develop and equip the mine, and ever thereafter pay a royalty on everything produced. This condition prevails for the United States in practically all the oil and gas fields, in most of the bituminous coal fields, in the zinc fields of Wisconsin and Missouri, in the Lake Superior iron fields—especially those of the Mesabi and Cuyuna ranges—and nearly everywhere else when the underground minerals can be located by surface borings in advance of extraction. If the author believes that the mines owned by this class are few or that they have not "increased in value to any extent through the non-activity of their owners" let him compare their yearly revaluations for local taxation, for several decades past.[19] Should he carry these comparisons far enough, he will doubtless discover that the mineral properties owned by inactive landlords who are permitted to collect "royalty" (king's or lord's tribute[20]) from the actual operators vastly exceed in number and value those owned by such venturesome explorers as B of his example.

Relation of Natural Resources to Industry and to State.—Before dissecting the paper further, I beg leave to make a few preliminary explanations from theoretical political economy on the relation of natural resources to industry and to the state. As I showed in 1913, all the factors necessary for producing wealth can be reduced to three: *land*, *labor* and *capital;* while the produced *wealth* is thereafter distributed among these three factors, labor receiving the *wages*, capital the *interest*, and land the *rent*. These seven italicized terms, in their strict economic sense,[21] can graphically be interpolated by the accompanying Trinitarian diagram.

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[18] H. J. Davenport: Extent and Significance of Unearned Increment. Am. Econ. Assn. *Bull.* (1910).

[19] R. B. Brinsmade: Mr. Ingalls and Walkerian Economics. *Min. & Sci. Pr.* **119,** 630.

[20] E. P. Cheyney: "Social and Industrial History of England."

The popular terms profits and income are complex rather than simple items of distribution; each term can be subdivided into interest, wages, or rent, according as the wealth represents the reward paid for the use of capital, of the labor, of management, or of land. The term insurance can readily be classed either as interest or wages, according as it represents the extra cost for capital or labor added to cover the risk of their use. Similarly, the popular terms for the factors of production, such as real estate, railway, mine, factory, can, after a little consideration, be easily subdivided into the fundamental factors of land, labor, and capital.

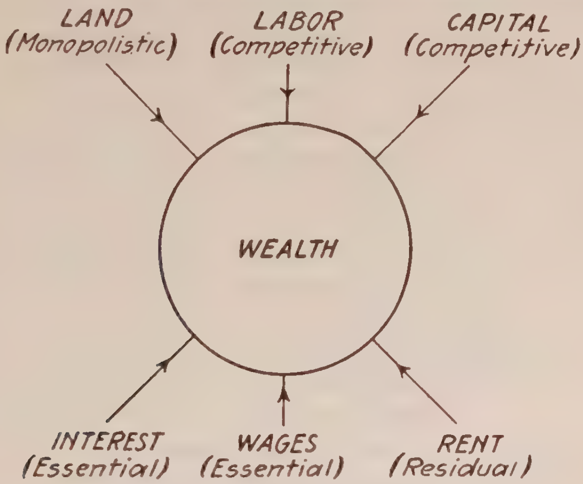

FIG. 1.—TRINITARIAN DIAGRAM.

As land, labor, and capital share among themselves the whole output of wealth, the share of one of the three productive factors in the output of any enterprise can only be increased at the expense of one or both of the other two. Also, the share of wealth that one factor can obtain depends entirely on its relative strength in its struggle with the other two factors. As productive operations grow in size, it is evident that any one of the three factors that can be indefinitely increased in quantity is subject to competition, and consequently its share of wealth tends to be cut down to the minimum necessary for the factor's survival. As both labor and capital are of this competitive nature, wages tend constantly to approach the lowest living wage necessary to keep up a labor supply of the required quality, and interest tends to approach the minimum rate needed to encourage the saving of sufficient capital. Land, on the contrary, being fixed in quantity, cannot be increased as production grows; it is therefore the only naturally monopolistic factor of the three. The final result of the competitive struggle in distribution between land, labor, and capital is that the monopolistic factor, land, takes the whole residue of wealth from any operation after labor and capital have been paid their essential living wages and interest.[22]

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[21] J. E. Symes, in Chapters 1 and 2 of his "Political Economy" thus defines these terms: "Land includes such material gifts of nature as can be monopolized; labor includes all manual and mental human exertions used to produce wealth; capital includes such material products of labor as are devoted to the production of other objects of desire; wealth includes all the material products of human labor that arouse desire. Assuming free competitive conditions, wages is the share of wealth secured by laborers, interest is the share secured by capitalists, and rent is the share secured by the proprietors of natural objects and forces."

As rent is a residue, not an essential payment to sustain production, like wages and interest, lands may be continuously worked which yield no rent; and, as a matter of fact, it is probable that the area of rentless lands under exploitation on the globe is greater than that of those yielding rent. For superior lands, the economic rent—as distinguished from the popular term rent, which generally includes interest and often wages too—is easily calculated from Rickardo's well-proved Law of Rent.[23] For mineral lands, the annual rent may be considered as the net value in the ground of the annual output of mineral. The greater the value in the ground of the mineral, the greater is the output of annual wealth that can be obtained by the productive activity of a given quantity of labor and capital. For the greatest national advantage, therefore, it is important that there exist no legal obstacles to the working of the richest mineral deposits known in any epoch, leaving the poorer classes for the future when improved methods may offset their greater leanness. Speculation or forestalling, as usually practiced, may be defined as the legal withholding from use of needed superior lands for individual profit, thus forcing part of the nation's labor and capital to resort to inferior lands where they consequently produce less wealth. This also raises the price of the output, for it artificially lowers the quality of marginal land[24] whose cost of production fixes the market price for the whole output of the commodity in question. Mr. Baer's theorem, quoted by Mr. Norris, expresses the last truth in different words.

While land speculation only indirectly affects the quantity and price of commodities, by lowering the quality of marginal land, land monopoly is able not only to control the quantity but also to fix the price of its output directly. If the monopoly is local, it can only fix prices as high as those of an imported product; if it is complete, it generally fixes its price at "what the traffic will bear," or where it will produce the maximum continuous income. As examples of practically complete mineral monopolies may be mentioned: the diamond trust of South Africa,[25] the anthracite trust of Pennsylvania,[26] and the potash trust of Germany, before the war.

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[22] J. S. Mill: "Political Economy," Book on Land.

[23] Symes ("Political Economy," 29) gives: "The normal rent of any land is got by deducting the produce of land on the margin of cultivation from the produce of the land in question."

[24] Marginal land may be defined as the leanest land that has to be worked to supply the current demand for its product. It is necessarily rentless and its expenditure for labor and capital fixes the price of the commodity yielded. Marginal land constantly fluctuates in quality; naturally as the current demand varies and artificially when affected by the operations of land speculators and monopolists.

The state is a silent partner in all productive enterprises as it furnishes more or less of its land, its capital, and its labor to assist every one. Accordingly taxes—the share of wealth apportioned to the state—may be classed under the heading of wages, interest, or rent, as they represent a payment made for the use or the labor, the capital, or the land of the state. Any general tax levied by the state on private capital or labor is bound to increase the essential payments for the sustenance of capital and labor (interest and wages) and, therefore, to increase the price of commodities that is set by the cost for capital and labor on marginal land. But a tax on the third, or residual, item in wealth distribution—rent—cannot raise prices, because rent is not a factor in price-fixing, marginal land being rentless. An increased tax on rent means, then, that the state gets more and the landlord less; while the increased tax on either interest or wages means, finally, its shifting to consumers in the form of higher prices for commodities.

Allowance for Depletion.—From this theoretical foundation, I will consider the author's views on depletion; he says:

> It has also long been recognized by economists as well as by mine owners and mining engineers, though not by every one, that mining is an industry of wasting assets and that the apparent income for a given year's operations is not true profit but part profit and part return of capital. * * *
>
> The entire conception, which some have seemed to feel is a complex and mysterious affair, becomes of utmost simplicity when the ore deposit is regarded merely as a great supply of raw material, the cost of which, along with the other costs, must be inevitably covered by the selling price of the finished product. Provision for this is made under the term "depletion," which, recognizing that ore, when once extracted from a mine, can never be replaced, aims to compensate for the inroads into the value of the mine thus occasioned by affording the money equivalent of the value in the ground, of the ore removed. Thus at any time, value of ore remaining plus the depletion fund accumulated to date should equal a constant, which represents the value in the mine of all the ore originally estimated to be present.

From the foregoing, it is clear that the author follows the unscientific custom of lumping land and capital together into one term "capital," for he makes no distinction between the land of a mine, represented by its raw mineral deposits, and its true capital in the form of machinery, buildings, underground developments, working funds, etc. Popularly, a raw orebody is a "supply of raw material" and therefore capital; but technically it is classed by political economy as land, along with wild forests, wild naturally enclosed fish or animals, and other "material gifts of nature that can be monopolized."[27] Provision for depletion then, as explained by the author, means that the mineral-land values of a property are to be kept constant by building up a cash sinking fund that will always represent the underground value of the mineral already extracted.

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[25] Gardner Williams: "The Diamond Mines of South Africa." N. Y., 1906.

[26] Scott Nearing: "Anthracite."

On this technically false foundation, the author proceeds to discuss "mining as a continuing industry." The advantage to the nation of perpetuating the life of our best mining corporations, by providing them with new mineral deposits as fast as the originial ones give out must be patent to any one, but I differ with him as to the proper way to accomplish this object. He first claims that the orebody of a mine is analogous to the leather supply of a shoe factory, and that just as the leather must be reckoned as a working expense and a fund built up for its replacement, so should it be for the orebody. But, economically, there is no analogy, because leather is capital and an orebody is land. Normally, the cost of capital enters into fixing the price of commodities, but the cost of land does not, for marginal land—where prices are fixed—is rentless.[28] Hence to allow a national combination of mine owners to figure in the land value of their orebodies as part of the cost of production would mean the overturning of competitive price fixing on marginal land and create a more or less monopolized market where rent would be added to the normal price and the consumer be mulcted. But, granting that such combinations might affect only a fraction of the mineral industry and that most prices would continue to fluctuate freely, irrespective of the Government's system of bookkeeping, the allowance for ore depletion as "cost," instead of "income," would still mean the entire exemption of mineral land values from the federal income tax, and the inflation of mineral prices by increasing the cost of capital on marginal land.

Under the heading "Point of View," the author says:

> The latest law clearly recognizes the wasting industries. The mining industry is thus among the few to which distinctive treatment of any kind is applied. The idea has gained hold in the minds of some engaged in other industries that mines are thus accorded special favors or advantages. This view, however, is entirely erroneous. For while certain allowances are made because of the exhaustible character of mining assets, they are only such as fairly provide for including cost of raw material in total cost of production, and serve only to remove what, in their absence, would be a harsh and unfair discrimination against mines and other wasting industries.

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[27] J. E. Symes: *Op. cit.*

[28] See footnote 24.

An income-tax law that, unlike the British practice, assesses earned and unearned incomes on an equal basis is certainly undemocratic enough; but when its application involves the exemption of the unearned income of mineral-land owners, it becomes not only undemocratic but grossly unfair to other industries, which have to pay on all income whether from capital or land value. Therefore the Supreme Court, in its refusal to allow depletion of minerals to be deducted from taxable income in 1913, showed itself much more competent to protect the public and insure equity among taxpayers than did the Congresses of 1913-19. No, the depletion allowance is not equity but a harsh discrimination for the benefit of the few owners of bonanzas—directed not only against all industries so unfortunate as not to be wasting but against a vast number of mine operators, whose ore is so lean as to have little or no value in the ground and is consequently entitled to no tax exemption.

And that my conclusion thus demonstrated from simple economic theory is also sustained by expert tax authority is proved by the following quotation from Mr. Gibson:[29]

> On the other hand, it is true that mines are sometimes profitable on a scale far beyond that of an ordinary commercial business. I need not stop to multiply instances. The copper mines of Michigan, Montana, and Arizona; the silver bonanzas of Virginia City or Cobalt; the gold mines and diamond diggings of South Africa; and the nickel deposits of Sudbury will come to your minds. Returns such as accrue from deposits like these are the sustaining force of the mining industry and enable it to maintain a vigorous existence despite countless individual failures. Nevertheless, if deposits so valuable pass into private ownership, there is nothing unreasonable in regarding them as charged with a lien in the form of a demand that a fair share of the profits derived from working them shall inure to the public benefit.
>
> In applying the principle of taxing profits or net earnings, it is, of course, implied that all the usual and necessary costs of operation shall be deducted from the gross proceeds. These include wages of labor, superintendence, power, explosives, timber, transportation, expenses of marketing, administration, and similar charges. Depreciation of mine buildings and plant should be allowed for on a basis that will amortize their cost during the lifetime of the mine. Mining machinery is valuable only so long as the mine is in operation; when the ore is exhausted, it rarely has more than a scrap value.
>
> Initial capital really invested, carefully distinguished from water, should be exempt from taxation, as until it is returned either in dividends or some other form, profits in the strict sense of the term cannot begin. By initial capital is meant the money invested in purchasing the land from the government and in opening up and equipping the mine. It is apparent that should the property be subsequently sold for a larger sum, it would not be equitable, so far as the state is concerned, to treat the additional cost to the new owner in a similar way, for the increase in value simply represents a capitalizing of future profits, the express object of taxation.

In the last two sentences, Mr. Gibson expresses the democratic view in a nutshell; and from the whole quotation it is easy to figure, as I did in discussing his paper, that the only profits left for assessment for taxation would be those due to land rent, or the value of the mineral in the ground. In exempting all true capital from taxation and taxing only the orebodies, Mr. Gibson is following the eminent economists Shearman[30] and Fillebrown[31] and standing at the opposite tax-pole from Mr. Graton, who advocates exempting the landlord's orebodies, and only taxing the profits due to the operator's capital and labor of management. My critics may object to this, that Mr. Gibson also believes in exempting the "money invested in purchasing the land from the government;" but this objection has no practical weight, when one considers that our eastern mineral deposits cost the first private owners nothing, being thrown in with the surface rights while, until recently, all our western mineral deposits were either sold by the government at $5 or less per acre or given away in land grants to railway and other promoters.[32]

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[29] *Trans.* (1919) 61, 641.

With the administrative section of this paper, I am mainly in agreement, also with the summary of the methods of mine valuation. So, as I have hitherto fully expressed my views on the latter subject,[33] I will skip at once to the consideration of the discussion of Mr. Norris, whose viewpoint seems to favor the owners even more than Mr. Graton's. On p. 1226, his explanation of the law of 1919 shows how the exemption for mineral depletion has been extended to include the lessee as well as the landlord. This is an important change and means a large additional loss of income-tax revenue to the government, for there are few long-term leases on a fixed royalty where the current value of mineral in the ground is not in excess of the prescribed rental to the fee-owner. He says:

> The 1910 census returns showed less than 2 per cent. return on invested capital for all bituminous-coal operations, not including depreciation, and only 4½ per cent. on anthracite operations under the same conditions.

But as these census returns are calculated on the unscientific basis of lumping land and capital values, they really mean nothing as to the yield of the genuine, as distinguished from the fictitious, investment. For instance, if an entrepreneur buys a tract of coal from the state for $2000 and gains a net income from its exploitation (after spending $98,000 in development) of $10,000 a year, he is getting a return of 10 per cent. on his genuine capital. Yet he may easily call his coal tract worth $102,000 instead of $2000—capitalize his enterprise for $200,000 and then figure his return as only 5 per cent. on the total investment, of which half is now the watered stock proscribed by Mr. Gibson.

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[30] Thomas G. Shearman: "Natural Taxation." 1898, Doubleday.

[31] C. B. Fillebrown: "Principles of Natural Taxation." 1917, McClurg.

[32] Gustavus Myers: "History of American Great Fortunes." 1910, Kerr.

[33] Calculation of Mine Values. *Trans.* (1908) **39**, 243.
Valuation of Iron Mines. *Trans.* (1913) **45**, 322.
Valuation of Coal Land. *Trans.* (1913) **47**, 143.

Necessity of Obtaining New Ore Supplies.—On p. 1203, the author says:

> In a comparatively short time, the question of finding the new supplies of ores to meet the world's needs is going to become pressing, if not acute. In a word, the next great purchases of raw material for the mining companies are going to cost far more than the supplies they are now using up.

Evidently he believes this gloomy future to be due to natural causes, like the exhaustion of orebodies, and therefore inevitable, but he may change his creed as to the sole guilt of Nature if he will follow my analysis of Mr. Norris's revelations on p. 1227 which are:

> The author's statement that the capital invested must be returned by reserves for depletion and depreciation is absolutely sound and fair. Such reserves should be sufficient to replace the exhausted coal at the going price at the time of exhaustion, not merely to amortize the cost or the estimated value March 1, 1913, the date at which the first income-tax law happened to be passed.
>
> That the coal business is and should be a continuing industry is, of course, unquestionable. The author indicates that this continuation of the invaluable organization must be by the purchase, from time to time, of new mines at the then going price. In the case of the coal industry, this long and indefinite life is more reasonably attained by the control of territory or tonnage not available at the present time, either by reason of its distance from shipping facilities, its location in relation to possible development, or its depth rendering its present exploitation financially impracticable. Such reserves should not be considered as available coal, which, in fact, they are not, but should be carried as "reserves unavailable." * * * They should not be considered in calculating the present value of coal property.
>
> Such deposits have no present value and if held as future reserves by present operating companies, such holding should be considered as a public benefaction and should not be discouraged by the imposition of taxes. Coal now commercially unavailable held by well-organized operators promises the continuation of a valuable organization to the ultimate benefit of the body politic.

Not satisfied with the depletion exemption on the present value of orebodies, Mr. Norris eagerly takes the author's hint, on p. 1201, and proposes boldly that the exemption be calculated "at the going price at the time of exhaustion." In this way, not only will the present mineral-land rental be exempt from the income tax, but a considerable extra sum will be subtracted from the profits due to capital and the labor of management. Thus, a mining company operating its own rich orebodies would soon be entitled to figure its exemptions so as to pay no income tax whatever.

But Mr. Norris's most astounding proposal is in the last paragraph, where he claims that the possession of unlimited quantities of submarginal coal lands by present companies "should be considered as a public benefaction" and go untaxed. While it is possible that his experience has been limited to coal companies organized for public benefaction instead of profits, such enterprises fall outside the scope of political economy whose working mottoes are: "self interest," and "the maximum reward for the minimum exertion." From this practical basis, it is clear that Mr. Norris advocates the encouragement of land speculation for "the ultimate benefit of the body politic." As I demonstrated in explaining economic theory, land speculation decreases the production of wealth and increases the cost of commodities, when it affects superior lands. While Mr. Norris's first proposal would, at present, affect only sub-marginal lands, there is no telling how soon these lands would rise to the marginal and superior classes (as consumption increased) were they public property and open to the adventurous explorer instead of being patented.

But that Mr. Norris sees no public danger in the fostering of forestalling of any quality of coal land, is shown by his enthusiastic approval of the valuation system for local taxation of the Virginians, which he outlines to be:

A. Coal immediately available, taxed at a high valuation = the acreage mined out the preceding year.

B. Available coal 10 years; A times 10, about one-half of the A valuation.

C. Available coal deferred, A times 10, at a fairly low valuation.

D. Unavailable coal, all other holdings, at a nominal valuation.

Classes A, B, and C provide for the valuation of an acreage sufficient to provide a working reserve for 20 years ahead, on the present scale of production of any company, and is perhaps as fair an assessment as can be obtained under the clumsy general-property tax system.[34] The "nigger in the woodpile," therefore, is concealed in Class D whose "unavailable" coal, unlimited as to quantity, is assessed at nominal price. Can a state tax-commission of politicians, ignorant of engineering, properly classify "available" coal; and, when dominated by the land ring, would it if it could?

Enough has been said, I hope, to convince the author that his plan for insuring a continuous supply of cheap raw material for our mining organizations will only aggravate their present difficulties. The more the active orebodies are subsidized by depletion exemptions, the dearer will they become; and the more the idle lands are untaxed, the greater is the profit in keeping them idle for a future rise. West Australia penalizes its mining parasites by a heavy acreage taxation and compulsory work,[35] and Mexico by both of these and, in her petroleum field, by a 50-per cent. tax on royalties in addition.[36]

W. Spencer Hutchinson, Boston, Mass. (written discussion).—The following observations are the result of an attempt to arrive at a classification of risks as an aid in determining discount rates, based on an examination into the nature of mining risks. The following three divisions are proposed. Risks related to stage of development, margin of operating profit, and estimate of life of mine.

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[34] Lawson Purdy: "Burdens of Local Taxation." 1901. N. Y.

[35] A. C. Veatch: Mining Laws of Australasia. U. S. Geol. Survey *Bull.* 505.

[36] R. B. Brinsmade: "Latifundismo Mexicano," Chap. 12. Libreria Bouret, Mexico City.

Stage of Development.—Risk bears an important relation to the stage of development. I mean by this that risk is greater in a young mine in which the determination of the extent of the deposit is only partly made, in which the mining method is not yet proved by actual operation, in which metallurgical treatment is not yet established, and in which operating organization is not yet created. The risk diminishes to its minimum when these things are done; in other words, when the enterprise is on a fixed operating basis.

It is certain that a higher discount rate should be adopted in making valuation in one condition than in the other. Thus, Hoover's rule that 7 per cent. above provision for capital return is an absolute minimum, and Hoskold's suggestion of rates as high as 25 and 35 per cent. are reconciled with Finlay's adoption of 5 per cent. for use in valuing the fully established businesses of copper and iron mining in Michigan. The higher rates are conceived to apply to mines in the stages of least development, which will still permit of valuation by the methods under discussion.

Upon reflection, it seems fair to view the sale of a property in the development stage, and still requiring equipment, as in the nature of a "forced" sale. The owner lacks the capital for its development and equipment, otherwise he would keep the property himself and reap the rewards of his discovery. Under these conditions he must sell the property for the best price he can get. If he holds it, he derives no income from it whatever and the purchaser clearly enough makes the best bargain he can. It is reasonable, under these conditions, to find relatively high discount rates used in valuation. If, on the other hand, we consider a fully equipped mine, the condition is totally different. The owner, if he sells at all, and it is less frequent that we have sales of this class of property, exacts of the buyer a price equivalent to the full value and, unless he can get it, he naturally will keep the property. Under these conditions, we may expect to find lower rates of discount used in valuing.

It has been the duty of the engineer in the past more often to appraise mines in the state of partial development and preceding equipment than in the period of established and profitable operation. But it does not appear to be the purpose of the Government to apply methods that would be equivalent to valuing on the basis of a forced sale, and the discount rates suggested by Hoskold would be absurd if applied to the determination of the value to the permanent owner for the production of minerals.

Margin of Operating Profit.—Risk bears an important and definite relation to the margin of operating profit. This principle results from the theory that the ore in a mine represents a stock of raw materials purchased under the necessary and natural conditions of mining to afford a supply over a considerable number of years. The miner is unable to do as the manufacturer does—purchase his raw materials season by season. He therefore cannot know, as the manufacturer does, at what price he will sell his product. The risk is greatest as costs rise nearer the unit price of the product and as the margin of profit is small. The risk grows less as operating costs grow smaller and the margin of profit is large.

Consideration of the relation between margin of profit and risk will show that a small margin of profit often influences the choice of mining methods. Thus, on the theory that it is better to take a chance and make a profit rather than adopt safer mining methods and not make anything, a mine might conceivably be brought to an unworkable condition; this is what seems to have been the case with the Atlantic mine. It was not the even and continuous mineralization of the Atlantic vein that caused the operators to mine without leaving pillars; it was the compelling necessity of keeping down costs. If this reasoning is correct, the risk that wrecked the Atlantic mine was a narrow margin of profit rather than a hazard of operation. If the ore had contained more copper, a safer mining policy must have been adopted on the theory that it would be better to make a moderate profit and preserve the property than to make a larger profit and risk the loss of the mine.

The risk of lower prices of product and declining business that may be caused by discoveries in other fields is covered in the main by ample margin of operating profit. Without ample margin the risk is serious and is well illustrated by the history of the iron-ore mines in New Jersey. In 1880, these mines were thriving and, according to Smock, the value of the product was $4.16 a ton, the average cost was $3.38, and the average profit $0.78 per ton. Probably no one foresaw then that the discovery of deposits in Michigan and, later, in Minnesota would supplant the New Jersey ores for a good many years, but the danger existed and was imminent because of the narrow profit margin. Only the most favorably situated mines survived, and the yield of iron ores for the state of New Jersey, which reached its peak in 1882, shrunk by two-thirds, where it remained stagnant for more than thirty years.

An examination of Smock's valuation of the New Jersey mines, taken in the light of the later history of the industry, emphasizes the chances of error from depending on averages. The average price of product is right enough, but the average cost of operation is different for each separate mine. The average margin of profit referred to average price was 18.7 per cent. But some of the mines had a profit margin exceeding 50 per cent. while others were squeezed down to 2 per cent. It is plain to see which mines succumbed first to the competition of the ores from the Lake region and obviously the risk in case of the narrow-margin mines was greater than the risk in the mines with ample operating margin. Thus, if we assume Smock was right in selecting a 10-per cent. discount rate applied to the average condition, it is reasonable to deduce that 6 per cent. might have been a fair rate for the mines with ample margin, whereas 10 per cent. must have been too little for the mines operating near the line.

Life of Mine.—Risk bears a relation to that part of the ore estimated over and above the proved exposure and risk varies according to the character of the evidence on which the estimates are based. If no ore is assumed to exist beyond what is actually measured and sampled, there is obviously no risk as far as quantity is concerned. There are some cases where this condition exists, where the life of the mine is definitely limited, as it is sometimes by property boundaries or by known structural features or geological conditions. But in the usual case, extension of the ore will be assumed in accordance with estimates based on the information available in each individual mine. But the degree of the hazard is dependent on the character of the information, on the type of the deposit and on the general development of the district. Thus, an estimate of a short life for one mine may involve as much risk as the assignment of a long life to another mine. This hazard must find expression in the discount rate. In this connection it is pertinent to observe that risk of life may be greater with a mine operated as a single unit than it is with a larger property operating in multiple units. I am thinking of mines like those in Butte, where perhaps ten or twelve separate mines are worked by one company and comparing it with an individual mine working only a single vein, perhaps through a single shaft.

BIOGRAPHICAL NOTICES

Robert Woolston Hunt

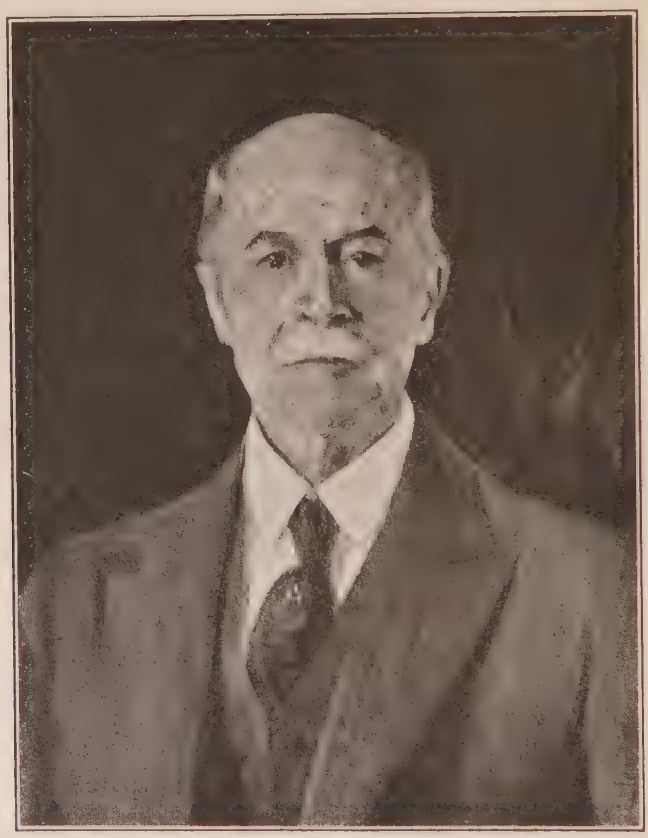
Robert Woolston Hunt, who joined the Institute in 1874, served twice as its President, was made Honorary Member in 1919, in whose honor the Hunt prize and medal were established, and who, within a month, had received the Washington Award, died at his home in Chicago on July 11, 1923.

Robert Woolston Hunt was born Dec. 9, 1838, in Fallsington, Buck County, Pa. His father, Dr. Robert A. Hunt, of Trenton, N. J., graduated from Princeton College in the class of 1824, and later from the University of Pennsylvania. His mother was Martha Lancaster Woolston. Mr. Hunt spent several years learning the practical side of iron making in the rolling mills of John Burnish & Co. at Pottsville, Pa. Later he took a course in analytical chemistry in the laboratory of Booth, Garrett & Blair, upon the completion of which he entered the employ of the Cambria Iron Co. at Johnstown and, for them, on Aug. 1, 1860, established the first laboratory in America forming an integral part of an iron or steel organization.

In the fall of 1861, he entered the United States military service and was in command of Camp Curtin at Harrisburg, serving as mustering officer for the State of Pennsylvania, with the rank of captain. In 1864, he assisted in recruiting Lamberts' Independent Mounted Company and was mustered into active service as a sergeant, having tossed up with a friend who had also participated in recruiting the company, as to which one should receive a lieutenant's commission. Upon being mustered out of service, Mr. Hunt returned to the employ of the Cambria Iron Co. which sent him to the works at Wyandotte, Mich., where experiments with the new bessemer process of making steel were being conducted. He was placed in charge of those works in July, 1865, and so continued until May, 1866, when the Cambria company called him back to Johnstown to take charge of its steel business, intending at once to begin the erection of a bessemer steel plant. The plant, however, was not built for several years, and in the meantime the Cambria company began the rolling of steel rails for the Pennsylvania Railroad, using ingots of bessemer steel produced by the Pennsylvania Steel Co. at its Steelton works. Mr. Hunt had charge of the steel for these operations; and in

August, 1867, the first steel rails made in America on a commercial order were rolled.

Later Mr. Hunt assisted George Fritz, chief engineer of the Cambria company, in designing and building the bessemer works for the Cambria plant, and on its completion, July 10, 1871, he assumed charge, so continuing until August, 1873, when he resigned. On Sept. 1, 1873, he moved to Troy, N. Y., becoming superintendent of the bessemer works of John A. Griswold & Co., and, in March, 1875, became general superintendent of the Albany & Rensselaer Iron & Steel Co., which had acquired the works of John A. Griswold & Co., and of Erastus Corning & Co. This organization later became the Troy Steel & Iron Co., and Mr. Hunt remained in charge of it until April, 1888.



Robert W. Hunt

On leaving the Troy works, Mr. Hunt established the Bureau of Inspection Tests and Consultation, now known as the Robert W. Hunt

Co., and moved to Chicago where the principal offices have always been located.

During the years of Mr. Hunt's active connection with steel rail manufacture at Troy, he almost completely rebuilt the various works and also erected a large blast-furnace plant of the most complete character. Mr. Hunt individually, and in conjunction with John E. Fry, Wm. R. Jones, Dr. August Wendel, and Max M. Suppes, patented various details of iron and steel metallurgical processes and machinery, among the most important of which were the first automatic rail mill tables, which immediately became very popular and were used under licenses by the majority of the rail mills in the United States.

Mr. Hunt served three terms as commander of the John A. Griswold Post No. 338, G.A.R., of Troy. He was also a member of numerous technical societies, among which may be mentioned the American Institute of Mining and Metallurgical Engineers (honorary member and twice president); American Society of Mechanical Engineers (honorary member and past president); American Society of Civil Engineers; Western Society of Engineers (honorary member and past president); Canadian Society of Civil Engineers; Institute of Civil Engineers; the Institution of Mechanical Engineers; the Iron and Steel Institute of England; American Iron and Steel Institute; and the American Society of Testing Materials (honorary member and past president).

In 1912, he was awarded the John Fritz Medal "for his contribution to the early development of the bessemer process." Almost his last public appearance was at the annual meeting of the Western Society of Engineers on June 18, 1923, when he was presented with the Washington Award "for his pioneer work in the development of the steel industry in the United States and for a life devoted to the advancement of the engineering profession."

Mr. Hunt contributed many papers to the proceedings of the various societies of which he was a member. Among the most important of these may be mentioned the "History of Bessemer Manufacture in America" which was presented at the Centennial meeting of the American Institute Mining Engineers on June 23, 1876; and "The Evolution of the American Rolling Mills" presented, as his presidential address, to the American Society of Mechanical Engineers on Nov. 16, 1891. These two papers record with great accuracy many interesting facts concerning the development of the bessemer steel process, and of rolling mills, in the United States, and they will unquestionably stand as Mr. Hunt's most important contributions. Mr. Hunt especially will be remembered for his personal activities in the development of steel rails for modern use. He was secretary of the committee of the American Society of Civil Engineers which designed the rail sections bearing that name, and he later contributed largely to the re-design of various sections; the one now known as type "A" section of the American Railway Association is virtually his work. His contact with rail manufacture was never lost, for necessarily his work in connection with the inspection of rails required the maintenance of intimate knowledge of the various details of manufacture. In 1912, at the instigation of one of the most important of American trunk lines, his organization inaugurated the plan of rail inspection since generally known as "Special Inspection," which contemplates the most thorough supervision of the manufacture of steel and its rolling into rails that has been attempted, the success of which has been unquestioned. As illustrative of Mr. Hunt's intimate touch with the steel rail problem, it may be mentioned that under date of Aug. 1, 1921, he issued a public letter proposing a new rail specification, advocating again the nick-and-break test for soundness on each ingot rolled, or the alternative of making the top metal of the ingots into tie plates.

In 1916, Mr. Hunt received the honorary degree of Doctor of Engineering from the Rensselaer Polytechnic Institute, of which he was a trustee for many years.

On Dec. 5, 1866, he was married to Miss Eleanor Clark, of Ecourse, Mich., who survives him.

William R. Walker

William R. Walker, assistant to the president of the U. S. Steel Corpn., died at St. Luke's Hospital, New York, on Dec. 20, 1922. He was born at LaPort, Ind., Nov. 26, 1857, and his whole career has been closely identified with the iron and steel industry. His first employment was with the North Chicago Rolling Mill Co., Chicago; then he was appointed chemist for the Crown Point Iron Co., at Crown Point, N. Y.; and later he was works superintendent for Spang, Chalfant & Co., of Pittsburgh. He was early associated with Robert Forsyth, the well-known engineer who constructed the South Chicago works of the North Chicago Rolling Mill Company.

Prior to 1890, Mr. Walker was works manager for the Union Steel Co., and upon organization of the Illinois Steel Co. he became general manager of its South Chicago works. At the time of the formation of the U. S. Steel Corpn., in 1901, he was appointed assistant to the president, which position he held until the time of his death.

Through the technical staffs and committees of the Steel Corporation, Mr. Walker had an excellent opportunity to study and direct the development of mechanical methods for coal mining, coal washing, beehive coking, including the collection and recovery of its byproducts, and to interest himself in byproduct coke ovens, and the design and operation of processes for the coking of special coal mixtures. He also brought his experience to bear on the design, construction and operation of the blast furnace and its appurtenances, as well as the bessemer, open-hearth and electric steel processes. Incidentally, he was interested in microscopy and metallography, and was familiar with the manufacture of products related to the iron and steel industry, such as tin, zinc, sulfuric acid and many other substances. Mr. Walker had the remarkable faculty of being able to discuss and conduct investigations in practically any technical branch of the ferrous industry.

In the words of President James A. Farrell, "Mr. Walker was a man of keen mental attainments and possessed an exceptional knowledge of the iron and steel industry, from both the scientific and the practical standpoint." He was, however, characterized by remarkable modesty and self-effacement.

Mr. Walker was a member of many of the technical associations of the country and took a leading part in their activities. He became a member of the American Institute of Mining Engineers in 1881, and for many years, up to the time of his death, rendered valuable assistance as a member of the Iron and Steel Committee. He was an active member also of the American Iron and Steel Institute, the Iron and Steel Institute of London, the Franklin Institute, and of a number of prominent social organizations.

Harry Harkness Stoek

HARRY HARKNESS STOEK, whose sudden death on March 1, 1923, was a great shock to his friends in all parts of the country, was a man of remarkable personal characteristics and mental ability. Through an acquaintance with him of nearly 35 years, during 12 years of which I was intimately associated with him both in business and socially, I learned to appreciate his many high qualities, his strict adherence to high professional and social ideals, and his lovable traits of character. In this appreciation I was joined by every one who was thrown in close personal contact with him.

He was a conscientious student, not given to hasty judgments, but when he formed an opinion he was positive in upholding it. At the same time, he was most courteous and kindly in combatting that which he believed was erroneous. In fact, I never knew a person of his attainments more modest, forceful and kindly, or one who had a keener sense of clean humor. In his private life, he was a devoted husband and father, a consistent communicant and worker in the Episcopal Church, and was charitable in all things. In fact, his whole life was marked by helpfulness to others and a determined and successful effort to "make the world better by his having lived in it."

Doctor Stoek was born in Washington, D. C., Jan. 16, 1866. His early education was obtained in the public schools of that city and he graduated from the Central High School in the first class to graduate from a high school in Washington.

He entered Lehigh University in 1883, graduated in 1887 with the degree of Bachelor of Science, and after a year of post-graduate study in mining and metallurgy he received the degree of Engineer of Mines.

During the summer vacation in 1885, he served as an assistant in the Department of Mineralogy in the National Museum at Washington, and during the summer of 1887 acted as U. S. Government inspector of dredging on the Thames River, Connecticut.

From June, 1888, to January, 1900, he was assistant engineer for the Susquehanna Coal Co., at Wilkes-Barre, Pa. From January, 1890, to June, 1893, he was Instructor in Mining, Metallurgy and Geology at Lehigh University, and had charge of all courses in mining and metallurgical design, and the visits of students to mines and metallurgical plants for inspection purposes. These visits made him familiar with the iron and zinc industries of eastern Pennsylvania and New Jersey, and with all types of mining employed in the anthracite regions of Pennsylvania.

From September, 1893, to January, 1898, he was Assistant Professor of Mining and Metallurgy at Pennsylvania State College, and while there he had unusual opportunities for becoming familiar with the methods employed in the coal and iron industries of central and western Pennsylvania.

Harry H. Stoek.

From January, 1898, to September, 1909, he was managing editor of *Mines and Minerals*, at Scranton, Pa. In his editorial capacity, Doctor Stoek visited most of the mining regions of the United States gathering material for articles by means of careful personal investigations. In this way, he secured an intimate knowledge of practically all the coal mining districts in the United States, except those on the Pacific slope.

During this period he revised and edited many of the instruction papers on mining used by the International Correspondence Schools; also the "Coal and Metal Miners' Pocketbook," Fulton's "Coke," and Lake's "Prospecting for Gold and Silver;" he also wrote for the United States Geological Survey a chapter on anthracite, included in the 22d Annual Report. He also prepared an "Economic History of Anthracite," as part of an "Economic History of the United States" for the Carnegie Institution, of Washington, which has not yet been issued.

In 1909, Doctor Stoek accepted the position of Professor of Mining Engineering and head of that department at the University of Illinois. From 1909 until the State Department of Mines was reorganized, so as to do away with the commission form of administration, he served as Secretary of the Illinois Mine Rescue Commission and as a member of the Mining Investigation Commission of Illinois.

He was given the honorary degree of Doctor of Science by the University of Pittsburgh in June, 1920, and his *alma mater*, Lehigh University, conferred the same degree on him on Oct. 14, 1922.

The day before his sudden death he was appointed, by the American Engineering Council, a member of a committee to study and report on practical and safe methods for the storage of coal.

Doctor Stoek became a member of the American Institute of Mining Engineers in 1888, and at the time of his death was a member of the Committee on Coal and Coke. He was a past-president of the Coal Mining Institute of America, and an Honorary Member of that organization. He was also a past-president of the Illinois Mining Institute. He was a member of the International Railway Fuel Association, the American Association for the Advancement of Science, the Society for the Promotion of Engineering Education, the American Mining Congress, and of the following collegiate societies: Delta Phi, Sigma Xi, Tau Beta Pi, and Delta Mu Epsilon.

Doctor Stoek was also a consulting engineer for the U. S. Bureau of Mines, and during the summer of 1922 he prepared *Technical Paper* No. 326, on "Fires in Bunker and Cargo Coal;" this paper is now in press. During the last Christmas holidays he went to Washington to aid in the preparation of the preliminary report on coal storage for the Fact-finding Coal Commission.

In addition, between the years 1916 and 1923, he was the author of over a score of important technical bulletins, papers, and articles for various technical journals, all of which were valuable contributions to mining literature.

On Dec. 20, 1894, Doctor Stoek was married to Miss Miriam Ricketts, of Wilkes-Barre, Pa., who died in 1919. Of his two children, a daughter, Miss Leigh Stoek, survives him; a son Lehr, an exceptionally bright and attractive youth, died while attending high school. The death of his son, followed in a comparatively short time by that of Mrs. Stoek, unquestionably brought such sadness into his busy life as to affect materially his general health. But true to his nature, he mourned privately, and did his utmost to make life cheerful for his daughter, to whom he was devoted, and all others with whom he came in contact. In a recent letter, Miss Pratt, secretary of the Department of Mining, at the University of Illinois, sums up his character in the following sentence: "He was a man with such a kindly manner and beautiful personality that every one grieves."

RUFUS J. FOSTER.

To me, Professor Stoek's outstanding work was in helping to remove mining engineering from a rule of thumb trade to an engineering science. I was with him from 1912 until 1917. During this period, I saw him handle a great number of researches, reports and bulletins. Many were published under the names of other authors and yet everywhere in them Professor Stoek gave a helping suggestion, devised ways and means of attacking the problem, and showed the author how to analyze and assemble his data.

As a teacher, he inspired his students with the dignity and vision of engineering. They loved and respected him. I once heard a student say, "No student would ever try to crib in Professor Stoek's classes because they respected him too much."

As an engineer, he found time during the past ten years to do considerable consulting and private work, especially in the appraisal and valuation of coal-mining properties and on devising ways and means of storing coal. It was my privilege to see some of these reports; they showed a breadth of engineering knowledge and method of arrangement and presentation unsurpassed in any engineering reports I have read.

As an editor, my recollection is that of a master with unfailing patience helping me, a novice, through the difficult art of properly preparing technical material for publication. And finally, as a Christian and as a gentleman, I shall remember him the clearest. If a man working for Professor Stoek qualified himself for a better position, the Professor worked the hardest to gain the promotion for the man, even though it meant the loss of his services to Professor Stoek himself and the necessity of finding and breaking in a new man for the work. At Illinois, I saw this occur not once, but at least a half a dozen times. Professor Stoek thought more of the advancement of his men than of his own time and trouble. At Urbana, he took the same interest in his church work as he did in his professional work. I often wonder how he was able to do it; I believe that it was the same combination of qualities that made Professor Stoek successful and well loved.

E. A. HOLBROOK.

Charles Mather MacNeill

Charles Mather MacNeill, President of the Utah Copper Co. and of the Chino Copper Co., and a life member of the Institute since 1899, died at his home in New York on March 17, after a very brief illness with pneumonia.

Mr. MacNeill was born at Oak Park, Ill., in 1871, and began his connection with mining enterprises immediately after graduating from high school, his first position being that of cashier with the Pueblo Smelting and Refining Co. He later went to Aspen, Colo., as manager of the Holden lixiviation plant. In 1893, he went to Cripple Creek as manager of the Lawrence Reduction Works, in which Mr. Holden and the late Capt. J. R. DeLamar were large owners. It was at this plant that Mr. Jackling became associated with him. The plant was destroyed by fire and, early in 1894, Mr. MacNeill became associated with Messrs. Spencer Penrose and Charles L. Tutt in a plan to finance and erect a chlorination plant at Colorado City, Colo., for the treatment of Cripple Creek ores. This plant was erected and was known as the Colorado-Philadelphia Reduction Works. Mr. MacNeill was general manager of the plant, Mr. Tutt having been president and Mr. Penrose, secretary and treasurer. The plant was very successful and was expanded, finally being reorganized under the name of the United States Reduction & Refining Company.

Late in 1903, or very early in 1904, D. C. Jackling called the attention of Messrs. MacNeill and Penrose to the copper properties in Bingham Cañon, Utah, which have since become the main properties of the Utah Copper Co. After extended examinations and milling tests, these properties were bought, Mr. MacNeill becoming president of the Utah Copper Co., Mr. Penrose, secretary-treasurer, and Mr. Jackling, general manager. Mr. MacNeill remained president of the company to the time of his death.

Mr. MacNeill was also a director, a member of the executive committee, and was largely interested in the Ray Consolidated Copper Co.; he was also president and a member of the executive committee of the Chino Copper Co. He had a variety of other mining interests, but the ones mentioned were distinctly the most important. He was a director of national banks in Denver and Colorado Springs, and for many years was a trustee of Colorado College.

Mr. MacNeill was married in Paris, in 1910, to Marion Parsons Irwin, of New York, who survives him.

As an executive and as an all-round business man, Mr. MacNeill possessed great ability and a great amount of resource and determination. His death is a great loss to us all.

SHERWOOD ALDRICH.

Robert Carl Sticht

Robert Carl Sticht, member of the Institute since 1886, an American metallurgist of world-wide reputation, died in St. Margaret's Hospital, Launceston, Tasmania, on April 30, 1922, after an illness of eleven months, against which he fought with the highest courage and characteristic fortitude.

The news of his passing came as a great shock to a host of his friends and former associates here, but on account of his long absence from the United States, exact biographical details have been heretofore lacking. Throughout Tasmania, the expressions of public sorrow were many and impressive, and the flags were flown at half-mast in Queenstown, Zeehan, and the entire Mt. Lyell district, in honor of the man who had won his place in the community as a great captain of industry and a man universally admired and loved.

Sticht had been connected with the Mt. Lyell Mining & Railway Co. in Tasmania throughout its entire productive period. He had originally been induced to go to Tasmania through the good offices of the late Dr. E. D. Peters, then professor of metallurgy at Harvard University, who after personally examining the undeveloped Mt. Lyell properties, recommended Sticht as the best possible man to work out the complex metallurgical problem which that great orebody presented. Sticht had already made himself the leading authority on pyritic smelting, having taken up the idea as suggested and partly worked out by Lawrence Austin and carried it through to a certain degree of perfection in the case of limited tonnages of special ores. After studying the situation at Mt. Lyell, Sticht found the conditions there to be almost ideal for pyritic smelting, and he had the courage to adopt as the basis for a permanent, and eventually successful, solution of the metallurgical problem, a process that was then little more than a theoretically interesting possibility. The thorough studies and remarkable results attained at Mt. Lyell were published as chapters in Doctor Peters' various publications on the metallurgy of copper, Doctor Peters giving full credit to Sticht for their authorship. These chapters by Sticht practically constitute the best literature of the art of pyritic smelting. The accuracy of observation, the logical reasoning, the brilliant results and the modesty with which these results are presented as mere chapters in another's work, are all characteristic of the man.

Sticht was born in Hoboken, N. J., Oct. 8, 1856, a son of the late John C. Sticht of Brooklyn, N. Y. He graduated at the Brooklyn Polytechnic Institute in the class of 1875 with the degree of Bachelor of Science, and went at once to the Mining Academy at Clausthal, Germany, where he graduated in 1880 with the degree of Mining Engineer. While at Clausthal, he became identified with a brilliant group of young men, many of them Americans, who have left their mark on American mining and metallurgical practice, among them being the late Harry L. Bridgman, the two brothers, James W. Neill and William L. Neill, Carl Koelle and A. L. Dean.

On returning to this country after graduation, Sticht went to Leadville, Colo., in 1880, just on the eve of the great mining and smelting boom. Here he secured a position as assayer and chemist at the Billing and Eilers smelter, one of the first modern smelters to be started in that great camp. The business connection thus formed with the late Anton Eilers continued without interruption for nearly 15 years, through three successive smelter enterprises. The Billing and Eilers smelter at Leadville was very successful, both technically and financially, and about the year 1883, the partners sold out to the late August R. Meyer and associates, afterwards known as the Consolidated Kansas City Smelting & Refining Co. The name of the plant was changed to the Arkansas Valley Smelter, the name it retains to the present day, under the ownership of the American Smelting & Refining Co. It is an interesting fact that this plant stands today as the only one now in operation of the many lead smelters that formerly existed in Colorado on the eastern slope of the Rocky Mountains, and though the main body of the plant has been many times modernized, the little old log-cabin buildings which Mr. Eilers constructed in 1879 for office and manager's residence are still serving their purpose as administrative headquarters. When the Billing and Eilers partnership dissolved, Mr. Eilers secured control of the famous Madonna mine at Monarch, Colo., and established his smelting plant at Pueblo, where it was known as the Colorado Smelting Co. It was built about 1883, and Sticht was the designing engineer. The plant stood for many years as a model of excellent arrangement and up-to-date practice, and there, under the guidance of Anton Eilers, many young men were trained who have since taken important places in the industry.

In 1888, Anton Eilers embarked on the organization of a new smelting enterprise in Montana, known as the Montana Smelting Co., with works at Great Falls. Sticht was selected to build and operate the new plant, and remained there until about 1892, when he assumed charge of the pyritic smelting enterprise which had been started by Lawrence Austin at Toston, Mont. The Toston enterprise was not entirely satisfactory, and was succeeded by another pyritic smelter in another locality. This also was of passing interest, for it was beginning to be demonstrated from practical experience that pyritic smelting, which showed such attractive possibilities on paper, was full of pitfalls and disappointments, and many details remained to be worked out before it could claim the confidence and stability that other smelting processes had attained.

In 1894, Sticht received the offer, above referred to, to take charge of the development of the Mt. Lyell property in Tasmania, and in January, 1895, on the eve of embarking overseas, he was married in Pueblo, Colo., to Miss Marion Oak Stage, daughter of the late N. D. Stage of Illinois, and took steamer from San Francisco for Tasmania on his wedding journey.

On his arrival in Tasmania, he seems to have won promptly the respect and confidence of his Directors, who backed his judgment to the limit and gave him a free hand in executing his plans, even to the extent of adopting the partly worked out procedure of pyritic smelting. For two years he served as chief metallurgists, while designing the reduction plant. In 1897, he was appointed the first general manager of the company, which position he held until his death.

The signal success which the enterprise quickly achieved justified to the fullest extent the confidence and the enlightened policy of the Directors. Mt. Lyell became one of the great copper producers of the world, and the complete success of Sticht's pyritic smelting program, supplemented by his mature technical experience and wise business management, placed him on the highest rung of metallurgical fame, while his personal traits of mind and heart won for him, in a remarkable degree, the affection and enthusiastic loyalty of his men and the esteem of the entire population of that region. This was evidenced in a remarkable manner by public demonstrations in his honor when, with his wife and three sons, he started for his first trip back to the United States in 1914. There were eulogistic editorials in the newspapers, public meetings in his honor, complimentary dinners with speeches, and his railroad journey to the port was marked by a series of popular ovations along the line such as few men in private life may expect to experience. It was a remarkable tribute to his rare personality.

This visit to the United States gave his friends here the first opportunity they had had in twenty years to renew acquaintance and hear his own account of his interesting metallurgical achievements, credit for which he was always ready to apportion to others. He returned to this country again in 1917 for a brief visit, to study some problems of ore treatment.

Though closely identified for so many years with affairs in Tasmania, he retained his American citizenship, of which he was very proud.

Sticht was a lover of the beautiful things of life and possessed artistic talents of a very high order. He was a versatile musician, but those who knew him in the old days in Colorado remember best his exquisite playing of the Tyrolean zither, which he learned abroad. A talented draftsman, he had a fine taste in paintings, and succeeded in bringing together an excellent collection of good paintings and rare editions, in spite of the remoteness of his location.

The Sticht home in Queenstown was noted for its atmosphere of quiet happiness and refinement, and that it was the center of fine influence in the community is evidenced by the appreciative references to it that occur in the many tributes to its owner's life and work that appeared in the press of Tasmania after his death.

Sticht was member of the American Institute of Mining and Metallurgical Engineers, member of the American Electrochemical Society, past-president of the Australian Institute of Mining and Metallurgy, life-member of the Institution of Mining and Metallurgy of London, member of the Australian National Research Council, Fellow of the Royal Geographical Society, Fellow of the Geological Society and the recipient of numerous honorary distinctions from scientific bodies.

The obituary notices in the local papers describe him as "Tasmania's foremost citizen," and on the occasion of his funeral, there was a five minutes' cessation of all work throughout western Tasmania. A public memorial service was held in Metropole Hall, Queenstown, on May 14, 1922. He is survived by his wife and three sons, Robert, Jr., Hadmar, and Chester.

We of his own people and his old associates would have joined reverently in those demonstrations, could we have done so, but we rejoice that for so many years he found a congenial home in far-away Tasmania, and that he was able to pass on to so many others the benediction of his life and character which still lives in the hearts and memories of his old friends in America.

ARTHUR S. DWIGHT.

George Edward Webber

George Edward Webber, a member of the Institute since 1906, died in San Francisco, May 29, 1922. Born in Dixmont, Me., in 1852, he went to California in 1857 where he lived until 1883 and received a thorough training in precious metal mining in the employ of his father and J. B. Haggin. In 1883, he went to the Black Hills, South Dakota, as superintendent of the Father DeSmit mine; later the Deadwood Terra also came under his management. These mines are now part of the Homestake Mining Co.

In 1889, the Rothchilds asked Hamilton Smith to recommend an engineer to take the place of the late Henning Jennings, who had resigned from the El Calloo Mining Co., of El Calloo, Venezuela, in order that he might proceed to South Africa. Mr. Webber accepted the position, remaining there until 1893 when he returned to the United States on his way to Johannesburg, South Africa, where he had been appointed manager of the Crown Reef Mining Co., Ltd.

Early in 1896, he became general manager of the Rand Mines, Ltd., and served in this capacity until 1911. During the latter part of this period, Mr. Webber was active in the greater amalgamations of H. Eckstein & Co., and it was largely on his advice that H. Eckstein & Co. carried out the negotiations preceding the formation of the Crown Mines, Ltd. On his retirement from South Africa, he returned to live in Ross Valley, near San Francisco, Cal.

The Rand Mines, Ltd., of which Mr. Webber was general manager, was the largest company controlled by Wernher Beit, the leading London house interested in South Africa mining, whose local agents in Johannesburg were H. Eckstein & Co. This firm is now known as the Central Mining & Investment Corpn., and after the departure of Henning Jennings, Mr. Webber was the firm's most valued and confidential advisor and was very highly thought of in mining circles in South Africa. He was a ma of most upright and straightforward high character, plain spoken and sincere; and his recognized high character was one more qualification which aided his great success.

Ruel C. Warriner.

Arthur Yates

Arthur Yates, lecturer in the mining department of the Royal School of Mines, London, died at Blackpool, on Feb. 14, 1923, at the age of 47 years.

My first acquaintance with him was made in 1902, when taking over the management of the Redjang Lebong gold mines in Sumatra, of which he was the responsible metallurgical officer. For that position he was well qualified by reason of his previous experience with the Rand Central Ore Reduction Co., in South Africa, and particularly by his early training with the Cassel Gold Extracting Co., owners of the MacArthur-Forrest patents, and the first commercial company formed to work the then recently discovered cyanide process; in this connection, Mr. Yates had the advantage of working under J. S. MacArthur himself.

Our association in Sumatra was maintained for six years, when I left. Mr. Yates remained and was eventually promoted to the management of the neighboring Soelit mine. During those six years and in the close association inseparable from life in an isolated mining camp, the sterling qualities which made him so good a man were unmistakable: industry and interest in his work, patience with everybody, a quiet and unassuming demeanor. On the exhaustion of the mine to which he had been promoted, he returned to England in 1913.

In 1914, the Mining Department of the Imperial College was fortunate in securing his services. Hardly, however, had he settled down when the war broke out and he spent some years in the Cleveland District promoting the efforts of the Government toward greater output.

Returning to college in 1919, Mr. Yates met the full flood of Service students. The qualities which made him a man amongst men in the world made him a man among students in the college; a senior student if you like. But, unsuspected, a dread disease had fastened itself upon him, making itself felt for the first time during the summer course he was conducting in the South Wales coal field, in 1922. Against its relentless onset he opposed the bravest optimism, and when at last he realized that he was going under, and was in fact dying, his pronounced regret was that he was leaving the boys.

In his death, at the age of forty-seven, staff and students in the Mining Department have experienced a great loss, that of a reliable friend, and their deepest sympathy goes out to the young widow and son in their irreparable loss.

S. J. Truscott.

Ellsworth Daggett

Ellsworth Daggett, who joined the Institute in 1873, and had been a prominent figure in the mining profession of Utah and other Western states for many years, died in San Francisco, Jan. 5, 1923. Mr. Daggett was a contemporary and intimate friend of Clarence King, James D. Hague, R. W. Raymond, and other Nestors of the mining engineering profession prominent in the last quarter of the nineteenth century.

He was born at Canandaigua, N. Y., May 24, 1845; graduated from Sheffield Scientific School in 1864; pursued post-graduate courses there for two years, and for one year at the Bergakademie, in Berlin. He was in active practice as mining engineer from 1866 to 1914, but devoted most of his time during the World War to the perfecting of a submarine torpedo.

He was connected with the United States Geological Exploration of the Fortieth Parallel in 1870. The plates illustrating mining industry, Volume III of the reports of that survey, were engraved from drawings obtained in the field during progress of the survey and prepared for the engraver by Mr. Daggett. He operated the Winnamuck mine and smelter at Bingham Cañon, Utah, in the early seventies; the mines and mill at Cusihuiriachic, Chihuahua, 1886–7; and the Berlin mine near Austin, Nevada, 1904–6.

On June 24, 1874, Mr. Daggett married Miss June Spencer, of Salt Lake City, who survives him, residing in New Haven, Conn.

Appointed United States Surveyor General for Utah by President Harrison, in 1888, he was the first engineer to hold that office in Utah. He reorganized the mineral division of the office, required the courses of mineral surveys to be referred to the true meridian determined by solar or stellar observations, and did much to improve the accuracy and reliability of such surveys.

Mr. Daggett was a member of the Alta and University Clubs of Salt Lake City, and president of the latter from 1890 to 1892; he was also a member of the Century Club, New York. Through his instrumentality, irrigation was brought to considerable tracts of land in northeastern Utah, and Daggett County of that state is named for him.

He wrote numerous technical papers, and an economic review entitled "A Quarter of a Century of Prices," in which was shown the close relation of the prices of commodities to the price of silver from 1872 to 1896; this was used in the presidential campaign of 1896.

Loyal to his friends, thoroughly independent, and indifferent to criticism, his private and professional life exemplified what is best and most worthy of emulation in the career of a gentleman and a mining engineer.

T. S. MATHIS.

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